NI 43-101 TECHNICAL REPORT AND FEASIBILITY STUDY

Lake Giles Iron Project
Menzies, Western Australia

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Important Notice

This Technical Report, following National Instrument 43-101 rules and guidelines, was prepared for Macarthur Minerals Ltd by Engenium Pty Ltd now Stantec Australia Pty Ltd (Engenium), Orelogy Consulting (Orelogy) and CSA Global Pty Ltd (CSA Global). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in Engenium, Orelogy and CSA Global's services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this Report. This Report can be filed as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under Canadian securities laws, any other uses of this Report by any other third party are at the party's sole risk.

Date and Signature Page

This report is effective as of 4 April 2022.

Signed by "Stephen Craig"

Stephen Craig
B. Eng. in Mining Engineering, FAusIMM

Signed by "David Williams"

David Williams
B. Sci. (Geo), RPGeo (MAIG)

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Certificate of Qualified Person Stephen Craig

This certificate applies to the NI 43-101 Technical Report entitled “Feasibility Study, Lake Giles Iron Project, Menzies, Western Australia” (the Technical Report), prepared for Macarthur Minerals Ltd (Macarthur) issued on 4 April 2022 and effective as of 4 April 2022.

I, Stephen Craig do hereby certify that:

1. I am the CEO for Orelogy Consulting Pty Ltd located at Unit 19, 162 Colin St West Perth WA 6005.
2. I am a professional mining engineer having graduated with a B Eng in Mining Engineering from South Australian Institute of Technology (1987).
3. I am a Fellow of the Australian Institute of Mining & Metallurgy (Member 112346).
4. I have practised my profession as a mining engineer for the past 35 years in the mineral resources sector and engaged in the assessment, development, and operation of mining projects both within Australia and internationally.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6. I have authored and take responsibility for Items 1.7, 1.8, 1.16.2, 1.17.2, 15, 16, 21.21, 25.3 and 26.2 of the Technical Report.
7. I have conducted a site inspection on the 1st/2nd November 2021.
8. I am independent of the Issuer as described in Section 1.5 of NI 43-101.
9. I have had not had a prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.
11. As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of April 2022 at Perth, Western Australia, Australia.

[“SIGNED”]

{Stephen Craig}

Stephen Craig, B. Eng in Mining Engineering FAusIMM
CEO, Orelogy Consulting Pty Ltd

Certificate of Qualified Person David Williams

This certificate applies to the NI 43-101 Technical Report entitled "Feasibility Study, Lake Giles Iron Project, Menzies, Western Australia" (the Technical Report), prepared for Macarthur Minerals Ltd (Macarthur) issued on 4 April 2022 and effective as of 4 April 2022.

I, David Williams do hereby certify that:

1. I am a Principal Resource Geologist with CSA Global Pty at its Queensland office located at Level2, 201 Leichhardt Street, Spring Hill, Queensland, Australia.
2. I am a professional geologist having graduated with a B.Sc. (Hons) in Geology from the University of Adelaide (1990).
3. I am a Member of the Australian Institute of Geoscientists (member 4176).
4. I have practised my profession as a geologist for the past 30 years in the mineral resources sector and engaged in the assessment, development, and operation of mineral projects both within Australia and internationally.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I have authored and take responsibility for Items 1.6, 1.16.1, 1.17.1, 14, 25.2 and 26.1 of the Technical Report.
7. I have not conducted a recent and current site inspection.
8. I am independent of the Issuer as described in Section 1.5 of NI 43-101.
9. I have had prior involvement with the property that is the subject of the Technical Report. I visited the Project on several occasions between 2010 and 2012 and participated in the planning of drillholes in 2019.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.
11. As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of April 2022 at Brisbane, Queensland, Australia.

["SIGNED"]

{David Williams}

David Williams, B. Sci. (Geo), RPGeo (MAIG)
Principal Resource Geologist, CSA Global Pty Ltd

Certificate of Qualified Person Nikolay Karakashov

This certificate applies to the NI 43-101 Technical Report entitled "Feasibility Study, Lake Giles Iron Project, Menzies, Western Australia" (the Technical Report), prepared for Macarthur Minerals Ltd (Macarthur) issued on 4 April 2022 and effective as of 4 April 2022.

I, Nikolay Karakashov do hereby certify that:

1. I am a Consultant Geologist from 25 Marlborough Street, Perth, Western Australia, Australia.
1. I am a professional geologist having graduated with a M.Sci. (Hons) in Geology and Geophysics from Imperial College London's Royal School of Mines (2009).
2. I am a Member of the Australian Institute of Geoscientists (member # 6237).
3. I have practised my profession as a geologist for the past 10 years in the mineral resources sector and engaged in the assessment, development, and operation of mineral projects both within Australia and internationally.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
5. I have authored and take responsibility for Items 1.5. 7, 8, 9, 10, 11, and 12 of the Technical Report.
6. I visited the Lake Giles Iron Project on 9–10 September 2020.
7. I am independent of the issuer as described in Section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report through reverse circulation and diamond exploration drilling between 2010 and 2011 at the Moonshine and Moonshine North prospects.
9. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.
10. As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of April 2022 at Perth, Western Australia, Australia

["SIGNED"]

{Nikolay Karakashov}
Nikolay Karakashov, M.Sci. (Hons), MAIG
Consultant Geologist

Certificate of Qualified Person Neville Dowson

This certificate applies to the NI 43-101 Technical Report entitled “Feasibility Study, Lake Giles Iron Project, Menzies, Western Australia” (the Technical Report), prepared for Macarthur Minerals Ltd (Macarthur) issued on 4 April 2022 and effective as of 4 April 2022.

I, Neville Dowson, BAppSci (Ext Met), MBA, FAusIMM, residing in Perth, Western, Australia do hereby certify that:

1. I am a Principal Process Engineer with Engenium/Stantec Pty Ltd at its Perth office located at Level 2, 88 William St, Perth, Western Australia, Australia.
2. I am a professional process engineer having graduated with a B. App. Sci. in Extractive Metallurgy from the Western Australian School of Mines (1977).
3. I am a Fellow of the Australian Institute of Mining and Metallurgy (Member 101049).
4. I have practised my profession as a metallurgist/process engineer for the past 45 years in the mineral resources sector and engaged in the assessment, development, and operation of mineral projects both within Australia and internationally.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6. I have authored and take responsibility for Sections 1.9, 1.15.3, 1.16.3, 1.17.3, 13, 17, 25.4, and 26.3 of the Technical Report.
7. I have conducted a recent and current site inspection.
8. I am independent of the Issuer as described in Section 1.5 of NI 43-101.
9. I have had prior involvement with the property that is the subject of the Technical Report. I visited the Project on several occasions between 2010 and 2012 and again in 2019.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.
11. As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of April 2022 at Perth, Western Australia, Australia.

[“SIGNED”]

{Neville Dowson}

Neville Dowson, B. App. Sci. (Ext Met), MBA, FAusIMM

Principal Process Engineer, Engenium Pty Ltd

Certificate of Qualified Person David Sourbutts

This certificate applies to the NI 43-101 Technical Report entitled “Feasibility Study, Lake Giles Iron Project, Menzies, Western Australia” (the Technical Report), prepared for Macarthur Minerals Ltd (Macarthur) issued on 4 April 2022 and effective as of 4 April 2022.

I, David Sourbutts do hereby certify that:

1. I am a Project Director with Engenium/Stantec Pty Ltd at its Perth office located at Level 2, 88 William St, Perth, Western Australia, Australia.
2. I am a professional civil engineer having graduated with a B. Engineering (Civil) from the University of Western Australia (1995).
3. I am a Member of the Australian Institute of Mining and Metallurgy (Member 303432), and Member of Engineers Australia (Member 981718).
4. I have practised my profession for the past 25 years in the mineral resources sector and engaged in the assessment, development, and operation of mineral projects both within Australia and internationally.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6. I have authored and take responsibility for Items 1.1 to 1.4, 1.10 to 1.15, 1.16.4, 1.16.5, 1.17.4 to 1.17.6, 2, 3, 4, 5, 6, 18, 19, 20, 21, 22, 23, 24, 25.1, 25.5, 25.6, 26.4 to 26.6 and 27 of the Technical Report.
7. I have conducted a recent and current site inspection.
8. I am independent of the Issuer as described in Section 1.5 of NI 43-101.
9. I have had prior involvement with the property that is the subject of the Technical Report. I visited the Project on several occasions between 2010 and 2012.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.
11. As of the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of April 2022 at Perth, Western Australia, Australia.

[“SIGNED”]

{David Sourbutts}

David Sourbutts, B. Eng. (Civil), B. Com., FAusIMM

Project Director, Engenium Pty Ltd

Glossary of Terms

Al_2O_3	Alumina
AIE	Analogue Initiation Explosive (Blasting system)
AH Act	Aboriginal Heritage Act 1972 (WA)
AMD	Acid Mine Drainage
ANFO	Ammonium Nitrate and Fuel Oil (bulk explosive)
API	Assessment on Proponent Information
ATV	Acoustic televiewer
AUD M	Australian Dollars (Millions)
Bcm	Banked cubic metre
BFA	Bench face angle
BIF	Banded Iron Formation
Ca Fe	Calcined iron (with water of crystallisation removed)
Cat	Caterpillar – equipment manufacturer.
CAWS	Country Areas Water Supply Act 1947 (WA)
CFR	Cost and Freight
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
Conc	Concentrate
CRM	Certified reference materials
CTR	Clearing, Topsoil removal and Rehabilitation
CY	Calendar Year
D&B	Drilling and Blasting
DAA	Department of Aboriginal Affairs
DCF	Discounted Cash Flow
DEC	Department of Conservation
DEE	Department of the Environment and Energy
DFS	Definitive Feasibility Study
DIA	Department of Indigenous Affairs
DMA	Decision Making Authorities
DMIRS	Department of Mines, Industry Regulation and Safety
dmtu	dry metric tonne unit
DOH	Department of Health
DoW	Department of Water
DRF	Declared Threatened Flora
DS	Direct Shear geotechnical test method
DSO	Potential Direct Shipping Ore
DTM	Digital Terrain Model
dtph	dry tonnes per hour
DTR	Davis Tube Recovery
EBS	Electronic Blasting System
ELH	Excavate Load and Haul, referring to mining using excavators and dump trucks
EMP	Environmental Management Plan

ENG	Engenium/Stantec
EP Act	Environmental Protection Act 1986 (WA)
EPA	Environment Protection Authority
EPBC Act	Environmental Protection and Biodiversity Conservation Act 1999
EVO	Evolution™ mine scheduling software from Maptek
Fe	Iron
FEL	Front End Loader
FEM	Finite element modelling
FOB	Free On Board
FW	Footwall
FY	Financial Year
GDA94	National co-ordinate system used in this area.
GDP	Gross Domestic Product
GIS	Geographical Information System
GPS	Global Positioning System
GSWA	Geological Survey of Western Australia
HQ3	Diamond drilling hole size
HW	Hanging wall
IDS	Inverse Distance Squared
IODEX	Platts Iron Ore Index
IRA	Inter ramp angle – measured to-to-toe over a stack of benches
IRR	Internal Rate of Return
JORC	Joint Ore Reserves Committee (Australian reporting standards for mineral projects)
km	kilometre
L&H	Loading and hauling
LEA	Limit equilibrium analysis
LGC	Large Generation Certificate
LGIP	Lake Giles Iron Project
LIDAR	Light Detecting and Ranging (survey method)
LIMS	Low Intensity Magnetic Separation
LOI-1000	Loss on Ignition at 1000^0C
LOM	Life of mine
mg/L	milligrams per litre
MCAF	Mining cost adjustment factor (used in pit optimisation)
MIO	Macarthur Iron Ore Pty Ltd
MOC	Mining Operations Centre
MoU	Memorandum of Understanding
MMS	Macarthur Minerals Limited
Mt	Million tonnes
Mtpa	Million tonnes per annum
NES	National Environmental Significance
NI43-101	National Instrument 43-101 (Canadian reporting standards for mineral projects)

NPV	Net Present Value
OK	Ordinary Kriging
OMP	Ore mining premium (used in pit optimisation)
OTC	Optical Televiewer
OSA	Overall slope angle – measured toe-crest over a stack of benches including ramps
P	Phosphorus
P80	80% passing size (of a Particle Size Distribution)
PEA	Preliminary Economic Assessment
PEC	Priority Ecological Communities
PER	Public Environmental Review
PFS	Preliminary Feasibility Study
PoW	Program of Works
PQ3	Diamond drilling hole size
PSD	Particle size distrubution
QAQC	Quality Assurance Quality Checked
QP	Qualified Person
RAB	Rotary Air Blast (refer to drilling method)
RC	Reverse Circulation (refer to drilling method)
RF	Revenue Factor (used in pit optimisation)
RIWI Act	Rights in Water and Irrigation Act 1914 (WA)
RMU	Rock mass unit (geotechnical classification)
ROM	Run of Mine, generally referring to stockpiles ahead of crusher.
RQD	Rock Quality Designation (geotechnical classification method)
RTKGPS	Real Time Kinematic Global Positioning System
RWS	Relative Weight Strength of bulk explosive
S	Sulphur
SAG	Semi Autogenous Grinding
SG	Specific gravity
SiO_2	Silica
SMU	Standard machine unit (engine hour)
SPA	Southern Ports Authority (operator of the Port of Esperance)
SRE	Short Range Endemics
SRF	Strength reduction factor
S/R	Strip ratio
TDS	Total Dissolved Solids
TEC	Threatened Ecological Communities
TMM	Total Material Movement
TXL	Triaxial limit geotechnical test method
UCS	Unconfined Compressive Strength
UHP	Ularring Hematite Project
USDm	Millions of United States of America Dollars
USD	United states Dollars
WA	Western Australia

WC	Wildlife Conservation (1950) Act
WRD	Waste rock dump
XRF	X-ray Refraction (analytical method)

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1 Summary

1.1 Project

The Lake Giles Iron Project ("Project") is located approximately 150 km northwest of the town of Kalgoorlie in the state of Western Australia. The Project is owned by Macarthur Iron Ore Pty Ltd (MIO), a 100% owned subsidiary of Macarthur Minerals Limited ("Macarthur" or "the Company" or "the Issuer").

The Project consists of a series of banded iron formation (BIF) hematite and magnetite prospects. This report presents the mineral resources of the magnetite mineralisation of the Snark, Clark Hill North, Clark Hill South, Sandalwood and Moonshine deposits, previously reported in 2020 (CSA Global, 2020).

The scope of this feasibility study concerns the development of the Moonshine North and Moonshine magnetite deposits. Feasibility study level engineering was completed across all areas of the Project's required infrastructure in addition to investigation of existing regional infrastructure to be utilised. Associated capital and operating costs were generated to develop a financial model and define a maiden Mineral Reserve estimate.

1.2 Company Strategy

Macarthur is an Australian public company listed on the Toronto Stock Exchange (TSX-V: MMS) and the Australian Securities Exchange (ASX: MIO) and commenced exploration in 2006 for magnetite iron resources and subsequently hematite iron resources on its Lake Giles tenements in Western Australia.

In 2020, the Company reported an updated Mineral Resource estimate (CSA Global, 2020) that underpins this feasibility study focussing on development of the Moonshine and Moonshine North magnetite deposits.

1.3 Property Description and Location

The Project is located about 450 km east-northeast of the coastal city of Perth, Western Australia (Figure 1-1). Macarthur manages 15 contiguous Mining Leases covering a total area of 62 km^2. Macarthur has also made application for several Miscellaneous Licences and General-Purpose leases to support future infrastructure development. The tenements are 100% held by Macarthur Iron Ore Pty Ltd (MIO), a 100% owned subsidiary of Macarthur.

The Project comprises hematite/goethite and magnetite mineralisation located within the mining leases.

Figure 1-1: Location plan

1.3.1 Permitting and Native Title Claims

The Project does not have any environmental liabilities from previous mining or exploration activities such as the rehabilitation of waste dumps or decommissioning of tailings storage facilities. No area of the site is registered as a contaminated site that requires remediation. Macarthur has not been fined or prosecuted under any environmental legislation or received any improvement notices for current or past exploration activities from the Western Australian Department of Mines, Industry Regulation and Safety (DMIRS).

Macarthur will need to undertake an environmental impact assessment in order to obtain environmental approval for development. The Company is not aware of any major environmental obstacles that would prevent approval of the Project.

The Project sits within the Marlinyu Ghoorlie native title claim. The claim was registered on 28 March 2019 and is currently not determined. Native title rights in registration or grant give claimants the right to negotiate during the grant of mineral tenure. Macarthur's mining leases were all granted prior to registration of the Native Title claim and the current claim does not confer rights to negotiate or affect the tenure. There were no native title claims over the area at the time of grant and therefore no access agreements were required to be negotiated with claimants. Current applications for tenure as described in Section 4.2 are subject to native title. Macarthur is currently progressing heritage agreements with the native title claimants to progress the tenure to grant.

1.3.2 Infrastructure

Site infrastructure is limited to an exploration camp and graded and ungraded site tracks. No mining activities have taken place to date.

The Project is located approximately 250 km by road from the regional centre of Kalgoorlie-Boulder, a city servicing many local gold and nickel mines throughout the Eastern Goldfields region of Western Australia. The Project is also located within 90 km of an open access rail line owned by Arc Infrastructure. The rail runs for approximately 500 km directly to the Port of Esperance, that has facilities for iron mineralisation handling, storage, and export.

1.4 Property History

1.4.1 Property Ownership

Since the late 1960s, several exploration companies have held the exploration rights to the project tenements prior to Macarthur acquiring the rights to the tenements in 2005. There have been three main phases of exploration; nickel exploration from 1968 to 1972, gold exploration from 1993 to 2004 and more recently iron exploration.

1.4.2 Previous Mineral Resource Estimation and Previous Mining

There are no known historical mineral resource or reserve estimates prior to 2007 for any commodity within the area now covered by the tenements.

Mineral Resources for the Lake Giles Iron Project were reported between 2007 and 2011 as detailed in Table 1-1. The Mineral Resources were classified and reported in accordance with 2005 CIM Definition Standards for Mineral Resources and Mineral Reserves and have been superseded by the Mineral Resource estimate reported in Table 1-2 and Table 1-3.

The Issuer is not treating the previous Mineral Resource estimates as current Mineral Resources. These previous Mineral Resource estimates are presented for historical information and context only. Current Mineral Resource estimates are presented in Section 14 of this Report.

No mining is known to have been undertaken in the Project area or anywhere on the tenements to date.

Table 1-1: Previous mineral resource estimates

Deposit	Hellman & Schofield						CSA Global		Snowden	
	2007		2008		2009		2009–2010		2011	
	Mt	Head Fe %	Mt	Head Fe %	Mt	Head Fe %	Mt	Head Fe %	Mt	Head Fe %
Snark	26.3	27.5	26.3	27.5	26.3	27.5	75	27.7	-	-
Clark Hill North	7.7	32.5	37.1	26.0	37.1	26.0	130.0	25.8	-	-
Clark Hill South	48.5	21.9	48.5	21.9	48.5	21.9	66	30.3	-	-
Sandalwood	-	-	84.7	28.3	84.7	28.3	335.0	31.1	-	-
Moonshine	-	-	-	-	144.1	25.9	510.9	27.8	710.5	30.2

1.5 Project Exploration

Macarthur took over the tenements in late 2005 with the purchase of Internickel Pty Ltd. Macarthur immediately continued with the ongoing exploration program for nickel and gold. Anomalies generated by a 2004 helicopter electromagnetic survey (HoistEM) were visited and many were mapped and sampled, with emphasis on the search for nickel bearing gossans.

Iron mineralisation associated exploration activities commissioned by Macarthur at the Project area since 2005 includes geological and geomorphological mapping and geophysics, including air and ground magnetic anomaly, ground gravity, rock chip, auger and regional soil sampling, in conjunction with drilling.

Early drilling between 2006 and 2009 delineated Mineral Resources at Moonshine, Moonshine North, Snark, Clark Hill North and South, and Sandalwood, with a number of geophysical surveys including ground gravity and magnetics as well as detailed outcrop mapping occurring in the same period.

From 2010, exploration mostly concentrated on Moonshine and Moonshine North as the two prospects showed the greatest potential and highest quality of resource as exploration targets. Both prospects were drilled in three main campaigns in 2010, 2011 and 2019 with minor drilling in between. Other means of exploration during this period included further detailed outcrop mapping, a geomorphological survey covering the project area as well as regional soil sampling campaigns.

In 2011, a light detection and ranging (LiDAR) survey was conducted over the entire project area, from which a high-resolution digital terrain model (DTM) was produced, as well as composite imagery useful in environmental assessments and visual geological data.

All drilling between 2009 and 2013 included downhole surveying as well as structural data for selected holes.

1.6 Mineral Resource Definition

1.6.1 Magnetite Resource

The magnetite Mineral Resource estimates completed by Qualified Person (QP) Mr David Williams for the Moonshine and Moonshine North deposits are presented in Table 1-2. Mineral Resource estimates for the Sandalwood, Snark, Clark Hill North, and Clark Hill South deposits are presented in Table 1-3.

Mineral Resources are reported above a Davis Tube Recovery (DTR) cut-off of 15%. The Mineral Resources are not believed to be materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors.

Mineral Resources have been reported in accordance with CIM Definition Standards for Mineral Resources and Reserves dated 10 May 2014 (2014 CIM Definition Standards) (CIM, 2014).

The QP has undertaken a review of sample assays, drilling data, data validation, quality assurance/quality control (QAQC), estimation parameters, material density, block model parameters and classification procedures. The following information summarises the steps and procedures taken, and data reviewed by the QP to ensure Mineral Resource estimates are reported in accordance with 2014 CIM Definition Standards.

Table 1-2: Mineral resource estimate – Moonshine and Moonshine North, DTR >15%

Category	Tonnes (Mt)	Head grade (%)					Concentrate grade (%)					
		Fe	P	SiO_2	Al_2O_3	LOI	DTR	Fe	P	SiO_2	Al_2O_3	LOI
Measured	53.9	30.8	0.05	45.4	1.6	2.7	32.2	66.0	0.031	6.2	0.2	-0.7
Indicated	218.7	27.5	0.046	51.1	1.4	1.6	31.0	66.1	0.017	6.7	0.1	-0.1
Subtotal	**272.5**	**28.1**	**0.047**	**50.0**	**1.4**	**1.8**	**31.2**	**66.1**	**0.02**	**6.6**	**0.2**	**-0.2**
Inferred	449.1	27.1	0.047	52.6	1.0	1.4	29.2	65.0	0.026	8.4	0.1	0

Table 1-3: Mineral resource estimate – Sandalwood, Clark Hill North, Clark Hill South and Snark, DTR >15%

Deposit	Category	Tonnes (Mt)	Head grade (%)				Concentrate grade (%)					
			Fe	SiO_2	Al_2O_3	LOI	DTR	Fe	P	SiO_2	Al_2O_3	LOI
Sandalwood	Inferred	334	31.1	48.4	1.5	-0.6	33.1	64.7	0.03	9.5	0.06	-2.7
Snark	Inferred	69	27.8	49.8	1.6	2.4	23.4	66.2	0.03	7.5	0.13	-2.8
Clark Hill North	Inferred	130	25.8	42.6	1.7	0.14	33.2	62.4	0.04	12.1	0.16	-2.6
Clark Hill South	Inferred	15	32.3	47.0	0.6	0.02	31	63.8	0.02	9.8	0.14	0.0

Notes:

- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
- *The Mineral Resource estimate was prepared by David Williams, B.Sc., MAIG, a CSA Global employee, and the QP for the estimate.*
- *Mineral Resources were estimated using Datamine Studio RM (Version 1.6.87).*
- *Assays were composited to regular 1 m or 5 m intervals, dependent upon the deposit.*
- *Composite assay grades were capped as required. Fe and DTR grades were not capped.*
- *Block-model grade interpolation was undertaken using ordinary kriging.*
- *Bulk density was calculated for each block in the Moonshine model using algorithms, based upon the estimated Head Fe block grade. Average bulk density of 3.3 t/m^3 was applied to the other deposit models.*
- *Mineral Resources are reported from a model with parent block dimensions of 25 m x 25 m x 10 m.*
- *Tonnage and grade have been rounded to reflect the relative accuracy of the Mineral Resource estimate; therefore, columns may not total due to rounding.*
- *Resource classification is as defined by the CIM in its document "CIM Definition Standards for Mineral Resources and Mineral Reserves" of 10 May 2014.*
- *The QP and Macarthur are not aware of any current environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might materially affect these Mineral Resource estimates.*

1.6.2 Mineral Resource Estimation

The outcropping geology of the project area is comprised of a combination of un-altered silica-rich BIFs and altered, enriched haematite/goethite BIFs. Weathering has resulted in the leaching of the majority of the silica from the BIFs, thus producing a rock with elevated iron and decreased silica grades, near surface.

These enriched bands vary from 10 m to 150 m in true thickness and are steeply dipping at 70–90°. The outcrop of weathered iron mineralisation is indicative of the fresh (non-weathered) magnetite mineralisation located down dip which is favourable for hosting a Mineral Resource.

The main zones of mineralisation are interpreted as a series of thick tabular units, with moderate to minimal structural deformation. More intense deformation is modelled at the south edge of the Moonshine prospect with several synclinal structures and possible shearing related to recumbent folds, which increase the apparent thickness of the zones of mineralisation.

Depth and consistency of mineralisation has been confirmed to be in excess of 250 m below surface as demonstrated by results from several drillholes, confirming a consistent easterly dip of the hanging wall for the majority of the Moonshine and Moonshine North prospects.

The Lake Giles magnetite deposits were drilled with RC and diamond core drilling. The RC holes are drilled with a 140 mm diameter hammer, often on track mounted rigs due to the rugged terrain of the deposit. Diamond holes were drilled with HQ diameter core, or larger PQ diameter core if metallurgical samples were required. A total of 359 RC holes (63,733 m) and 14 diamond holes (2,809.5 m) were drilled in the Lake Giles Magnetite Project. Not all holes penetrated mineralisation. The Moonshine and Moonshine North deposits, hosting the Measured and Indicated Mineral Resources, recorded nine diamond holes (1,807.5 m) and 236 RC holes (43,156 m) in the drillhole database. There are no significant risks or uncertainties that have been identified in the exploration data or programs.

Macarthur provided the geological and mineralisation interpretations to CSA Global Pty Ltd (CSA Global), an ERM Group company, as three-dimensional (3D) wireframe solids and surfaces. The drillhole samples were flagged within the mineralisation domains, and geostatistical studies carried out for the head and concentrate assay data, including variography to ascertain the spatial variation of the various grade variables.

A block model was constructed for the Moonshine and Moonshine North deposits using Datamine software, with parent block sizes 25 m (along strike) x 25 m (across strike) x 10 m (vertical). A larger block size was used for the magnetite deposits to the north of Moonshine (Sandalwood, Clark Hill, and Snark). Head and concentrate grades, and mass recovery, were estimated into the block model using ordinary kriging.

A minimum of eight and maximum of 18 samples were used in any one block estimate, with a maximum of four samples per drillhole. Search ellipsoid radii varied between the deposits. Typically, a primary search ellipse of 240 m along strike and down plunge x 120 m down dip x 40 m across strike was used.

Block grades were validated by visually comparing block and adjacent drill sample grades, by the use of swath plots, and by comparing mean sample and block grades by mineralisation domain.

A total of 624 drill samples with bulk density measurements were captured within the mineralisation domains and statistically assessed to determine the mean and ranges, and to see if any excessively low or high bulk density values were present. Three mineralisation domains contain bulk density data. A further 400 samples were taken from the BIF oxide zones, or the footwall and hanging wall waste zones. Core samples, both from the fresh and oxidised zones, were highly competent, without any fractures or voids, and were not required to be wax sealed prior to immersion in water. A conventional Archimedes wet/dry weighing was used to measure density.

Algorithms were developed to calculate the density to apply to the Moonshine and Moonshine North block models based upon correlations between the head iron grade from assays, and the corresponding bulk density value of the sample. The density algorithms as applied to the Mineral Resources, are given here:

- Moonshine: DENSITY = (0.0241*FE) + 2.624
- Moonshine North: DENSITY = (0.0295*FE) + 2.468; and
- Moonshine (East): DENSITY = (0.0293*FE) + 2.492.

The Sandalwood, Clark Hill North, Clark Hill South, and Snark Mineral Resources were all applied a bulk density value of 3.3 t/m3, which is a typical density value for the style of mineralisation.

The Measured Mineral Resources were based upon a confirmed understanding of the geological and grade continuity. Drill spacing is typically 25 m along the northerly strike, with often two to three holes per section. The Measured volumes also contain samples subject to DTR testwork, with associated assays from the recovered concentrates. Bulk density measurements were also available.

The Indicated Mineral Resources were based upon an assumed understanding of the geological and grade continuity. Drill spacing is typically 25–50/100 m along the northerly strike, with at least one hole per section. The Indicated volumes also contain samples subject to DTR testwork, with associated assays from the recovered concentrates. Bulk density measurements may also be available.

The Inferred Mineral Resources were based upon an implied understanding of the geological and grade continuity. Some mineralisation domains are only cut by one drillhole, and the geological models are strongly guided by surface mapping of the BIF outcrops. Drill spacing is typically ≥100 m along the northerly strike. DTR and bulk density results are generally absent from within the Inferred volumes, although the Sandalwood, Clark Hill North, Clark Hill South, and Snark Mineral Resources are supported by sufficient DTR testwork results to support the reporting of concentrate grade estimates.

1.7 Mineral Reserves

1.7.1 Reserves Statement

Mineral Reserve estimates for the Moonshine and Moonshine North open pits has been prepared by Steve Craig (QP) of Orelogy Consulting Pty Ltd as of 21 February 2022. The Mineral Reserves have been reported in accordance with Australian JORC 2012 and Canadian NI43-101 Technical Reporting standards.

The Mineral Reserves reflect that portion of the Mineral Resource which can be economically extracted by open pit mining methods. The Mineral Reserves considers the modifying factors including but not limited to the mining, metallurgical, social, environmental, statutory, and financial aspects of the project.

The Proven and Probable Mineral Reserves total 236.6 Mt at an average grade of 28.2% Fe using a cut-off grade of 15% DTR after applying dilution and ore losses. The total tonnage to be mined is estimated at 853.4 Mt at a strip ratio of 2.6:1. The Mineral Reserves are summarised in Table 1-4.

Table 1-4: Mineral reserve estimate – Lake Giles Iron Project, Moonshine and Moonshine North, DTR >15%

Category	Tonnes (Mt)	Head Grades (%)					Concentrate Grades (%)					
		Fe	SiO_2	Al_2O_3	P	LOI	DTR	Fe	SiO_2	Al_2O_3	P	LOI
Moonshine												
Proven	34.2	28.1	51.6	1.2	0.04	1.7	30.5	65.9	6.8	0.15	0.02	-0.6
Probable	166.4	27.2	51.9	1.4	0.05	1.4	30.7	66.6	6.2	0.11	0.02	0.0
Sub-total	**200.6**	**27.4**	**51.9**	**1.4**	**0.05**	**1.4**	**30.6**	**66.5**	**6.3**	**0.12**	**0.02**	**-0.1**
Moonshine Nth												
Proven	17.8	35.4	35.4	2.2	0.06	4.2	34.3	66.5	5.0	0.32	0.03	-0.9
Probable	18.2	30.4	44.7	1.3	0.05	2.9	35.9	63.2	9.4	0.24	0.04	-0.3
Sub-total	**36.0**	**32.9**	**40.1**	**1.7**	**0.05**	**3.5**	**35.1**	**64.8**	**7.3**	**0.28**	**0.05**	**-0.6**
Combined												
Proven	51.9	30.6	46.0	1.5	0.05	2.6	31.8	66.1	6.1	0.22	0.03	-0.7
Probable	184.7	27.6	51.2	1.4	0.05	1.5	31.2	66.2	6.6	0.13	0.02	-0.1
TOTAL	**236.6**	**28.2**	**50.1**	**1.4**	**0.05**	**1.8**	**31.3**	**66.2**	**6.5**	**0.15**	**0.02**	**-0.2**

Notes

1. Canadian Institute of Mining, Metallurgy and Petroleum "CIM Definition Standards for Mineral Resources and 2. Mineral Reserves" (CIM, 2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves are reported using a Davis Tube Mass Recovery (DTR MR) cut-off grade of 15% after applying dilution to the resource model.
3. Mineral Reserves were estimated using a 62% Fe benchmark price of USD100/dmt with a 20% premium for 65% Fe and concomitant Fe concentrate grade bonus.
4. Mineral Reserves account for mining dilution of 2.5% at a graded 14% DTR and mining ore loss of 2.0% at a grade of 30% DTR.

5. Mineral Reserves are reported on a Dry Tonnage Basis with an average bulk density of 3.2 t/m3.
6. The average strip ratio is 2.6:1.
7. Mineral Reserves are a part of Mineral Resources.
8. Proven Mineral Reserves are based on Measured Mineral Resources only and Probable Mineral Reserves are based on Indicated Mineral Resources only.
9. The sum of individual amounts may not equal due to rounding.

1.8 Mining

1.8.1 Mining Method

The Moonshine and Moonshine North pits will be mined using conventional open pit mining methods based on 350-400 t class hydraulic excavators loading 180 t class rear dump trucks. The operation is proposed using experienced mining contractors with the Owner maintaining orebody definition, quality control and medium to long term mine planning functions and management. The mining services include:

- Supply of personnel, equipment and mining infrastructure required for the mining services excluding diesel fuel which is to be supplied by The Owner.
- Mobilisation of buildings, equipment, and personnel.
- Clearing and stripping of suitable material from all disturbed areas into discrete stockpiles.
- Construction of haul roads and light vehicle service roads in the mine area and ongoing maintenance of haul roads.
- Construction of the Run-of-Mine (ROM) pad and skyway using bulk waste.
- Grade control drilling.
- Drilling and blasting of ore and waste on 10 m benches.
- Load and Haul utilising 350-400 t class excavators and 180 t class haul trucks mining on 5 m high flitches.
- Hauling waste to external waste dumps.
- Hauling ore to the ROM pad where it will be direct fed to the crusher or placed onto a finger from skyway or stockpile adjacent to the ROM pad.
- Rehandle of ore from ROM fingers or adjacent stockpiles.
- Rehandle of dry LIMS reject from the plant to the waste dump.
- Ongoing pit dewatering from in-pit sumps.
- Rehabilitation of waste dumps and roads.

1.8.2 Pit Optimisation

The open pit optimisation process undertaken for this study has the following key assumptions on the constraints and parameters utilised:

- Only material classified as Measured and Indicated in the Mineral Resource model were considered as potential ore during the optimisation process.
- Mining dilution (averaging 2.0%) and mining recovery (averaging 97.5%) were modelled in the block model.
- Waste mining costs were applied in the mining model based on unit rates averaging A$2.54/dmt.
- A net product price of A$145.50 (after deducting 5% government royalty).
- Ore processing rate of 9.68 Mt/year at a cost of A$13.45/dmt.
- Ore handling costs of A$2.99/dmt were added for additional ore mining cost, grade control, ore feed and reclaim from stockpile using Contractor unit rates.
- Annual fixed mining overheads for the Owners team were applied as a unit rate of A$1.26/t to the ore tonnes processed.
- Logistics costs of A$29.64/dmt for road, rail and port charges were supplied by MIO.

- Overall pit slope angles of 27-33º in oxide and 27-41º in Fresh rock were based on geotechnical recommendations by Pells, Sullivan & Meynink.
- Shell 20 with a revenue factor of 0.88 and a mine life of 25 years was selected as the basis for design. This shell captured 99% of the value within a pit containing 93% of the ore and 89% of the waste (when compared to the revenue factor 1.0 shell).

1.8.3 Mine Design

The design process provides a practical solution to the Whittle shells by adding an arrangement of benches, berms, roads and ramp systems. Dual lane ramps of 29 m and 10% gradient were designed to accommodate Caterpillar 789D trucks.

The final pit design comprises two separate pits with a total of seven internal stages. An overview of the final pit showing internal stages is presented in Figure 1-2.

Moonshine North pit is approximately 1,450 m long, 500 m wide and 225 m deep and Moonshine is approximately 3.7 km long, 700 m wide and 250 m deep. Each stage has a separate ramp system that exits on the west side to provide short hauls to waste dumps and ROM pad. The design process captured 0.9 % additional ore and added 6.6% additional waste than defined by the Whittle shell.

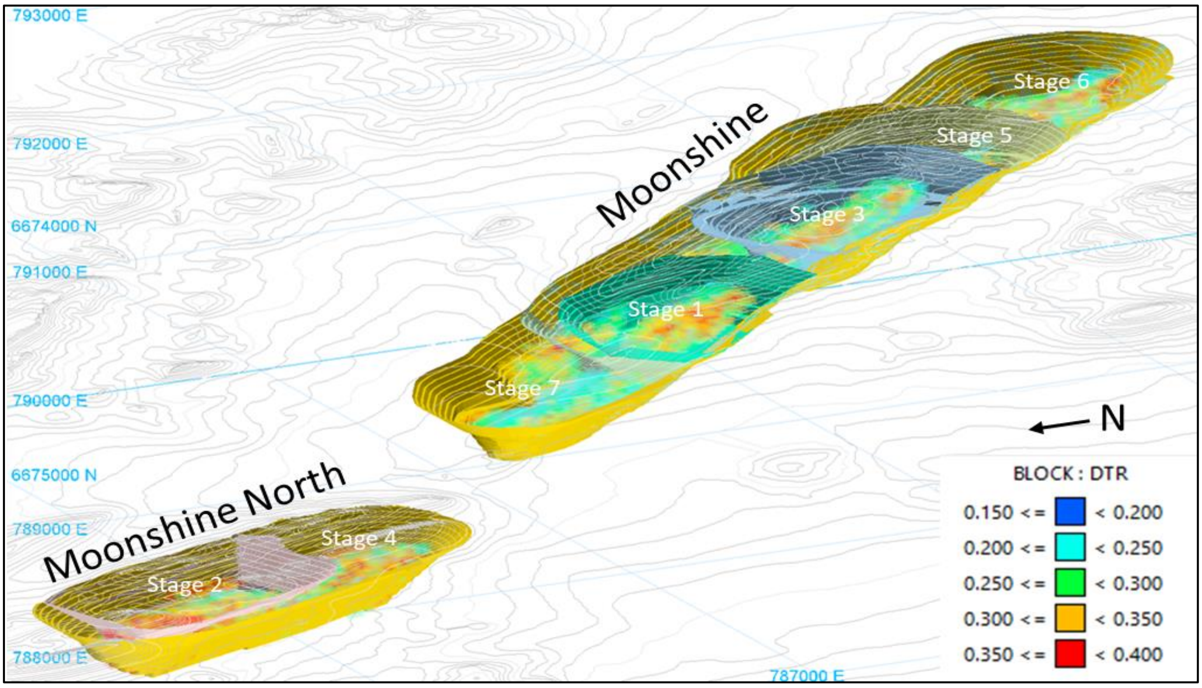


Figure 1-2: Pit designs showing stages and mineralisation coloured by DTR

The final pits contain a total of 236.6 Mt at an average grade of 28.2% Fe and 31.3% DTR reported above a cut-off grade of 15% DTR. The total tonnage to be mined is estimated at 861.5 Mt at a strip ratio of 2.6:1. The Moonshine pits contains 85% of the magnetite ore with a lower strip ratio at 2.4:1 compared to the smaller Moonshine North pit which has a strip ratio of 3.8:1. The inventory by stage is presented in Table 1-5.

Table 1-5: Moonshine and Moonshine North pit inventories reported by stage

Stage	Ore	Grades							Waste	Total	S/R
	Mt	Fe %	SiO_2 %	Al_2O_3 %	P %	S %	LOI %	DTR %	Mt	Mt	W:O
1	22.4	28.3	50.5	1.5	0.05	1.2	1.2	31.2	53.8	76.2	2.4
3	22.2	27.8	51.6	1.2	0.05	0.9	1.4	31.2	65.3	87.5	2.9
5	69.9	27.3	51.9	1.3	0.05	1.0	1.4	30.7	154.0	223.8	2.2
6	55.9	27.4	52.1	1.3	0.05	0.9	1.4	31.1	135.3	191.3	2.4
7	30.2	26.7	52.7	1.8	0.04	1.1	1.7	28.5	79.1	109.2	2.6
Moonshine	**200.6**	**27.4**	**51.9**	**1.4**	**0.05**	**1.0**	**1.4**	**30.6**	**487.5**	**688.0**	2.4
2	6.4	31.8	43.8	1.3	0.05	1.3	3.0	35.0	52.1	58.5	8.2
4	29.6	33.1	39.3	1.8	0.06	1.5	3.7	35.1	85.4	115.0	2.9
Moonshine Nth	**36.0**	**32.9**	**40.1**	**1.7**	**0.05**	**1.4**	**3.5**	**35.1**	**137.5**	**173.5**	3.8
TOTAL	**236.6**	**28.2**	**50.1**	**1.4**	**0.05**	**1.1**	**1.8**	**31.3**	**624.9**	**861.5**	2.6

The blending strategy to manage silica levels reporting through to the concentrate requires stockpiling on long term stockpiles. All material above a DTR cut-off of 29% was categorised as primary ore feed. The material below 29% and greater than 15% DTR was split into low and high silica categories as follows:

- Low Silica stockpile: SiO2 in concentrate < 6.7%; and
- High Silica stockpile: SiO2 in concentrate >= 6.7%

Primary feed ore will be hauled to the ROM pad and direct tipped into one of two crusher pockets or placed on temporary finger stockpiles from a skyway for later rehandle using a FEL. Each of four fingers has been designed with a capacity of 96,000 dt ore, sufficient for 14 days of feed. Based on the disparity between the primary crusher (1,265 t/h) and the excavator (2,080 t/h), the proportion of direct tip into the primary crusher is estimated to be approximately 60%.

A total of 295 Mt of oxide overburden requires pre-stripping to expose the ore within the Fresh BIF rock units. A further 328 Mt of fresh waste rock will be mined over the life of the operation. The waste material will be stored in three external waste dumps designed to a maximum height of 60 m.

The overall strategy for haul road design was for a central road linking the Moonshine and Moonshine North pits. The ROM pad was located close to the centre of mass between the two pits. Access to the waste dumps branch off the main haulage corridor providing flexibility for dumping of waste material.

Prior to commencement of mining the disturbed areas will be cleared and the topsoil removed and stored in various stockpiles around the site. These have been strategically located to minimise haulage distances both during stripping and when reclaimed for rehabilitation of the waste rock dumps. The topsoil locations are shown in the general site layout plan, Figure 1-3.

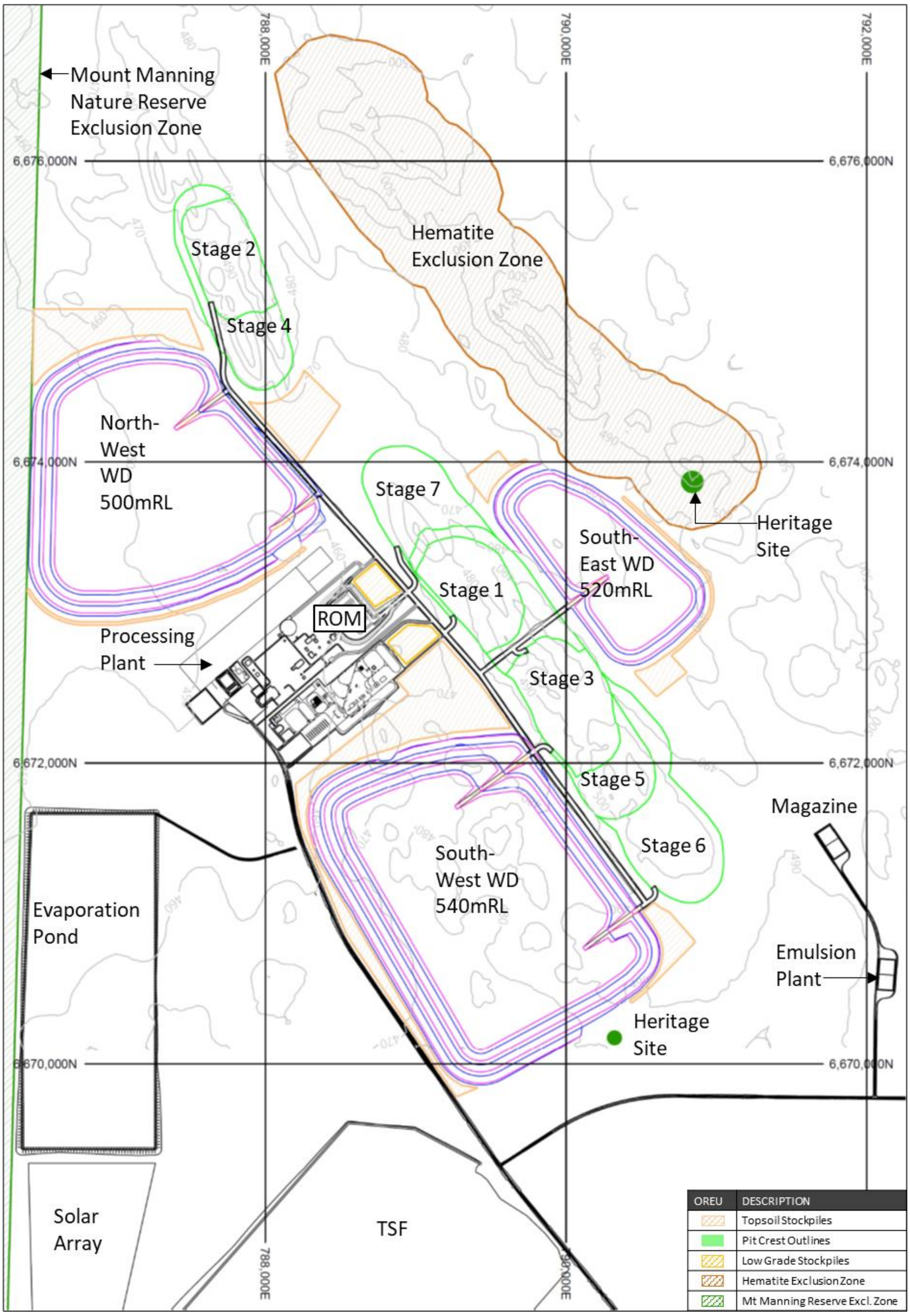


Figure 1-3: General site layout

1.8.4 Mine Production Schedule

Pre-production required a total of 28.1 Mt mined over an 11-month period comprising mostly oxide waste from Stage 1 with 164 Kt ore stockpiled for processing. The peak mining rate of approximately 43 Mt/year utilising 3 excavators is reached in year 2 and maintained for 14 years. The mining rate is reduced to two excavators for 7 years before reducing to a single excavator for the final 5 years of operation. Figure 1-4 illustrates the oxide pre-strip and fresh waste movements compared to the ore mined to the ROM pad for processing or to stockpile for blending.

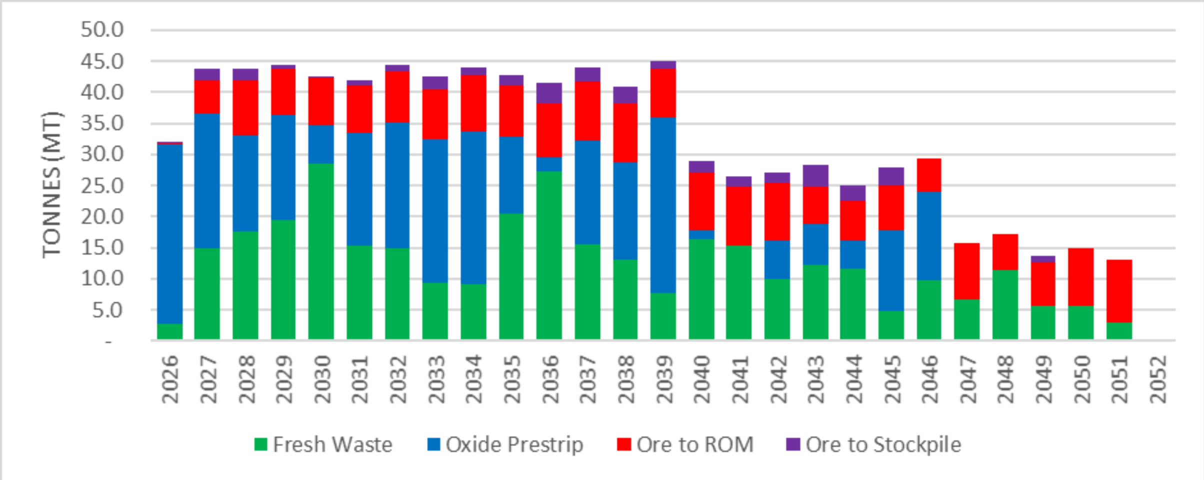


Figure 1-4: Annual material mined by type

The Contract mining operation will be conducted with two 12-hour shifts per day. Both Owner and Contractor management, technical and support personnel will work a 10-hour day shift. All personnel will be sourced from either Perth or Kalgoorlie on fly-in fly-out (FIFO) basis.

The operation will require a total fleet of 42 mining units comprised of three primary 350t excavators, eighteen 180t dump trucks, five dozers, two graders, two water carts, five drills, a wheel dozer,a small excavator for ancillary work, two large Front-end loaders, two trucks for rehandle and a single RC rig. Manning levels will vary over the life of mine peaking at 307 personnel in 2029, including 38 Owners staff and 13 contractor staff.

1.9 Mineral Processing and Metallurgical Testing

Engenium (2010) carried out preliminary studies based on samples from two RC holes (LGRC199 and LGRC203); one each from the Moonshine and Moonshine North deposits.

The previous testwork encouraged the latest recent drilling programme to assess the deposits in some detail. The drilling programme collected HQ sized core and split the core for assays and Davis Tube testwork. Half core was available for testing. As core was limited, sample selection focussed on maximising the inclusion of mineralised ore, whilst also including diluting intervals not rejectable by selective mining. The interval considered was a half bench height of six metres.

The composite details are provided in Table 1-6. The drillhole locations are shown as green in Figure 1-5.

Table 1-6: Testwork composite details

Prospect	Hole Identification	Core Selected at BV			Sample Mass (kg)
		Start	**End**	**m**	
Moonshine	LGDD_006	144.0	265.0	121.0	965
Moonshine	LGDD_066	83.7	165.3	81.6	407
Moonshine	LGDD_067	69.0	135.6	66.6	332
Moonshine	LGDD_068	83.0	193.5	110.5	551
Moonshine	LGDD_069	88.0	115.0	27.0	135
Moonshine	LGDD_070	88.0	132.8	44.8	223
Moonshine	LGDD_070	143.0	152.4	9.3	47
Moonshine	LGDD_070	166.0	173.5	7.5	37
Moonshine	LGDD_072	56.3	117.2	61.0	304
Moonshine	LGDD_073	110.4	140.9	30.6	152
Moonshine	LGDD_073	200.0	269.7	69.7	347
Moonshine	LGDD_023	101.1	198.7	97.6	973
Moonshine Total				**630**	**4473**
Moonshine North	LGDD_071	81.8	162.0	80.2	400
Moonshine North	LGDD_074	47.1	71.0	23.9	119
Moonshine North	LGDD_074	80.7	98.9	18.2	91
Moonshine North Total				**122**	**610**

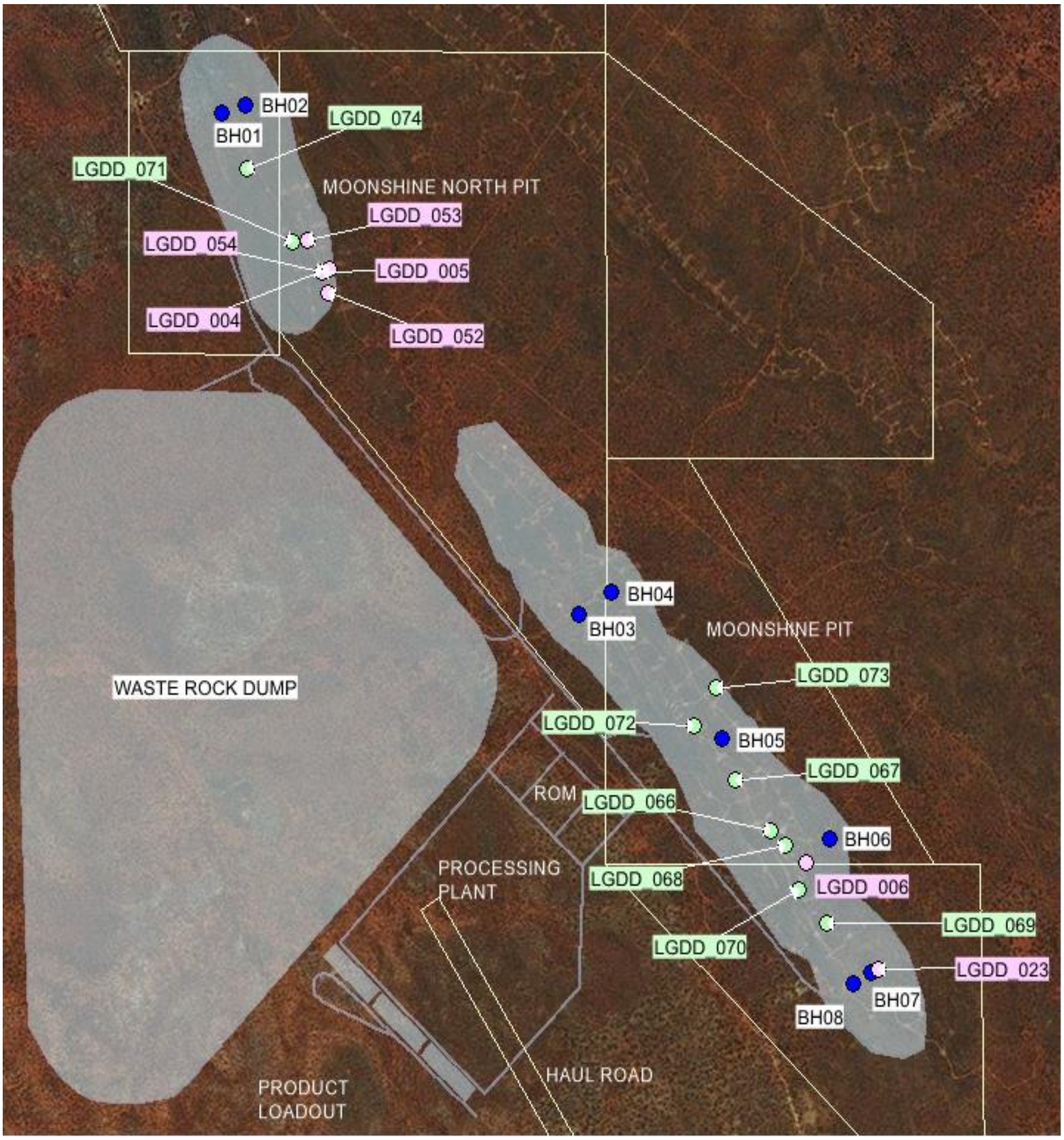


Figure 1-5: Magnetite testwork core drillhole locations (green) shown on preliminary layout

The testwork was performed at the Bureau Veritas Laboratory (BV) in Canning Vale, Western Australia, an ISO9001 certified laboratory.

There were two test plans developed, one for magnetic separation and one for high pressure grinding rolls (HPGR) testwork. The test plans are detailed in in the standalone metallurgical test work report.

1.9.1 Head Assays

The composite head assays for Moonshine and Moonshine North are below.

Table 1-7: Composite head assays

Composite	Fe	SiO_2	Al_2O_3	P	S	LOI-1000
	%	%	%	%	%	%
Moonshine Actual	30.7	50.3	0.37	0.04	0.56	-0.12
Moonshine Expected	30.1					
Moonshine North Actual	32.8	47.4	1.03	0.05	1.21	0.68
Moonshine North Expected	32.7					
Note: Hole 23 assays not included in Moonshine calculations, as they were unavailable.						

The results were assessed as they were made available, and this changed some test parameters during the programme. The results are tabulated below.

Table 1-8: Testwork summary

Testwork	Unit	Moonshine	Moonshine North
Head Assays			
Assay Fe Grade	%	30.7	32.8
SiO2 Grade	%	50.3	47.4
Al2O3 Grade	%	0.37	1.03
P Grade	%	0.04	0.05
S Grade	%	0.56	1.21
LOI Grade	%	-0.12	0.68
In Situ SG		3.46	3.46
Concentrate BD Unconsolidated	t/m^3	1.88	1.95
Concentrate BD Consolidated	t/m^3	2.39	2.48
Abrasion Index		0.58	0.53
BWI @ 75 µm	kWh/t	13.5	14.9
BWI @ 125 µm	kWh/t	13.5	14.9
SMC A*b		37.6	38.7
DTR @ 38 µm Fe Grade	%	65.0	65.7
SiO2 Grade	%	12.7	8.5
Mass recovery	%	40.9	43.7
HPGR			
Press Force	N/mm2	4.1	
Total Throughput	t/h	38.6	
-2.8 mm in centre sample	%	51.4	
-2.8 mm generated	dtph	19.8	
Specific throughput	$(t/h)/(m^3/s)$	259.3	
Specific power input	kWh/t	2.1	
Predicted recirculating load	%	116	
Predicted power input (of product)	kWh/t	4.54	
Magnetic Separation			
Coarse Cobbing at -6 mm			
Mass Recovery	%	83.9	74.8
Fe Grade	%	32.5	38.6
SiO2 Grade	%	45.9	41.3

S Grade	%	0.36	1.08
BBWi Of CC Product @ 125 µm	kWh/t	12.8	13.3
Single Stage LIMS @ 212 µm			
Mass Recovery	%	66.5	57.6
Fe Grade	%	42.5	46.6
SiO2 Grade	%	37.0	31.5
Al2O3 Grade	%	0.14	0.20
P Grade	%	0.043	0.049
S Grade	%	0.52	0.81
2 stage LIMS @ 106 µm			
Mass Recovery	%	51.8	45.8
Fe Grade	%	52.3	55.6
SiO2 Grade	%	24.7	20.0
Al2O3 Grade	%	0.10	0.16
P Grade	%	0.030	0.036
S Grade	%	0.22	0.69
2 stage LIMS @ 38 µm			
Mass Recovery	%	43.6	37.7
Fe Grade	%	61.3	64.3
SiO2 Grade	%	13.6	9.4
Al2O3 Grade	%	0.05	0.07
P Grade	%	0.020	0.022
S Grade	%	0.19	0.60
Reverse Flotation			
Mass Recovery	%	35.3	32.3
Fe Grade	%	68.3	68.2
SiO2 Grade	%	4.2	3.9
Al2O3 Grade	%	0.04	0.07
P Grade	%	0.018	0.019
S Grade	%	0.19	0.54
Tailings Thickening			
Solids Loading	t/hr m2	1.5	1.5
Flocculant dosage	g/t	20	10
Flocculant		Magnafloc 155	Magnafloc 155
Overflow Clarity	mg/L	140	130
Underflow density	% solids w/w	63	64
Diameter @ 691 dt/h	m	25	25
Tailings Filtration			
Pressure Filter			
Filtration Rate	kgDS/m2 h	305	358
Cake moisture	% solids w/w	11.4	11.8
Filtrate clarity	ppm	280	110
Vacuum Filter			
Filtration Rate	kgDS/m2 h	394	746
Cake moisture	% solids w/w	16.1	17.0

A discussion of the results alongside the resource model, led to the project product being defined as below. If this specification is found to be unsuitable, due to the high sulphur content, further work will be needed to address the issue.

Table 1-9: Product specification

Fe %	Al_2O %	SiO_2 %	P %	LOI %	S %
66.1	0.10	4.9	0.02	-2.7	0.6

1.9.2 Recovery Methods

In order to produce 3.0 dMtpa of concentrate, assuming a weight recovery of 31%, 10 Mtpa of feed to the process would be required. Two stages of conventional crushing would crush the ore to a size suitable for feed to a High-Pressure Grinding Rolls (HPGR) unit. The fine ore grinding section contains two streams in parallel each containing two stages of mills, with Low Intensity Magnetic Separation (LIMS) units after each stage. This is followed by reverse flotation and a final LIMS stage. The final concentrate moisture is reduced by pressure filtration allowing stockpiling and transport by truck.

A flowsheet for the operation is shown below:

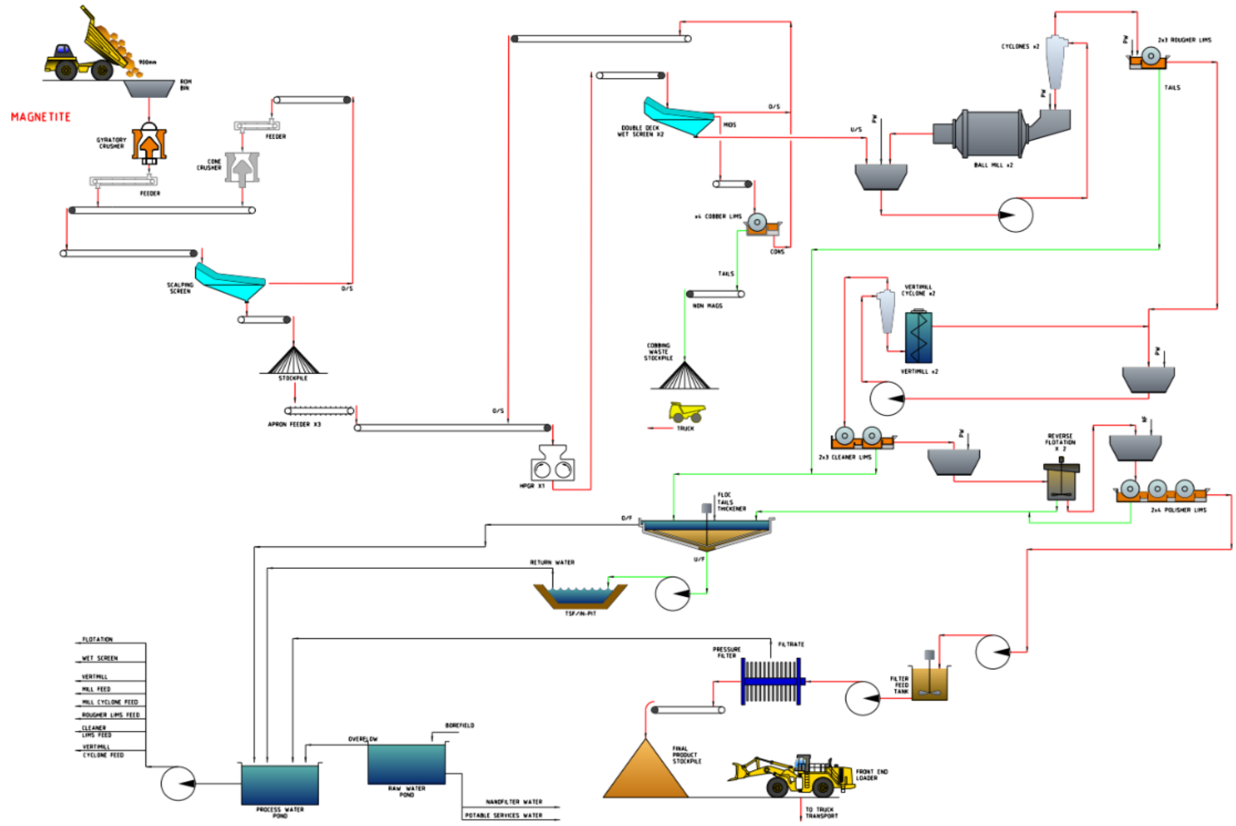


Figure 1-6: Conceptual project flowsheet

1.10 Site Infrastructure and Logistics

1.10.1 Logistics

The study has identified the preferred logistics option of hauling the product by private road to a rail loop on the existing Eastern Goldfields Railway (EGR), transport by rail to Esperance and then loading onto cape class vessels for export.

Product will be transported from the mine by road to a rail loop, east of Mt Walton station, approximately 93 km south of the Project and then by rail to the Port of Esperance for export. The logistics chain includes road haulage along a private haul road utilising triple road-trains with side tip trailers, stockpiling at the rail siding, rail transport with rotary tippling wagons to the Port of Esperance, unloading by a rail car dumper, stockpiling in a covered shed, reclaim by FEL and loading onto ships via the Berth 3 ship loader.

1.10.2 Port

To facilitate export from the Port of Esperance, new infrastructure is required to store the concentrate and discharge it onto existing bulk product ship loading facilities. Existing rail unloading facilities, expected to have adequate capacity, will be utilised to unload rotary car wagons and direct product to the storage shed. Reclaim of material from the storage shed will be managed by SPA contractors with ship loading via the existing outload circuit and Berth 3 ship loader. The Feasibility Study also considered an alternate case should rail unloading capacity be unavailable. Under this scenario a new rotary car dumper would be constructed at the port in addition to a rail loop to allow a full consist to enter the port.

Engineering studies were completed, and costs developed for the development. However, given recent reductions in iron ore throughput at the port, the Company considers sufficient capacity is likely to exist and therefore, the base case assumes access to existing facilities.

1.10.3 Infrastructure

Non-process mine infrastructure has been designed and costed by Engenium. Where applicable, the Company has elected to develop several facilities under a build-own-operate (BOO) model funded and managed by interested third parties. Such facilities include the laboratory and power station with pricing treated as an operating cost over the term of the proposed contract.

1.10.4 Power

40 MW of power supply will be required for the Project inclusive of the magnetite process plant and supporting non-process infrastructure. Macarthur engaged power supply analyst firm, Veckta to review and determine the most cost-effective power solution for the main power plant located at the mine. The preferred option for the Project is a hybrid solution of solar, battery storage and natural gas reciprocating engines.

1.10.5 Water

The total annual water requirement for the Project is estimated to be 4 Gl, supporting a mineral processing facility operating at a nominal product production 3.0 Mtpa rate along with all associated non-process infrastructure (excluding the port) and dust suppression.

A bore field will be constructed to source water to supply the Project's construction and subsequent process and potable water requirements.

Project estimates indicate 26 fully equipped bores will be required to meet the Project water demand of 466 kL/h, based on nominal flow rates of 5 L/sec. Macarthur is currently progressing an application for tenure across this area to complete the required drilling activities.

1.10.6 Communications

A conventional VHF radio system would allow communications coverage for the entire site. These VHF radio systems will be mounted on five of 20 m communications towers located at the mine site, the tailings storage facility, the village and two along the haul road leading to the rail loop. The same communications towers will provide internet services for the mine site and the village via a microwave link from a communications carrier.

1.10.7 Access

The project can be accessed by heading approximately 130 km west from Kalgoorlie via the Great Eastern Highway, north approximately 45 km along the unsealed access road and then east after the Eastern Goldfields Railway (EGR) level crossing. The access road adjacent to the EGR follows the track towards Kalgoorlie before heading north towards the gatehouse. Once the mine is operational the gatehouse will be the only access into the Project. The product haul road intersects the access road immediately after the gatehouse and is utilised to access the mine, village and airport to the north and the rail loadout area to the south.

1.10.8 Mine Administration

The Mine Operations Centre will include the following:

- main administration area,
- mining contractor area,
- haulage contractor area,
- fixed plant area,
- process buildings,
- permanent laydown area, and
- mining contractor facilities.

The Mine Operations Centre would largely comprise of modular buildings and dome shelters. Larger facilities such as warehouses and workshops will be constructed in-situ.

1.10.9 Accommodation

Due to the size and location of the Lake Giles Iron Project, accommodation is required to support the construction and operation of the mine. It is proposed to construct the camp approximately 10 km east of the mine. The camp will be a “design and construct” package to be performed by a specialist construction contractor with the operation of the village to be performed by a specialist camp operator.

Initially, the camp would accommodate the mining contractor for pre-strip operations and the mine construction contractors for a period of approximately one year. Based on availability at the time of construction of the village, it is envisaged that of 720 rooms required at peak capacity, 280 rooms will be leased for the duration of the construction with the remaining 440 rooms owned by Macarthur.

1.10.10 Airport

The proposed airport is located approximately 2 km south of the camp and was chosen as the preferred site due to geotechnical and hydrological conditions, topography, and proximity to the camp and mine site.

1.10.11 Tailings Storage Facility

A total of 137.5 Mt of tailings (dry) will need to be stored at the tailings storage facility during the 25-year LOM. The previous study designated a suitable site for the tailings storage facility to minimise the volume of embankments by utilising a ridge as containment, maximise stored capacity by selecting a relatively uniform area, ensure seepage / runoff is contained within a single catchment and to ensure runoff from the catchment upstream can be diverted around the tailings storage facility.

Utilising the centreline construction method, an initial starter embankment for the tailings storage facility will be constructed, with the elevation subsequently increasing in years 3, 8 and 15. A pontoon pump will be required for the decant water return to the process water dam until a tailings beach can be established. Once a tailings beach has been established, the pontoon will be replaced by a decant tower.

1.10.12 Approvals

The main legislation that governs environmental protection at the Federal level is the EPBC Act. It provides a legal framework to protect and manage nationally and internationally important flora, fauna, ecological communities and heritage places – defined in the EPBC Act as Matters of National Environmental Significance (MNES). Matters of NES have been identified within the Project area that will trigger referral of the Project to the Department of Agriculture, Water and Environment (DAWE).

The primary legislation for environmental protection in Western Australia (WA) is the Environmental Protection Act 1986 (EP Act). Regarding mining approvals, projects may require assessment under two separate parts; Part IV and Part V, administered by the Environmental Protection Authority (EPA) and the Department of Water and Environment Regulation (DWER), respectively.

Under Part IV of the EP Act, proposals are referred to the EPA for a decision on whether the project has the potential to cause significant impacts on the environment.

Under Part V of the EP Act, secondary approvals such as Works Approvals and Operating Licences will be required for Prescribed Activities and facilities that result in discharges to the environment. These applications will be submitted upon receiving more detailed information on Project design and infrastructure requirements. Works approvals and licences will be required to operate the beneficiation process plant, tailings storage facility, sewage pond and site landfill.

Approval under the Mining Act 1978 is also required for mining projects and is administered by the Department of Mines, Industry Regulation and Safety (DMIRS). Approval under this Act involves the assessment of a Mining Proposal and Mine Closure Plan. In addition, if the Project is not assessed by the EPA under Part IV of the EP Act, then DMIRS are also required to assess Native Vegetation Clearing Permit applications.

Approval to disturb Aboriginal heritage sites may be required under the Aboriginal Heritage Act 1972. At this stage, there are no registered sites and consultation with the Traditional Owners is ongoing to determine the significance of potential sites.

An environmental impact assessment is required to obtain environmental approval for development. Macarthur has commenced the scoping process to identify the key environmental risks and level of survey to be undertaken. An approval pathway and schedule for the primary and secondary approvals required has been mapped out and Macarthur intends to commence desktop and baseline surveys at the conclusion of the feasibility study. Macarthur has previously gained EPA approval for its adjacent hematite project and is not aware of any major environmental obstacles that would prevent approval of the Project.

As the Lake Giles deposit lies within the Goldfields Groundwater Area, a DOW 26D licence is required to construct bores, and a 5C licence is needed to take water. Water bore drilling contractors are required to hold a water well drillers licence to construct bores. Approval for a Programme of Works (POW) is needed from the Department of Industry and Resources (DOIR) to clear areas for drilling.

1.11 Native Title

The Project sits within the Marlinyu Ghoorlie native title claim. The claim was registered on 28 March 2019 but is currently not determined. Native title rights in registration or grant give claimants the right to negotiate during the grant of mineral tenure. Macarthur's Mining Leases were all granted prior to registration of the Native Title claim and the current claim does not confer rights to negotiate or affect the tenure. There were no Native Title claims over the area at the time of grant and therefore no access agreements were required to be negotiated with Claimants. Current applications for tenure as described in Section 4.2 are subject to native title. Macarthur is currently progressing heritage agreements with the native title claimants to progress the tenure to grant.

A search of the Department of Indigenous Affairs (DIA), Aboriginal Heritage Inquiry System confirms that there are no registered heritage sites on any of the tenements within the Project area (DIA 2011).

Heritage surveys have been conducted in accordance with EPA Guidance Statement No.41 (EPA 2004a) across some areas, including both archaeological and ethnographical surveys. To date, one archaeological site has been identified within the Project area. The location of the heritage site does not impact the Project and a suitable buffer distance has been employed to avoid any impact to the site. Additional surveys will be undertaken with the traditional owners across outstanding project areas in due course.

Macarthur will work towards mutually beneficial outcomes through a commitment to community consultation and ongoing liaison. Macarthur facilitates local direct employment and indirect employment, endeavours to support training and development initiatives related to exploration, future mining and ancillary services. Macarthur respects cultural diversity, connection to country and encourages sustainable business relationships.

1.12 Market Studies

The forward iron ore price adopted for the Lake Giles Iron Project in this Report is based on the Company's assessment of published consensus pricing, forecasts derived directly from steel mills, the various analyst reports described below and a comparison of historical analyst forecasts against actual pricing over time. The Company has then adopted an adjustment for grade using historical and projected premiums to arrive at a long-term price for the Lake Giles Concentrate.

Information on current and forward product demand characteristics, product marketing and pricing were supplied by Glencore and were also derived from published research reports prepared by Wood Mackenzie and major global iron ore producers and marketers (such as BHP's published price and market forecasts).

Macarthur also engaged LFJ Consulting Pty Ltd (LFJ) to undertake an iron ore market and price analysis. LFJ's analysis has been considered and utilised in the preparation of the market studies chapter in conjunction with the other market data referred to above.

A long-term CFR China sales price of US$131.40/dmt for Macarthur's 66.1% Fe concentrate product specification has been adopted, based on forecast pricing for 62% Fe CFR China of US$99/dmt through to 2050 with an adjustment for grade and a magnetite premium. This is expected to result a realised free on board (FOB) sales price of USD$120.30/dmt after shipping and marketing costs.

The pricing scenario is consistent with the Company's determination of current consensus price forecasts to 2050. The Company considers that the pricing scenario is appropriate for the Lake Giles concentrate product specification, and it has been normalised for the highs and lows experienced throughout early 2020 and into mid-2021 (which was a period that largely reflected the market response to the uncertainties of the Covid-19 pandemic).

The pricing analysis is based on iron ore industry knowledge, experience, and on information available from company, industry, trade, government and other sources that may be limited. The analysis, estimates or projections considered for this Study, together with other sources of input, are based upon information and upon assumptions that are subject to significant degrees of economic, commercial, market, industrial and other uncertainties.

1.13 Capital Costs

The capital estimate has been developed with an expected accuracy range of between +/-10% to +/-15% (AUSIMM Class 3), based on engineering to 25% definition.

All costs are estimated based on the pricing for labour and materials existing in Q4 2021. Escalation of costs beyond this date is not included in the capital cost estimate.

The cost estimate for initial development capital excluding deferred and sustaining capital is shown in Table 1-10. Total project capital inclusive of LOM sustaining capital and deferred capital costs are shown in Table 1-11.

Table 1-10: Summary of direct & indirect capital costs

Area	USD M	AUD M
DIRECTS		
Facilities process plant	11.6	16.4
Process plant	227.6	320.5
Product transport logistics	36.5	51.4
Port storage & ship loading	24.2	34.0
Infrastructure & headworks	72.0	101.3
General and administration	1.3	1.8
Total direct costs	**373.1**	**525.5**
INDIRECTS		
Construction Indirects	83.6	117.8
EPCM	52.2	73.5
Spares & Commissioning	4.8	6.8
Freight	11.2	15.7
Contingency	43.9	61.9
Total indirect costs	**195.7**	**275.7**
Total Direct & Indirects	**568.8**	**801.1**
MINE DEVELOPMENT		
Capitalised pre-strip	43.8	61.6
TOTAL PROJECT CAPITAL	**612.5**	**862.7**

Table 1-11: Summary of initial and deferred capital costs

Initial Capital Expenditure	USD M	AUD M
Construction Capex	568.8	801.1
Capitalised Pre-Production Operational Costs	43.8	61.6
Total Initial Capital Expenditure	**612.5**	**862.7**
Future Capital Expenditure		
Sustaining Capex	143.8	202.5
Tailing Storage Facility Lifts	28.2	39.8
Capitalised Non-Operational Waste Mining Costs	252.5	355.7
Total Future Capital Expenditure	424.6	598.0
Closure & Rehabilitation Costs		
Closure and Rehabilitation Cash Expenses	41.32	58.20

1.14 Operating Costs

The operating cost estimate was compiled by Engenium utilising information provided by Macarthur Minerals, Orelogy and the Engenium Project team.

The cost estimate is a Feasibility Study level estimate with an expected accuracy range of between +/-10% to +/-15% (AUSIMM Class 3), based on engineering to 25% definition.

All costs are estimated based on the pricing for labour and materials existing in Q4 2021. Escalation of costs beyond this date is not included.

Table 1-12: Operating Costs

Area	USD/t	AUD/t
Mining	26.08	36.73
Crushing & Processing	22.41	31.56
Logistics	21.25	29.93
General & Administration	2.00	2.82
Subtotal	**71.74**	**101.05**
Royalties	6.05	8.51
Total operating costs ($/t concentrate)	**77.79**	**109.56**

1.15 Financial Analysis

A discounted cash flow model at a discount rate of 6% was used to derive pre- and post-tax NPV for the Project. All figures are presented in AUD unless otherwise specified.

At a 6% discount rate, the model shows a pre-tax NPV of $816 M with an IRR of 13.0%. After tax the NPV is $443 M with an IRR of 10.1%.

Total operating cash flows equal $2,979 M with an after-tax cash flow of $2,106 M.

The project generates a total of $1,475 M payable to government comprising $844 M in Federal taxes and $631 M in royalties for the Western Australian Government.

The outcomes of the base case financial valuation at 6% discount rate are shown in Table 1-13.

Table 1-13: Summary of Economic Outcomes

Production	
Ore mined	236.6 Mt
Waste mined	624.9 Mt
Total mined	861.5 Mt
Strip ratio	2.64
Concentrate produced	74 Mt
Concentrate iron grade	66.1
Plant recovery	31%
Financials	**AUD M**
Sales revenue	12,614
Operating Expenses	8,116
Initial Capital Expenditure	
Construction capex	801.1
Mining overburden pre-strip	61.6
Total initial capital	862.7
Future Capital Expenditure	

Sustaining capital	203
Deferred capital - Tailings	39.8
Capitalised non-operational waste mining	355.7
Total future capital	598.0
Closure Expenditure	
Closure and rehabilitation	58.2
Total Capital Expenditure	1,460.7
Total Operating Cash Flows	3,625
Taxes & Royalties	
Tax paid	873
Royalties	631
Valuation	**AUD M**
NPV (6%) Pre-tax	816
NPV (6%) Pre-tax	443
IRR Pre-tax	13.0%
IRR Post-tax	10.1%

1.16 Conclusions

The Feasibility Study confirms an economically viable project producing 3 million tonnes per annum (dry basis) of high-grade magnetite concentrate over a 25-year mine life. The project will leverage off access to existing regional rail and port infrastructure and deliver a premium concentrate (66% Fe) product with low impurities. The Feasibility Study underpins a maiden Ore Reserve of 237 million tonnes.

1.16.1 Geology and Mineral Resources

A Mineral Resource estimate has been prepared for the Lake Giles Magnetite Project, based upon a total of 359 RC drillholes and 14 diamond holes. Results from these drillholes, and from geological field mapping and observations, provided the basis for the geological interpretations. Macarthur provided the geological and mineralisation interpretations to CSA Global as 3D wireframe solids and surfaces. CSA Global flagged the drillhole samples within the mineralisation domains, and geostatistical studies carried out for the head and concentrate assay data, including variography to ascertain the spatial variation of the various grade variables.

3D block models representing the mineralisation was created using Datamine software. High quality diamond and RC drillhole samples were used to interpolate head and concentrate grades into the block model using ordinary kriging. The block models were validated visually and statistically.

Mineral Resources are reported for the Moonshine, Moonshine North, Sandalwood, Clark Hill North, Clark Hill South, and Snark magnetite deposits. The Mineral Resource estimates are classified as a combination of Measured, Indicated and Inferred, in accordance with 2014 CIM Definition Standards. The classification level is based upon an assessment of the geological understanding of the deposit, QAQC of the samples, mass recovery results, density data and drillhole spacing.

1.16.2 Mining and Mineral Reserves

The Mineral Reserve estimate has been prepared for the Moonshine and Moonshine North pits in accordance with 2014 CIM Definition and Standards with more than 20% based on material classified as Measured and the remainder as Indicated.

The geometry of the wide sub-vertical orebodies is amenable to bulk mining methods with low dilution and ore loss. The continuity of the orebodies is also favourable for blasting along strike to minimise dilution on the edges of the orezones.

Optimised pit shells bottom out on the Measured and Indicated resources, however, deeper drilling is not warranted at the current iron ore price due to the high strip ratios to access the ore at depth.

Silica reporting through to concentrate was identified as a primary driver of ore feed to the plant. The Moonshine North pit contains ore with higher mass recovery based on the DTR grade but this is associated with substantially high SiO2 in concentrate compared to the Moonshine pit. The blending strategy, to manage the DTR grade and the silica reporting to the concentrate, is sensitive to the extraction sequence and operation of the mine will require tight controls to ensure that the short-term schedules are kept in line with the life of mine plan.

The operating strategy using experienced mining contractors with the Owner maintaining orebody definition, quality control, supervision and management reduces the operational risk at start-up and provides opportunity for value improvement by transitioning to Owner mining once the operation becomes steady state.

Orelogy has relied upon foundation data supplied by other experts in the preparation of the mine plan for the Lake Giles Project. The QP assessed the information provided and is confident that the data is of a standard for reporting the Mineral Reserve at Feasibility level.

1.16.3 Mineral Processing

A discussion of the metallurgical testwork results alongside the resource model, led to the project product specification being defined as below.

A process flowsheet was developed to achieve 3 dMtpa of product, with conventional crushing and screening, followed by HPGR and wet screening, two stage fine grinding and magnetic separation, reverse flotation and a final wash before filtration for storage and loadout.

Table 1-14: Product specification

Fe %	Al_2O %	SiO_2 %	P %	LOI %	S %
66.1	0.10	4.9	0.02	-2.7	0.6

1.16.4 Infrastructure

Required mine infrastructure has been designed and costed, including:

- power
- water
- airport
- accommodation
- access and plant area roads
- communications
- mine administration facilities
- tailings storage
- landfill
- wastewater management
- laboratory
- gatehouse and security, and
- explosives storage.

Where applicable, the Company has elected to develop several facilities under a build-own-operate (BOO) model funded and managed by interested third parties. Such facilities include the laboratory and power station with pricing treated as an operating cost over the term of the proposed contract.

1.16.5 Port

To facilitate export from the Port of Esperance, new infrastructure is required to unload the concentrate from rail wagons, handle and store it locally, and finally deliver it onto existing bulk product ship loading facilities at the Port.

In December 2021, Macarthur presented to SPA a design allowing for a new iron ore circuit to align with the SPA Masterplan for a multi-user iron ore facility. The design allows for integration with existing operations with minimal disruption. Further criteria were:

- 100-year design life
- rail line extension and new rail loop to remove existing port rail constraints
- a new twin-cell rotary car dumper (RCD) with unload capacity of 4500 t/h
- provision for three new storage sheds of approximately 250,000 t each
- direct unloading to ship or shed
- integration of circuit to existing iron ore and spodumene sheds, and
- land reclaim as required but minimised for environmental and capital reasons.

The proposed development excluding the concentrate storage shed is to be funded by a third-party infrastructure asset group. Under this scenario, the Company would be charged a tariff for material handled through the circuit, operated by SPA or the asset owner.

1.17 Recommendations

There are several areas that will require additional focus prior to Macarthur fully committing to the execution of the Project. These works are summarised below.

1.17.1 Geology and Mineral Resources

CSA Global recommend the following actions be completed to support the ongoing exploration and evaluation effort at the Lake Giles Magnetite Project:

- Continue to develop a deposit scale geological model incorporating lithology, mineralisation, weathering and structural features that locally control the occurrence and location of BIF host rock
- Consider domaining a zone exhibiting higher magnetite concentration, and lower SiO_2 levels, for future Mineral Resource estimates. The domain would need to exhibit sufficient strike and down dip extent to be justified for future use
- Maintain field geological procedures with respect to drill rig inspections and sampling procedures, vetting the maintenance and cleanliness of sample splitters and sample recovery
- Monitor the performance of certified reference materials (CRM) and field duplicates immediately upon receipt of assays
- Macarthur geologists to compile a QAQC report prior to future Mineral Resource estimates.
- Merge the drillhole databases containing the pre-2019 and 2019 drill data; and
- Complete additional drilling in Indicated and Inferred Mineral Resource areas to increase geological confidence of individual mineralised units.

Future exploration work would initially proceed with one phase of work, focusing on infill drilling to increase the confidence in the Mineral Resources within areas currently classified as Indicated or Inferred. An update to the Mineral Resource estimate would follow, irrespective of the impact the drill results would have on the Mineral Resource. A budget of A$730,000 is proposed for this work.

1.17.2 Mining and Mineral Reserves

Orelogy recommends the following actions before moving to implementation through to early operations:

- Variability test work program to correlate with the DTR grades within the resource model and improve estimation of blending requirements for the silica reporting through to concentrate
- Examine the cost benefit of adopting alternative rosters weighted towards an improved work-life balance to improve recruitment and retention of the workforce in Western Australia
- Explore the option of automation of mining equipment in more detail through early engagement with Mining contractors. This may have significant cost benefits both operationally and in reduction in capital for construction of accommodation at site
- Conduct blasting trials as performance data is gathered when the mine is opened up to fine-tune the blasting parameters; and
- Conduct further design work on the final pit and internal stages to improve operability and reduce waste before engaging in the Tender process for the mining services.

1.17.3 Mineral Processing

The following recommendations arise from the completed metallurgical testwork and analyses.

Further drilling should be performed, in order to produce representative composites based on the ore types in the deposit, in sufficient quantities to allow the performance of a comminution and pilot plant programme. The number of drill holes should be determined by addressing any ore types made evident in the geological modelling so that sufficient sample of each ore type to make feed for a significant flotation programme as well as for a Pilot Plant programme using a master composite. This would be approximately 5-10 tonnes of sample.

For the comminution programme CMD Consulting Pty Ltd recommend that, assuming a payback period of 7 years, at 10 Mtpa would require at least 70 samples, each sample representing 1 Mt of ore.

The plant will need to be designed to treat a highly abrasive ore.

The removal of material during dry LIMS processing is small compared to industry benchmarks, so an assessment of the benefit of the dry LIMS processing should be included during a value engineering stage.

The final size for the grinding circuit will be 80% passing 38 µm.

Further bench-scale reverse flotation work will be required to optimise reagent selection, dosing and recovery profile. A scale-up factor will be needed in sizing the flotation cells, expected to be in the range of 2 – 2.5 times the laboratory retention times. This should be vendor advised.

Further assessment of the sulphide mineralisation, in order to determine a mechanism to address desulphurisation and provide a path going forward should be performed.

Further recommendations from the CMD report include:

- Algorithms that correlate ore properties with geological data such as RQD and fracture frequency could be an economical way of defining the ore over time
- Forecast modelling is recommended to better manage the operating conditions of the circuit if and when the ore blends change; and
- Metso-Outotec will need to provide process guarantees for the Vertimills and show methods for the design and scale-up procedures.

1.17.4 Infrastructure

Macarthur will need to undertake further investigation and discussions with potential 3rd party providers for power supply, rail infrastructure and access, port infrastructure and access. Continuing engagement with these providers will ensure that the Project meets the proposed development timeframes.

Further works should also be undertaken to see if alternative fuel / power supply facilities can be utilised for the proposed infrastructure to simplify the Project and further reduction the Project’s carbon footprint.

1.17.5 Water Exploration

Engenium recommendations

Macarthur will need to undertake further investigation of water sources for the Project. To validate the potential water supply sources, field drilling and water testing will be required. All holes are to be geologically and hydro-geologically logged with water strike and flow rate data recorded during drilling. Sustainability tests will need to be undertaken along with water quality analysis to determine each of the selected areas ability to supply water at the volumes and quality required for continuous mining operations for the Project.

1.17.6 Permits and Approvals

Macarthur needs to commence desktop surveys and baseline environmental surveys as identified in Section 20 to facilitate environmental approval of the project. In accordance with the EPA Guidance notes for flora and fauna surveys, baseline studies need to be undertaken in appropriate seasons with some studies requiring multiple seasons. To ensure the Project meets the development timeframes proposed, Macarthur needs to ensure it is sufficiently resourced to commence field studies this year.

To avoid delays in final grant of approvals, tenure for outstanding project areas needs to be progressed and the development envelope clearly defined.

2 Introduction

2.1. Issuer

This Technical Report has been prepared for Macarthur Minerals Limited ("Macarthur", "the Company" or "the Issuer") by independent consultants Engenium Pty Ltd now Stantec Pty Ltd (Engenium) with contributions from CSA Global Pty Ltd (CSA Global) and Orelogy Consulting Pty Ltd (Orelogy).

Macarthur Minerals Limited is an Australian public company listed on the TSX Venture Exchange (TSX-V: MMS), OTC Markets (OTCQB: MMSDF) and the Australian Securities Exchange (ASX: MIO). The Company is incorporated in Australia and registered in Queensland. Macarthur Minerals owns the Lake Giles Project through its 100% owned subsidiary, Macarthur Iron Ore Pty Ltd.

2.2. Terms of Reference

The content of this report describes the Feasibility level studies undertaken, for mining, processing, transport logistics and marketing of the magnetite concentrate from the Moonshine and Moonshine North magnetite deposits at Macarthur's Lake Giles Iron Project.

This Technical Report discloses material changes to the Property including a maiden Mineral Reserve estimate.

The report is specific to the standards dictated by National Instrument 43-101 (NI 43-101) (30 June 2011), companion policy NI 43-101CP, and Form 43-101F1 (Standards of Disclosure for Mineral Projects). The Mineral Resource and Mineral Reserve estimates reported in this Technical Report have been prepared in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) (2014 CIM Definitions and Standards). The report is intended to enable the Issuer to reach informed decisions with respect to the Project.

The Issuer reviewed draft copies of this report for factual errors. Any changes made because of these reviews did not include alterations to the interpretations and conclusions made. Therefore, the statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this report.

2.2.1. Independence

Neither Engenium, CSA Global, Orelogy, nor the authors of this report, has any material present or contingent interest in the outcome of this report, nor do they have any pecuniary or other interest that could be reasonably regarded as being capable of affecting their independence in the preparation of this report. The report has been 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. No member or employee of Engenium, CSA Global, or Orelogy is, or is intended to be, a director, officer or other direct employee of Macarthur.

2.3. Sources of Information

This geological interpretation and Mineral Resource estimates are based primarily on the information sources listed as references in Section 27. The Mineral Resource estimates were completed by Mr David Williams of CSA Global based on the technical data provided by Macarthur and its consultants.

Capital and operating costs for mineral processing, non-process infrastructure and logistics were derived from quoted rates to Macarthur and Engenium or developed by Engenium consistent with the level of engineering detail undertaken. Engineering designs for the tailings storage facility and Esperance port land reclaim was undertaken by Stantec under the supervision of Engenium.

Capital and operating costs for mining studies were developed from first principals by Orelogy with quoted rates as applicable.

Information on current and forward product demand characteristics, product marketing and pricing were supplied by Glencore and were also derived from published research reports prepared by Wood

Mackenzie and major global iron ore producers and marketers (such as BHP's published price and market forecasts). Macarthur also engaged LFJ Consulting Pty Ltd (LFJ) to undertake an iron ore market and price analysis. All sources have been considered and utilised in to determine the product marketing and forecast pricing approach for the purposes of this Study.

2.4. Qualified Persons

The QPs have prepared or supervised the preparation of each section as presented in Table 2-1.

Table 2-1: Technical report items and responsible authors

Section	Section title	Responsible author
1	Summary	All authors
2	Introduction	All authors
3	Reliance on Other Experts	David Sourbutts
4	Property Description and Location	David Sourbutts
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	David Sourbutts
6	History	David Williams
7	Geological Setting and Mineralisation	Nikolay Karakashov
8	Deposit Types	Nikolay Karakashov
9	Exploration	Nikolay Karakashov
10	Drilling	Nikolay Karakashov
11	Sample Preparation, Analysis and Security	Nikolay Karakashov
12	Data Verification	Nikolay Karakashov
13	Mineral Processing and Metallurgical Testing	Neville Dowson
14	Mineral Resource Estimates	David Williams
15	Mineral Reserve Estimates	Stephen Craig
16	Mining Methods	Stephen Craig
17	Recovery Methods	Neville Dowson
18	Project Infrastructure	David Sourbutts
19	Market Studies and Contracts	David Sourbutts
20	Environmental Studies, Permitting and Social or Community Impact	David Sourbutts
21	Capital and Operating Costs	David Sourbutts
22	Economic Analysis	David Sourbutts
23	Adjacent Properties	David Sourbutts
24	Other Relevant Data and Information	David Sourbutts
25	Interpretations and Conclusions	All authors
26	Recommendations	All authors
27	References	All authors

The authors are QPs with the relevant experience, education and professional standing for the sections of the report for which they are responsible.

2.5. Qualified Person Property Inspection

Mr Nikolay Karakashov, Consultant Geologist, visited the property on 9–10 September 2020. Mr Karakashov was also on site in 2021 (6-19th Sep and 29th Sep-10th Oct), to assist with and observe the geotechnical drilling program at Moonshine. The authors consider Mr Karakashov's site visit current under Section 6.2 of NI 43-101. While on site, Mr Karakashov inspected the overall geology of the project including outcropping magnetite mineralisation of the Moonshine, Moonshine North, Sandalwood, Clark Hill North, Clark Hill South, and Snark deposits. Representative drill core and RC chips of mineralised intervals from the deposits were inspected. Multiple drillhole locations were visited and collar coordinates for 28 drillholes were surveyed with a handheld Garmin global positioning system (GPS) device, with an accuracy of ± 3 m on the GDA94 grid system. In all cases, the surveyed collar coordinates were confirmed.

Mr David Williams, CSA Global Principal Resource Geologist, could not complete a current site inspection due to domestic travel restrictions decreed by the Western Australian Government because of the COVID-19 pandemic. Mr Williams previously visited the Project on several occasions between 2010 and 2012, where he observed drilling and sampling procedures in progress at the time, inspected BIF outcrop, and held discussions with the Macarthur staff regarding the geology of the deposits, and potential future development of the Project.

Mr Stephen Craig, Orelogy Principal Mining Consultant, visited the property on 29 September 2022. While on site, Mr Craig inspected the terrain and vegetation cover of the proposed pits, waste dumps and ROM pad. RC chips across the site were inspected as well as representative diamond drill core for both ore and waste zones of oxide and fresh rock types.

Mr Neville Dowson, Engenium Principal Process Engineer, visited the property on 6 April 2022. While on site, Mr Dowson inspected the overall geology of the project including outcropping hematite mineralisation of the Moonshine and Moonshine North deposits.

Mr David Sourbutts, Engenium Project Director, visited the property on 6 April 2022. While on site, Mr Sourbutts inspected the overall topography, existing infrastructure and geography of the Project.

The QPs are satisfied there has been no new material scientific or technical information about the property since the last site visits by the qualified persons.

3 Reliance on Other Experts

No reliance on other experts who are not QPs was made in the preparation of this report other than outlined below.

The QPs have relied upon and disclaims responsibility for information provided by the Issuer concerning legal and environmental matters relevant to the Technical Report in a document titled "Lake Giles Iron Project – Tenure & Environment", dated 15 March 2022 authored by Dr Dean Carter, General Manager, Macarthur Minerals.

The QPs have not independently verified the legal status, ownership of the properties described in Sections 4.2 and 4.3.and rely upon the above cited document. The Property description presented in this report is not intended to represent a legal, or any other opinion as to title.

Mr Sourbutts has relied on information regarding environmental impacts, approval status and native title rights in the above cited document with respect to Sections 4.2 and 4.3.

4 Property Description and Location

4.1. Location of Property

The Lake Giles Iron Project is located approximately 450 km east-northeast from the coastal city of Perth and 175 km northwest from the historic gold mining town of Kalgoorlie-Boulder, in the state of Western Australia Figure 4-1.

Unless otherwise stated, all coordinates referenced in this report are in Geocentric Datum of Australia (GDA 94, Zone 50). The Project tenements are centred at approximately 788,000 mE and 6,687,000 mN.

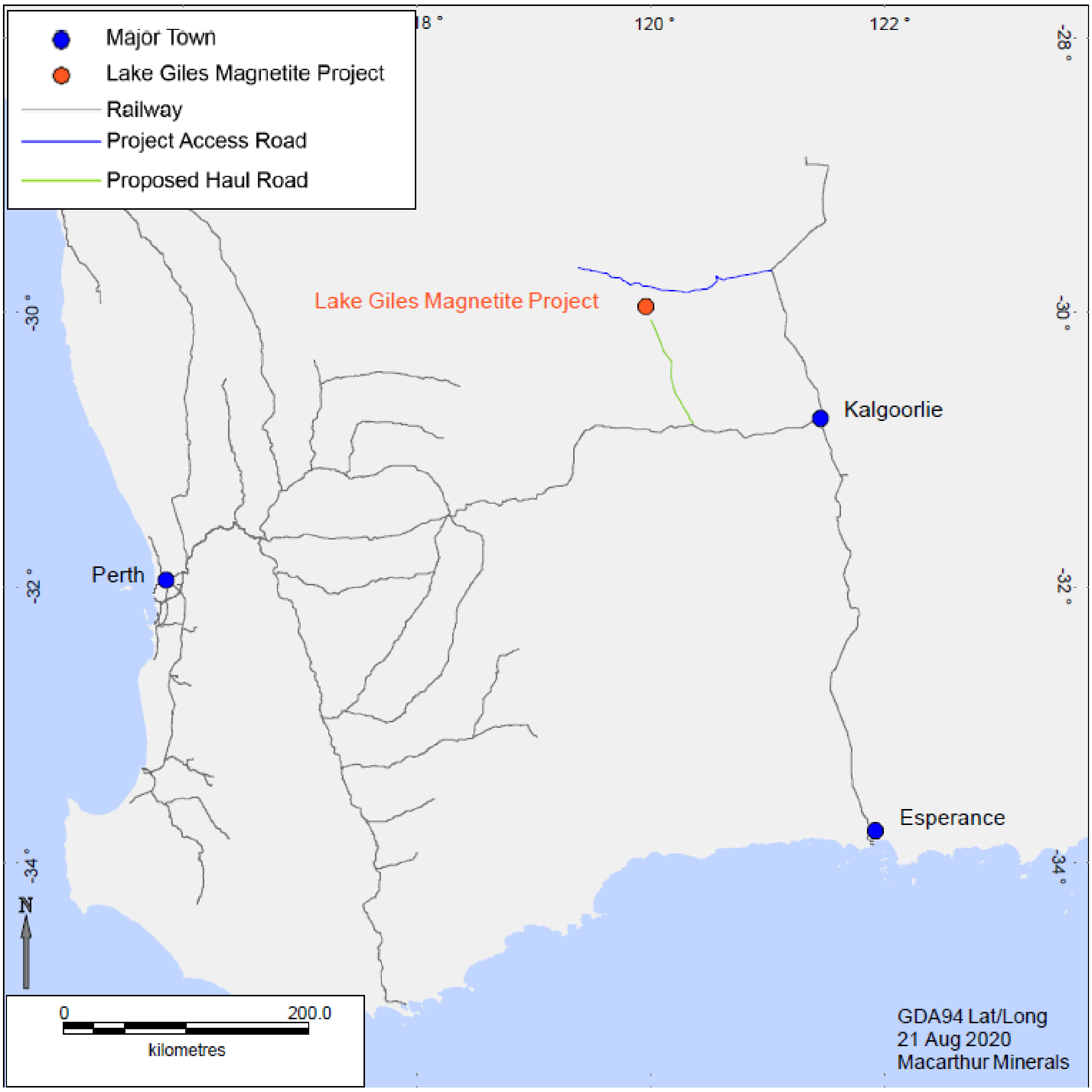


Figure 4-1: Location of the project area with local infrastructure and localities

Source: Macarthur (2020)

4.2. Details of Tenure

At present, Macarthur manages 15 granted mining leases covering a total area of approximately 6,256 Ha All tenements are 100% controlled by MIO, a 100% owned subsidiary of Macarthur, as itemised in Table 4-1 and Figure 4-2.

Mining Lease boundaries are defined by the location of corner claim pegs with approximate coordinates based on GPS readings recorded in claim documentation. They must be accurately surveyed by an Approved Surveyor after the lease is granted.

Table 4-1: MIO Tenure details and expenditure commitments as at 21 March 2022

Tenement ID	Holder	Area (ha)	Grant or (Application) Date	Expiry date	Annual expenditure Commitment (A$)
M30/0206	MIO	189	02/07/2007	01/07/2028	$18,900
M30/0207	MIO	171	02/07/2007	01/07/2028	$17,100
M30/0213	MIO	258	13/06/2011	12/06/2032	$25,800
M30/0214	MIO	260	13/06/2011	12/06/2032	$26,000
M30/0215	MIO	521	13/06/2011	12/06/2032	$52,100
M30/0216	MIO	55	13/06/2011	12/06/2032	$10,000
M30/0217	MIO	114	13/06/2011	12/06/2032	$11,400
M30/0227	MIO	504	13/06/2011	12/06/2032	$50,400
M30/0228	MIO	362	02/07/2007	01/07/2028	$36,200
M30/0229	MIO	205	02/07/2007	01/07/2028	$20,500
M30/0248	MIO	585	22/02/2012	21/02/2033	$58,500
M30/0249	MIO	1206	22/02/2012	21/02/2033	$120,600
M30/0250	MIO	102	05/03/2013	04/03/2034	$10,200
M30/0251	MIO	1246	27/11/2012	26/11/2033	$124,600
M30/0252	MIO	478	27/05/2013	26/05/2034	$47,800
E15/1775	MIO	589	24/06/20		$15,000
L15/409	MIO	97	25/06/20		NA
L16/133	MIO	923	25/06/20		NA
L30/89	MIO	23663	26/03/21		NA
L30/92	MIO	31660	26/03/21		NA

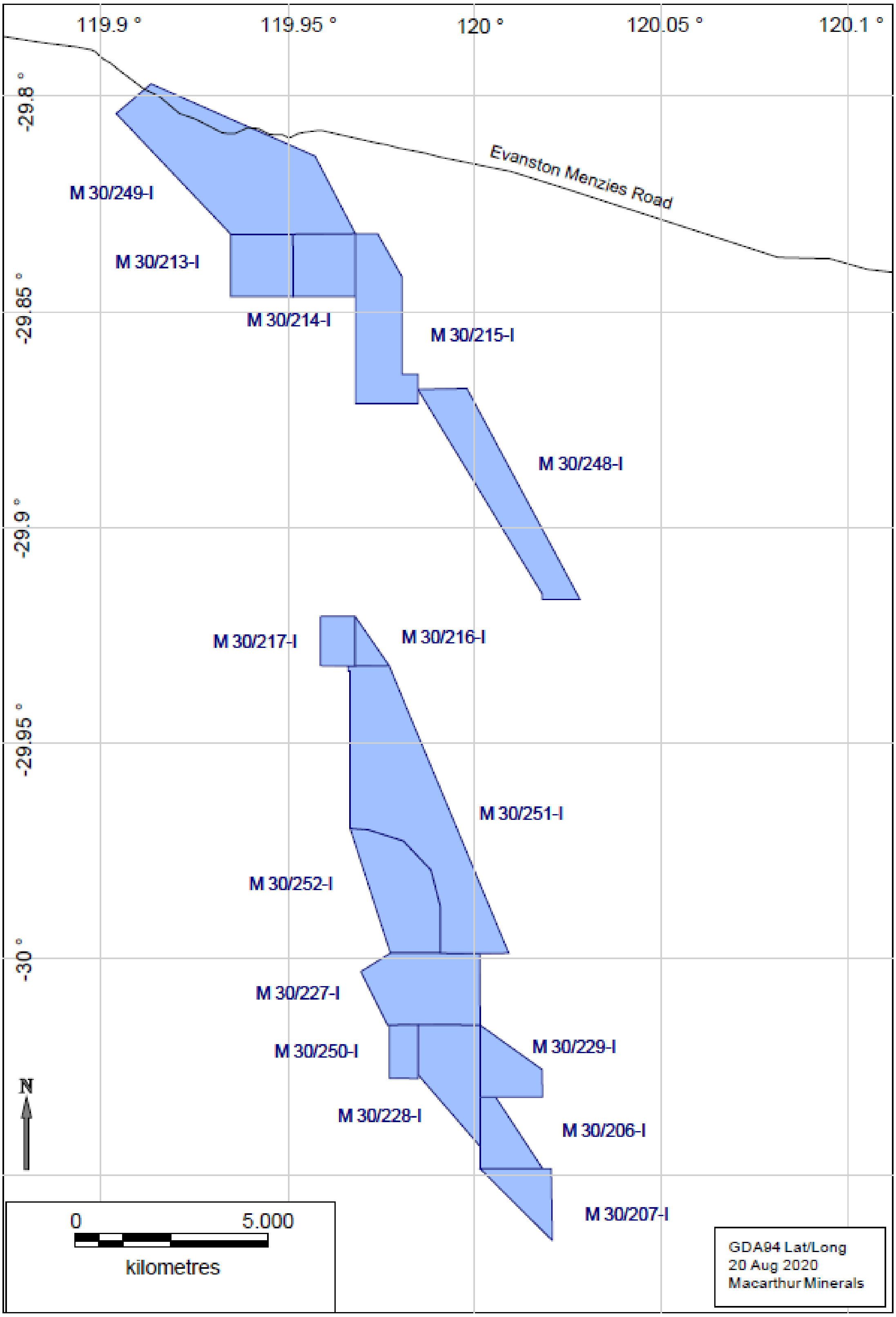


Figure 4-2: Macarthur mining lease holdings at March 2022

Source: Macarthur (2020)

4.3. Tenure Conditions and Liabilities

Macarthur's tenements occur on vacant Crown Land which is defined as Crown Land not currently being used or reserved for any future purpose. As the registered tenement manager, Macarthur has the right to access the land for the purpose of mineral exploration, subject to the conditions of tenure described in Table 4-2

The tenements are not subject to any royalty agreements or encumbrances other than described below.

There are no heritage agreements in place as the tenements were granted prior to the current native title claim. There are no other known significant risks that could affect access, title or the right to perform work on the tenements. All exploration activity is conducted according to the tenure conditions as listed below, including the requirement to obtain Program of Works (POW) approvals before any drilling is undertaken.

The project does not have any environmental liabilities from previous mining or exploration activities such as the rehabilitation of waste dumps or decommissioning of tailings storage facilities. No area of the site is registered as a contaminated site that requires remediation. Macarthur has not been fined or prosecuted under any environmental legislation or received any improvement notices for current or past exploration activities from the Western Australian DMIRS.

Table 4-2: Tenure conditions

Applicable Tenement	Condition
All Mining Leases (listed below): • M30/206 • M30/207 • M30/213 • M30/214 • M30/215 • M30/216 • M30/217 • M30/227 • M30/228 • M30/229 • M30/248 • M30/249 • M30/250 • M30/251 • M30/252	All surface holes drilled for the purpose of exploration are to be capped, filled or otherwise made safe after completion. All costeans and other disturbances to the surface of the land made as a result of exploration, including drill pads, grid lines and access tracks, being backfilled and rehabilitated to the satisfaction of the Environmental Officer, DMIRS. Backfilling and rehabilitation being required no later than six months after excavation unless otherwise approved in writing by the Environmental Officer, DMIRS. All waste materials, rubbish, plastic sample bags, abandoned equipment and temporary buildings being removed from the mining tenement prior to or at the termination of exploration program. Unless the written approval of the Environmental Officer, DMIRS is first obtained, the use of scrapers, graders, bulldozers, backhoes or other mechanised equipment for surface disturbance or the excavation of costeans is prohibited. Following approval, all topsoil being removed ahead of mining operations and separately stockpiled for replacement after backfilling and/or completion of operations. The construction and operation of the Project and measures to protect the environment being carried out generally in accordance with the POW approvals (where present). Where a difference exists between the POW approvals and the following (tenement) conditions, then the following (tenement) conditions shall prevail.
M30/249	No interference with Geodetic Survey Station NMF 395 and mining within 15 m thereof being confined to below a depth of 15 m from the natural surface.
M30/229	The development and operation of the Project being carried out in such a manner so as to create the minimum

	practicable disturbance to the existing vegetation and natural landform. All topsoil being removed ahead of all mining operations from sites such as pit areas, waste disposal areas, mineralisation stockpile areas, pipeline, haul roads and new access roads and being stockpiled for later re-spreading or immediately re-spread as rehabilitation progresses.
M30/213, M30/214, M30/215, M30/216, M30/217 and M30/227	Portions of these licences are overlain by the Mount Manning Nature Reserve. This reserve was granted in April 2000 and is identified by Western Australian Government reference number 36208. The iron mineralisation of the Project does not encroach on the nature reserve. Consent to explore on DEC – Managed Lands Conservation of Flora and Fauna Reserve 36208 granted subject to the following conditions: • Prior to lodgement of a POW, the lessee preparing a Conservation Management Plan (CMP) to address the conservation impacts of the proposed activities and submitting the CMP to the relevant Regional Manager of the Department of Environment and Conservation (DEC). This CMP shall be prepared pursuant to DEC-prepared "Guidelines for Conservation Management Plans Relating to Mineral Exploration on Lands Managed by the Department of Environment and Conservation" to meet the requirements of the Minister for Environment for acceptable impacts to conservation estate. A copy of the CMP and of DEC's decision on its acceptability under the guidelines is to accompany the lodgement of the POW application with the DMIRS. • At least five working days prior to accessing the reserve or proposed reserve area, unless otherwise agreed with the relevant Regional Manager of the Department of the Environment and Conservation (DEC-R), the holder providing the DEC-R with an itinerary and program of the locations of operations on the lease area and informed at least five days in advance of any changes to that itinerary. All activities and movements shall comply with reasonable access and travel requirements of the DEC-R regarding seasonal/ground conditions • The licensee submitting to the Director of Environment, DMIRS, and to the relevant Regional Manager, Department of the Environment and Conservation (DEC-R), a project completion report outlining the project operations and rehabilitation work undertaken in the program. This report is to be submitted within six months of completion of the exploration activities.
M30/213, M30/214, M30/215, M30/216, M30/217 and M30/227	All Mining Proposals submitted for the commencement, alteration or expansion of operations within the tenement boundary are to contain information that demonstrates the proponent has genuinely engaged with the DEC on the Mining Proposal. The level of engagement will be to the satisfaction of the Director Environment, DMIRS.
M30/213	Rights being reserved to persons authorised by the Chief Executive Officer of the DEC to enter the Lease and carry out

	land management operations and other duties and exercise such powers as may be necessary or expedient for the administration of the Conservation and Land Management Act 1984 and Regulations, the Wildlife Conservation Act 1950 and Regulations, the Bush Fires Act 1954 and Regulations and the Emergency Management Act 2005 and Regulations.
M30/207	No interference with Geodetic Survey Station SSM - Kalgoorlie 93 and mining within 15 m thereof being confined to below a depth of 15 m from the natural surface.
M30/227	No interference with Geodetic Survey Station SSM-KALGOORLIE 138 and mining within 15 m thereof being confined to below a depth of 15 m from the natural surface.
All Mining Leases (listed below): • M30/206 • M30/207 • M30/213 • M30/214 • M30/215 • M30/216 • M30/217 • M30/227 • M30/228 • M30/229 • M30/248 • M30/249 • M30/250 • M30/251 • M30/252	Mining Leases must be surveyed by an Approved Surveyor upon grant of the tenement or approval of a Mining Proposal. The lessee submitting a plan of proposed operations and measures to safeguard the environment to the Director, Environment, DMIRS for his assessment and written approval prior to commencing any developmental or productive mining or construction activity. Mining on any road, road verge or road reserve being confined to below a depth of 15 m from the natural surface.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1. Topography, Elevation and Vegetation

The topography of the Project area is comprised of low ridges associated with the BIF units, striking in a general northwest - southeast direction, that rise up from the surrounding sandy plains. The range in elevation is approximately 120m with the highest point at approximately 520 mRL. Adjacent to the low ridges are flat to gently undulating areas of sheetwash and soil covered areas.

The vegetation of the Project area is dominated by mulga scrub with local patches of low to medium eucalypt woodland and areas of salt tolerant shrub and spinifex.

5.2. Access to Property

The Project can be accessed from Kalgoorlie-Boulder via the sealed Menzies Highway north for 130 km, then west from the town of Menzies for 120 km along the unsealed graded Evanston-Menzies Road (refer **Figure 4-1).**

Kalgoorlie-Boulder is serviced by daily commercial flights from Perth.

Access within the Project area is by a number of tracks cleared by previous explorers, and more recently by Macarthur. These tracks may become impassable after heavy rain.

5.3. Climate

The climate at the Project is characterised as semi-arid. The Diemals weather station, located 65 km west of the Project at latitude 29.67°S and longitude 119.30°E, was operated by the Australian Bureau of Meteorology between 1970 and 1994 (Australian Bureau of Meteorology, 2011). Diemals recorded a mean annual rainfall of 275.7 mm with rainfall mostly in the winter months. The temperature averages over 40°C for 15 days in the summer months (from November to March) while in the winter months (from June to August) the temperature averages a minimum range from 3.9°C to 5.0°C. See Figure 5-1 for more details.

The climate at the project area allows an operating season covering the full length of the year. In the Kalgoorlie region, mining and exploration activities are conducted throughout the year, with infrequent generally short disruptions during and after periods of heavy rain.

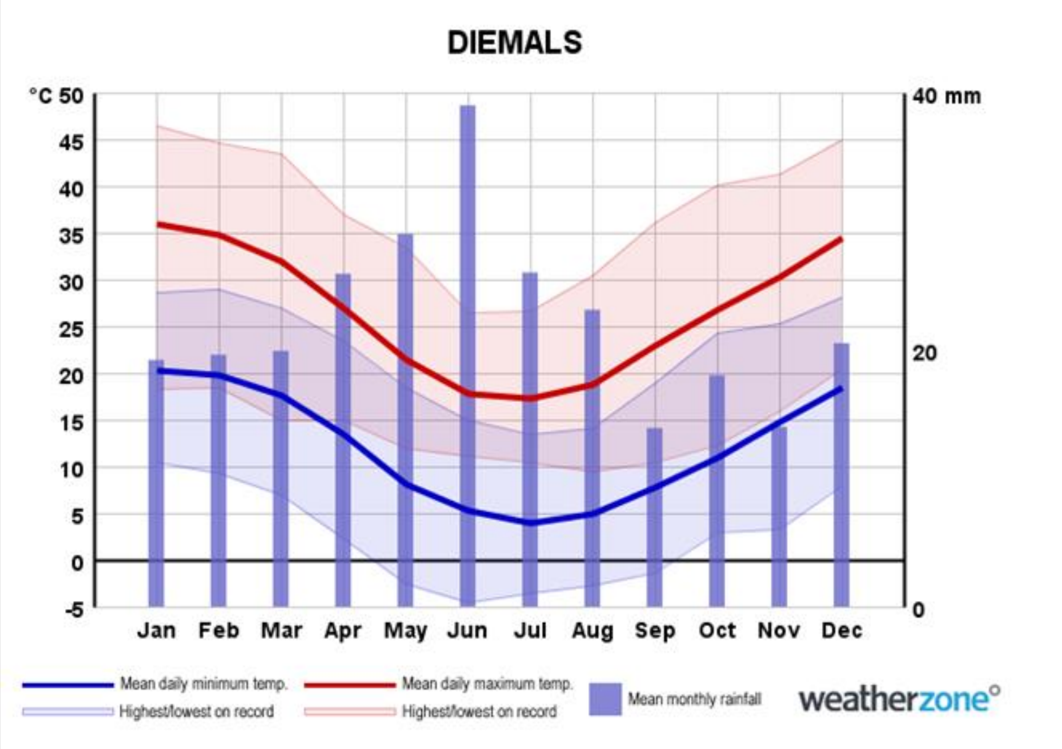


Figure 5-1: Average temperature ranges and rainfall on a monthly basis for Diemals weather station (Weatherzone, 2011)

5.4. Infrastructure

The Project is serviced from the city of Kalgoorlie-Boulder, with a population of 30,000 people (Australian Bureau of Statistics, 2016), which provides services and mining personnel to a large number of operating mines and exploration properties in the region.

Some limited facilities are available in Menzies including fuel, accommodation, and meals. A railway line passes through Menzies, and road freight lines deliver to the town.

The Project site itself is remote with no existing infrastructure other than unsealed roads and an exploration camp.

Power generation for the exploration camp is by diesel powered generators with potable water trucked in from Kalgoorlie.

Network power is available in Kalgoorlie via the existing West Kalgoorlie substation approximately 130 km southeast of the Project. An overhead powerline would need to be constructed to utilise network power at the project. The Kalgoorlie gas line is also located 130 km east of the Project.

Potable quality water is available from the water pipeline owned by the Water Corporation that is located approximately 130 km south of the Project. Saline groundwater supply is likely to occur in the region, such as the Scorpion Paleovalley located approximately 30 km east of the Project (Rockwater, 2020).

The Eastern Goldfields Railway runs between Perth and Kalgoorlie and then south to the Esperance Port which has facilities for iron mineralisation handling, storage and export. The Eastern Goldfields Railway is located approximately 90 km south of the Project with total rail haul to the port approximately 500 km.

Macarthur has commenced discussions with the railway owner, Arc Infrastructure regarding capacity on the rail line. Future economic studies will be required to investigate the Project's requirements for a haul road to be constructed to access the rail line.

At this time, it appears that Macarthur holds sufficient mining leases necessary for proposed exploration activities and potential future mining operations (including potential tailings storage areas, potential waste disposal areas, and potential processing plant sites) should a mineral deposit be delineated at the Property for which any future mining studies may provide positive economic results. The adequacy of the Property area for required mining and processing infrastructure will be further assessed as engineering studies advance.

6 History

6.1. Property Ownership

Since the late 1960's several exploration companies have held the exploration rights to the project tenements. There have been three main phases of exploration; nickel exploration from 1968 to 1972, gold exploration from 1993 to 2004 and more recently iron exploration. The following summary has been derived from Revell (2006), Farmer (1997a, 1998a, 1998c) and Busbridge (1998a, 1998b).

Between 1968 and 1972 the area was explored primarily for nickel sulphide mineralisation by Amax Exploration (Australia) Inc, Consolidated Goldfields Australia Limited, Geotechnics Pty. Ltd., on behalf of Welcome Stranger Mining Company Limited, Kia Ora Gold Corp. NL, Delta Minerals NL and Le Nickel (Australia) Exploration Pty Ltd.

Between 1972 and 1993 there are no records of any significant exploration activity.

From 1993 to 1998, the region was explored primarily for gold by several companies, generally operating in joint ventures:

- In May 1993, Battle Mountain Australia Incorporated (Battle Mountain) was granted Exploration License E30/93 which partially overlaps with the southern portion of the area now covered by Macarthur's currently granted mining lease M30/249.
- In August 1993, Aztec Mining Company Limited (Aztec), a subsidiary of Normandy Exploration Limited (Normandy) was granted Exploration License E30/100 covering western parts of the current tenements, and in December 1993 Aztec were granted E30/99 which encompasses the area now covered by M30/213-217, 251, 252.
- In 1995-1996, Noble Resources NL (Noble) formed a Joint Venture with Battle Mountain to explore E30/93, with Noble managing exploration activities. Noble's interest in the joint venture was subsequently transferred to Barclay Holdings Ltd, a wholly owned subsidiary of Titan Resources NL.
- Titan withdrew from the joint venture in 1998, and Battle Mountain surrendered the tenement in 1998.
- In September 1994, Evanston Mines NL formed the Dodanea joint venture with Aztec to explore E30/99 and E30/100.
- Following Evanston's unsuccessful float, Evanston's share of the joint venture passed to Noble Resources, and subsequently after an asset swap, on to sister company Titan Resources in February 1997; and
- In June 1998, Titan withdrew from the joint venture, and in December 1998 Normandy surrendered the tenements.

From late 1998 to 2003, the area was consolidated into the "Lake Giles Magnetite Project" by Mr Troy Dalla-Costa who was granted a number of tenements covering the area. In 2003, the tenements were purchased from Mr Dalla-Costa by Internickel Australia Pty Ltd (Internickel).

In early 2004, Internickel was purchased by Adex Holdings Limited (Adex). Macarthur purchased Internickel and the Project assets from Adex. In late 2005, Macarthur's wholly owned subsidiary Internickel was renamed to Macarthur Iron Ore Pty Ltd in 2010.

6.2. Project Results – Previous Owners

6.2.1. Nickel Exploration (1968 to 1972)

The 1968–1972 phase of nickel-focused exploration is reported by Ward (1970a, 1970b, 1970c) and Ward and Pontifiex (1970).

Exploration undertaken during this period included grid establishment, geological mapping, rock chip sampling, magnetic, electromagnetic and induced polarisation geophysical surveying, and petrographic analysis of rock samples.

Geotechnics Pty Ltd was the only company to drill in the area during this period. Table 6-1 summarises the drilling completed by Geotechnics Pty Ltd; however, the grid that Geotechnics Pty Ltd used has not been re-established and the exact location of the drillholes is unknown.

Table 6-1: Summary of drilling 1968 to 1972 (modified from Ward 1970a, 1970b, 1970c)

Type	No. of drillholes	No. of metres	Max. depth (m)
Diamond core	7	523	127
Open-hole percussion	15	658	60
Total	**22**	**1,181**	

It is unclear where these drillholes lie in relation to the areas of current interest for iron mineralisation. Rock chip sampling conducted by Geotechnics Pty Ltd during this phase of exploration returned assays from samples of outcropping BIF with iron assay results of 36.1% to 63.5% (Cooper, 2007). Although these results provided an indication of the Project's exploration potential they were not followed up, and no exploration specifically targeting iron mineralisation was conducted until Internickel commenced exploring the tenements in 2000.

6.2.2. Gold Exploration (1993 to 1998)

In May 1993, Battle Mountain was granted the tenement E30/93 that partly overlies the tenement M30/249, which is part of the Lake Giles area (Famer 1997a, 1998a, 1998c). Battle Mountain established a grid over E30/93 from which Macarthur collected 37 rock chip samples and completed a 50 m by 500 m soil sample program, which Macarthur subsequently in filled to a 50 m by 100 m spacing for a total of 1,175 samples. This soil sample program identified several gold anomalous zones with maximum grades of 3–12 ppb Au (Anon 1994).

In August 1993, Aztec was granted the E30/100 lease which is immediately west of the current Project tenements, and in December 1993 Aztec was granted tenement E30/99 (now covered by Macarthur tenements M30/213-217). Aztec collected 715 soil samples, 31 stream sediment samples and 901 soil auger samples with identified several anonymous gold zones which peaked at 53 ppb. Aztec drilled 80 rotary air blast (RAB) holes (Table 6-2) to test the anomalous gold zones, which returned weak mineralisation, with the best result being from drillhole DON06 with 25 m at 0.4 g/t (Smith and Govey, 1995; Busbridge 1998b).

Battle Mountain drilled 41 RAB drillholes Table 6-2 in 1994–1995, targeting the anomalies identified in the soil sampling. These anomalies were named Soapbox and Enfield prospects in tenement E30/93. The best result from the RAB drillholes was from DOP8 for 4 m at 0.4 g/t at the Soapbox prospect (Anon 1995).

In 1995, Noble formed a joint venture with Battle Mountain to explore E30/93; however, Noble's interest was transferred to Barclay Holdings Limited, a wholly owned subsidiary of Titan, in February 1997.

Titan commissioned Telsa Airborne Geophysics in 1997 to complete an airborne geophysics survey of tenements E30/93, E30/99 and E30/100. The airborne survey included magnetics and radiometric surveys and was flown at a height of 50 m on 100 m line spacing. In the same year, Titan completed a 537-auger soil sample program over tenement E30/93 (Famer 1997a, 1997b 1998a).

In early 1998, Titan collected 311 soil samples on a 50 m by 80 m grid within tenement E30/93 but failed to define any anonymous gold zones (Busbridge 1998a). In mid-1998, Titan commissioned G&B Drilling to undertake a vacuum drilling program on tenement E30/100.

The drillholes went down to a maximum depth of 1.5 m and a total of 1,275 samples were collected on a drill spacing of 100 m by 400 m. In December 1998, Titan withdrew from the joint venture and the tenement was surrendered (Busbridge, 1998a).

Table 6-2: Summary of the gold exploration drilling from 1993 to 1998 (modified from Smith and Govey 1995; Busbridge 1998b; Anon 1995)

Company	Type	Tenement	No. of drillholes	No. of metres
Aztec	RAB	E30/99, E30/100	80	3,442
Battle Mountain	RAB	E30/93	41	1,897
Total			**121**	**5,339**

6.2.3. Iron Exploration – Internickel (2001 to 2005)

From late 1998 to 2003, Mr Troy Dalla-Costa was granted a number of tenements in the Lake Giles area which were to become the foundation for the MIO tenement holding. Mr Dalla-Costa consolidated his holdings in the name of Internickel.

Internickel undertook detailed evaluation of all the historical data. In early 2004, Adex purchased Internickel from Mr Dalla-Costa and then Adex changed its name to Internickel. Macarthur purchased Internickel in late 2005.

The following exploration history is summarised from Fox (2001, 2002, 2003) and Cooper (2003, 2004, 2005, 2006). Internickel's initial exploration effort targeted gold and nickel. Mapping and sampling were undertaken by Keith Fox, resulting in the generation of a number of gold and nickel targets (Fox, 2003). Fox estimated that more than 100 km strike length of komatiitic ultramafic sequence prospective for nickel sulphides existed on the tenements.

In December 2003, following the observation of fine gold in panned soils, a program of metal detecting was completed in the area of gold-in-soil anomaly G14 (Fox, 2003). Two costeans were excavated and metal detecting within and adjacent to them resulted in recovery of a single large 26-ounce (about 0.8 kg) nugget together with a number of small nuggets between 1 g and 12 g in weight. The anomalous gold geochemistry is associated with zones of quartz veining. The orientation and dip directions of these zones are unknown.

In April 2004, GPX Airborne Pty Ltd undertook a helicopter Hoistem electromagnetic survey over the central part of the Lake Giles Iron Project (Figure 6-1). This area was known to be mainly covered by thin (<2 m) soils. Data were collected along east-west flight lines spaced 200 m apart and the total survey comprised 950-line km. Interpretation of the data indicated the presence of a large number of electromagnetic anomalies.

By 2004, iron mineralisation was also recognised as a significant target in the Project area. In early 2005, a scout surface outcrop sampling program of 29 BIFs was completed. All samples were analysed for iron, as well as for a large number of other elements. Seven samples were found to contain more than 50% Fe and two contained more than 60% Fe. Subsequently, applications were submitted (and granted) for the inclusion of iron mineralisation in the commodities listed for all the tenements.

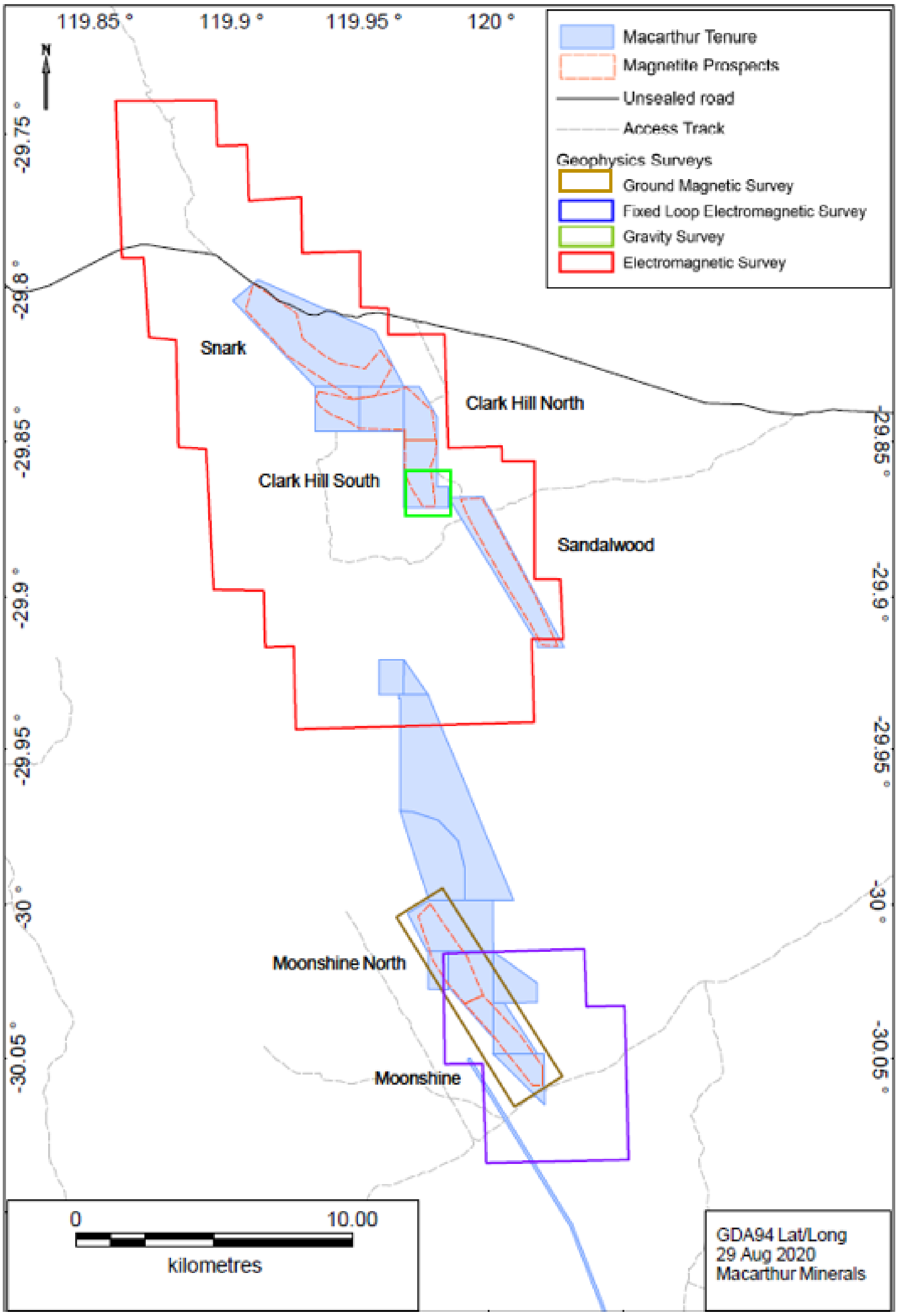


Figure 6-1: Location of exploration activity by Internickel

Source Macarthur (2020)

6.3. Previous Mineral Resource Estimates

Between 2007 and 2011, Macarthur completed Mineral Resource estimates for the magnetite deposits which were classified as Inferred and reported in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (11 December 2005), disclosed in accordance with NI 43-101 (30 December 2005), and filed on SEDAR.

The first Mineral Resource estimates were completed by Hellman and Schofield Pty Ltd (H&S) in 2007 for the Snark, Clark Hill North and Clark Hill South deposits (Abbot and van der Heyden, 2007). In 2009, H&S updated the Clark Hill North estimate and provided the first Mineral Resource estimate for the Sandalwood deposit (Abbot and van der Heyden, 2009a). In 2009, a Mineral Resource estimate was then completed for the Moonshine deposit and reported in addition to the previous estimates (Abbot and van der Heyden, 2009b). In 2009, CSA Global updated the Mineral Resource estimates for the Clark Hill North, Sandalwood and Moonshine deposits based on additional RC drilling (Allen, 2009), followed by updates to the Snark and Clark Hill South Mineral Resources in February 2010 (Macarthur, 2010). The most recent Mineral Resource estimate for the Moonshine deposit was reported by Snowden in 2011 (Fieldgate et al., 2011). The history of the Inferred Mineral Resource estimates for the Lake Giles Iron Project is shown in **Table** 6-3.

All the previous Mineral Resource estimates were prepared using RC and/or diamond drillhole data available at the time. Geological interpretations were digitised in cross section using industry standard modelling software available at the time, with 3D wireframes created to domain the magnetite mineralisation. Block models were constructed, and grades were interpolated into the block model using ordinary kriging. Where DTR results were available, concentrate grades were interpolated along with head grades. Bulk density values were assigned to the block model. The Mineral Resources were classified in accordance with the CIM Definition Standards.

The Issuer is not treating the previous Mineral Resource estimates as current Mineral Resources. These previous Mineral Resource estimates are presented for historical information and context only. Current Mineral Resource estimates are presented in Section 14 of this report.

Table 6-3: Previous inferred mineral resource estimates – Lake Giles Magnetite Project

Deposit	H&S						CSA Global		Snowden	
	2007		2008		2009		2009–2010		2011	
	Mt	Head Fe %	Mt	Head Fe %	Mt	Head Fe %	Mt	Head Fe %	Mt	Head Fe %
Snark	26.3	27.5	26.3	27.5	26.3	27.5	75	27.7	-	-
Clark Hill North	7.7	32.5	37.1	26.0	37.1	26.0	130.0	25.8	-	-
Clark Hill South	48.5	21.9	48.5	21.9	48.5	21.9	66	30.3	-	-
Sandalwood	-	-	84.7	28.3	84.7	28.3	335.0	31.1	-	-
Moonshine	-	-	-	-	144.1	25.9	510.9	27.8	710.5	30.2

6.4. Previous Mining

No mining is known to have been undertaken in the project area or anywhere on Macarthur's tenements to date.

7. Geological Setting and Mineralisation

7.1. Regional Geology

Macarthur's tenements cover a portion of the Yerilgee Greenstone Belt which is over 80 km in length and up to 10 km wide and lies within the Southern Cross Province of the Yilgarn Craton. The Yilgarn Craton consists of multiple lenticular greenstone belts surrounded by variably foliated gneissic granitoids.

The greenstone belts consist of metamorphosed ultramafic, mafic and sediments, including BIF which are Archean in age and are commonly intruded by mafic, intermediate and granitic rocks.

The greenstone belts are generally metamorphosed to mid greenschist facies towards the central parts of the belt and lower amphibolite facies on the edges of the belt where they are in contact with the granitoids.

The greenstone belts are highly deformed, faulted and folded. Four deformation events (Svensson, 2012) are recognised regionally throughout the Yilgarn Craton:

- D1 – Movement along the south-north direction
- D2 and D3 – Shortening and shear movements in the east-northeast to west-southwest compression direction; and
- D4 – Lateral extension of the greenstone belt in a north-northwest and south-southeast direction.

Figure 7-1 shows the regional geology of the Macarthur tenement area and its surroundings, derived from Geological Survey of Western Australia (GSWA) (2020).

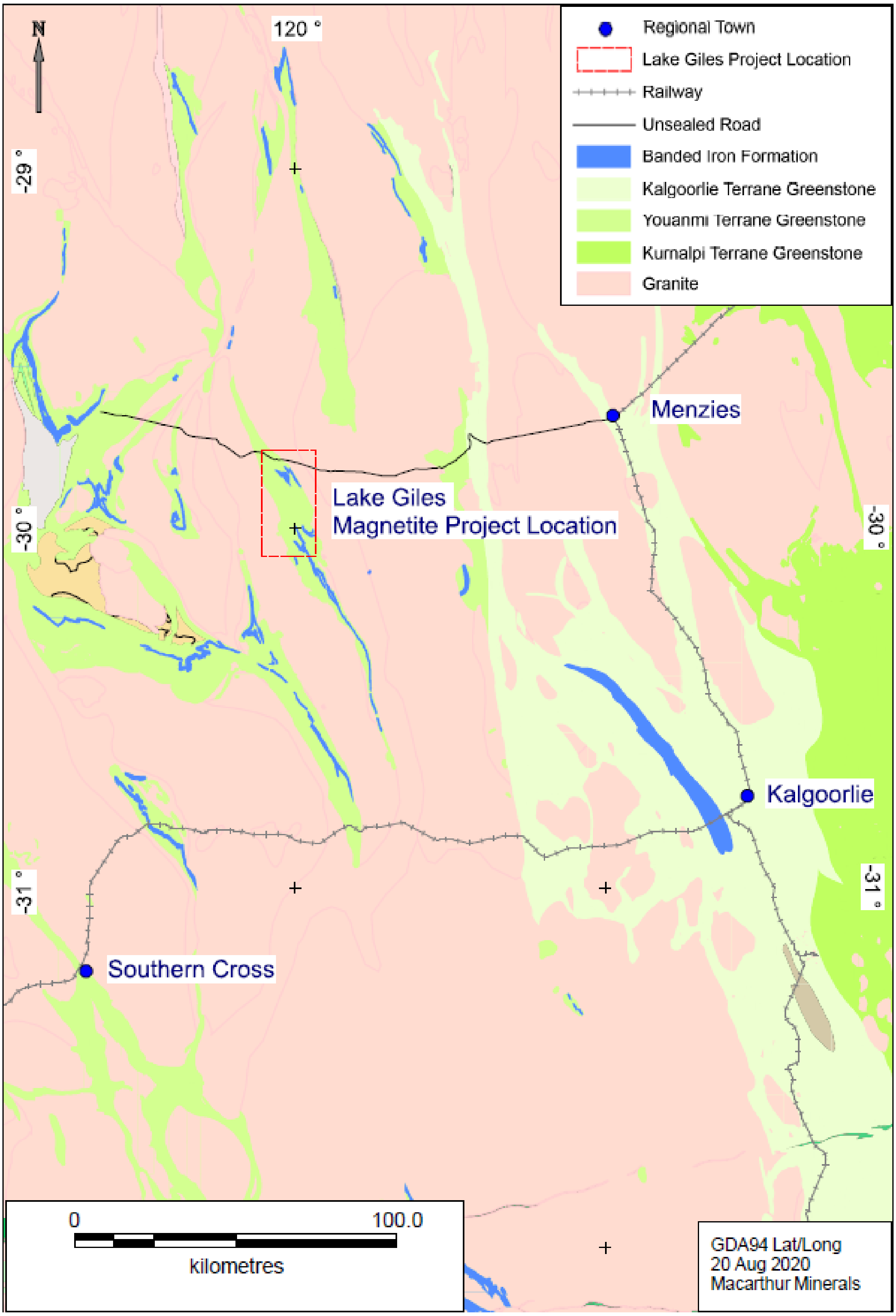


Figure 7-1: Project area with regional interpreted geology and infrastructure

Source: GSWA (2020)

7.2. Local Geology

The parts of the north-northwest trending Yerilgee greenstone belt covered by the Project tenements comprise a layered succession of Archean rocks. At the interpreted base of the succession is a sequence of high-magnesium basalt flows more than 1 km thick overlain by komatiitic ultramafic volcanic rocks with narrow interflow BIFs and in some cases, other sedimentary rocks (Svensson, 2012). Further high-magnesium basalt lavas with occasional interflow BIFs overlain, possibly unconformably, by sedimentary rocks (cherty, silicified, pyritic and graphitic) are interpreted to form the top of this sequence. In places, gabbroic sills interpreted to be co-magmatic with the upper high-magnesium basalts, have been intruded into the lower mafic and ultramafic lavas. The elongated lens shaped Yerilgee belt is bounded by major north-northwest trending fault/shear zones.

The Archaean sequence has been intensely folded. At least five possibly sinistral fault zones of similar but slightly more north-westerly trend are interpreted within the widest part of the belt and are believed to successively repeat the layered succession. Two northerly trending sinistral faults obliquely crosscut the belt in this area.

A number of large synclinal fold structures have been identified. These appear to be located adjacent to the eastern margins of the fault blocks. These folds have north-westerly and north-north-westerly trending axes and were mapped in detail (Greenfield, 2001) show plunges at 30–60° in the same direction. In general, the fold axes are steeply dipping. The folding appears to have been contemporaneous with faulting. In plan, the movement on the fault planes was sinistral but in a true sense is believed also to have been reverse faulting with the direction of movement on the western down-throw sides of the fault planes being inclined at 30–60° towards the east-northeast. The synclines and anticlines are considered to be drag fold structures.

The most recent notable tectonic event was approximately 2.6 billion years ago and appears to have dilated the north-northwest trending shear zones, generating north-northeast trending and conjugate northeast to easterly trending structures. These brittle fractures have in many places been intruded by granitic dykes or quartz veins. The Project tenements cover about 60 km of the greenstone belts strike length but because of fault repeats, they are estimated to cover more than 150 km of BIF sequence strike length.

7.2.1. Property Geology and Mineralisation

Figure 7-2 shows the locations of the main prospect areas of the Project superimposed on the local geology.

The iron ore mineralisation consists of secondary pisolite mineralization, primary magnetite mineralization associated with un-oxidized BIF and ultramafic rocks, and goethite-hematite mineralization associated with oxidized BIF.

The hematite/goethite units exist largely as a supergene product. Weathering has resulted in the leaching of the majority of the silica from the BIFs, thus producing a rock rich in iron and low in silica. These enriched bands vary from 1 to 30 metres in true thickness and are largely steeply dipping by 70º to 90º.

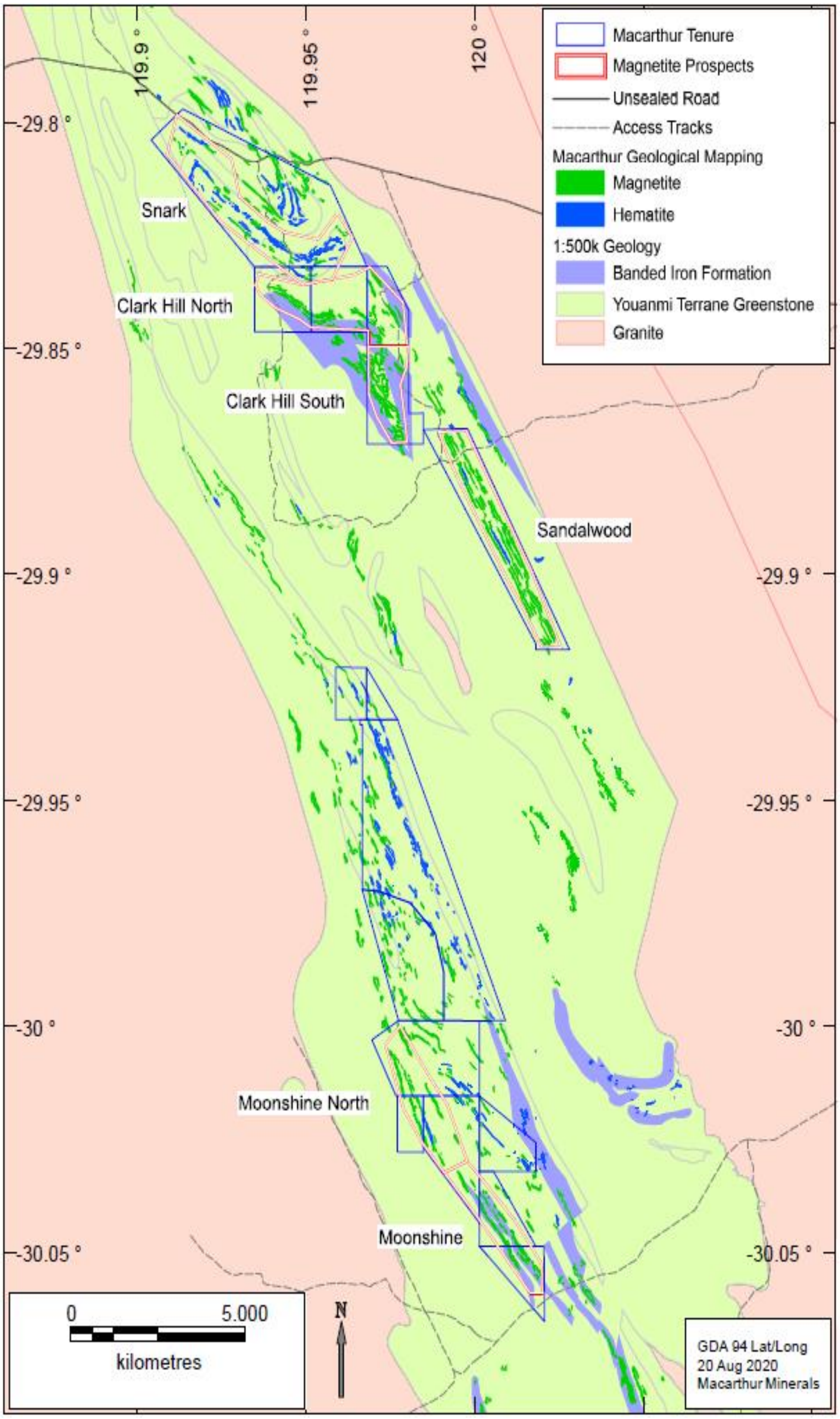


Figure 7-2: Interpreted geology of the Lake Giles Iron Project

Source: GSWA (2020)

The magnetite mineralisation is associated with primary magnetite hosted by BIF. The multiple BIF units steeply dip 75–85° to the west and strike approximately 320° and 335° respectively. The units have an average thickness of 15 m, over a strike length of 17 km. Examples of outcropping BIF are presented in Figure 7-4 and Figure 7-3, which show the distinctive laminar style of lithology present in BIFs. The geological hammer has been placed to indicate scale.

Note: that the width of outcrop does not necessarily equate to the width of the BIF unit below surface, with erosion often delaminating the exposed rock resulting in a thinner width of host rock, compared to the non-eroded equivalent rock unit down dip and below surface.

Figure 7-3: Outcrop of BIF containing magnetite mineralisation, near LGRC_0084 Sandalwood

Source: Macarthur (2020)

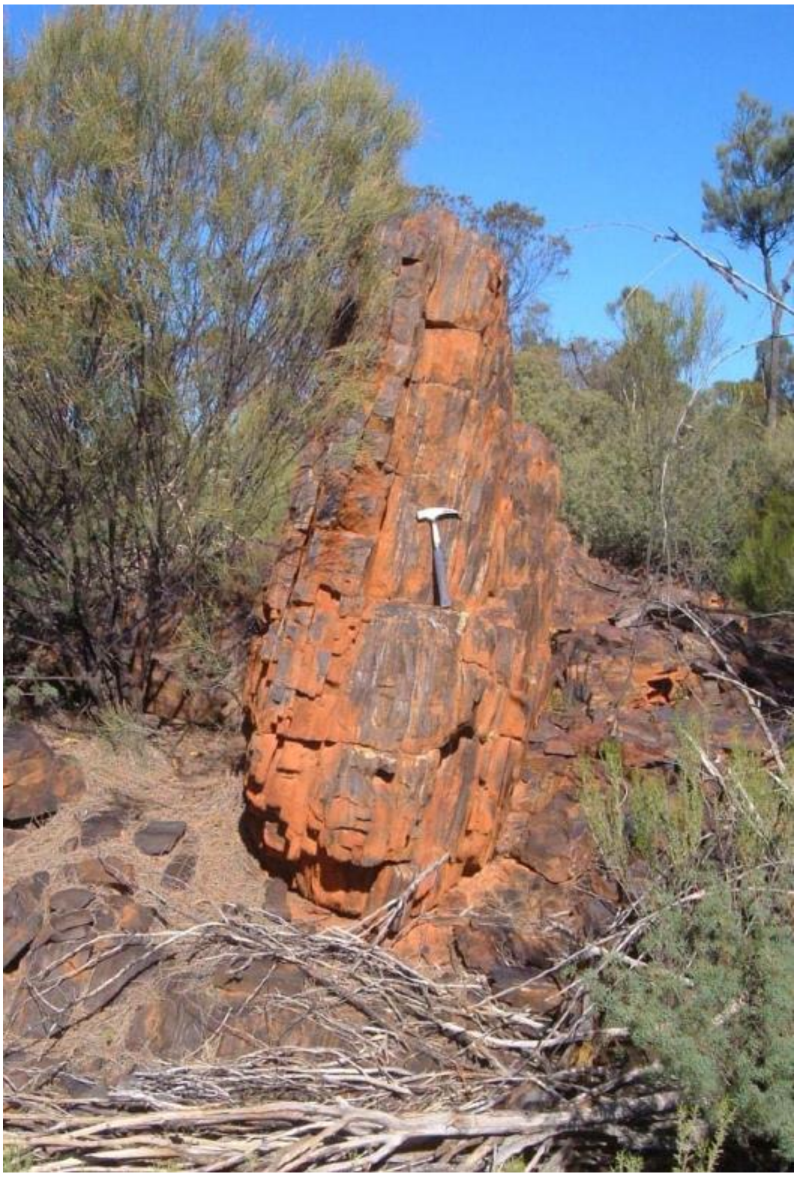


Figure 7-4: Outcrop of BIF containing magnetite mineralisation, Moonshine

Source: Macarthur (2020)

A number of folds with a northwest plunge have been identified. Further work towards interpreting the structural geology of the Project is ongoing.

The outcropping geology of the project area is comprised of a combination of unaltered silica rich BIFs and altered, enriched haematite/goethite BIFs. Weathering has resulted in the leaching of the majority of the silica from the BIFs, thus producing a rock rich in iron and low in silica, near surface. Below the depth of oxidation (generally between 45 m and 90 m from surface), the BIF units are comprised almost entirely of ferrous/ferric Fe(II,III) iron, silica and small amounts of alumina with occasional incipient iron sulphides (predominantly pyrite).

The iron grades are generally normally distributed, as opposed to log-normally for the altered haematite/goethite BIF, with grades consistently between 20% Fe and 40% Fe. Macarthur believes the majority of the underlying BIF units have experienced minimal metamorphism beyond their original formation. A notable exception to this is a pocket of high-grade magnetite mineralisation, up to 15 m true thickness, continuous along strike for >200 m, and >60% Fe, located in the Moonshine North deposit. This pocket of high-grade magnetite mineralisation is interpreted to be the result of structural and geothermal alteration of the primary BIF fabric.

The mapped outcrops range from locally dark, rich and dense mineralised BIF to porous and lateritic weathered BIF with locally enriched layers. An example of Macarthur's outcrop mapping is presented in Figure 7-5.

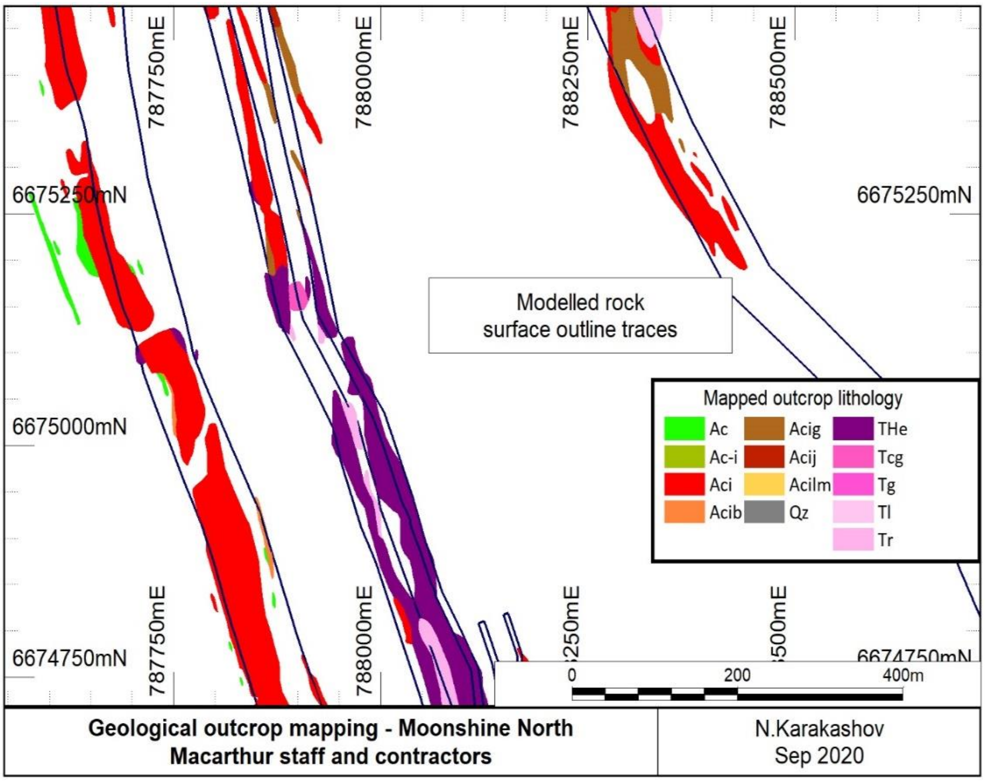


Figure 7-5: Example section of Macarthur's outcrop mapping style at Moonshine

Source: Karakshov (2020)

In RC chips, the mineralised material is dusty and red-brown to purple, and generally very fine grained.

The local high-magnesium basalts and ultramafics do not have significant outcrops due to strong weathering, especially proximal to the BIF ridges, where Macarthur has concentrated its mapping and interpretation efforts. Although some observations of ultramafic textures, such as spinifex and possible cumulate have been described, no petrological or geochemical analyses have been performed on samples from within the Project area.

Logged komatiite and ultramafic units are typically thin (<10 m true thickness) and strongly weathered near the surface and are only identifiable at depth through drilling. The ultramafics are usually found proximal to the hanging wall of the BIF units.

Serpentinised high-magnesium basalts form the bulk of the geology at Lake Giles, forming thick, continuous, fine to medium grained granular units, occasionally cut by minor quartz veins and hosting sulphidic shales, locally including several metres of massive iron sulphides. Mafic intersections of interest have been occasionally investigated for gold mineralisation, but no specific targeting for gold has been recorded.

Local faults are mostly interpreted from surface outcrop mapping aided by geophysics (particularly aerial magnetic anomalies) and are rarely observed within drillholes. This is most likely due to majority of drilling targeting the main section of undeformed tabular BIF ridges. Most of the interpreted local faults tend to be sub vertical shear structures, truncating or occasionally displacing BIF bodies.

Structural deformation within the main BIF packages is generally weak, forming gentle kink banding and box folding, although some sections are interpreted as showing intense recumbent folding with sub-vertical axial planes, such as the southern edge of Moonshine. The larger BIF bodies at Moonshine and Moonshine North have relatively consistent thickness and dip to depths of over 250 m from surface as tested by a number of drillholes, increasing confidence that the remainder of the BIF ridges at Moonshine behave in a similar way and are not truncated at depth by synclines or other structural mechanisms. Figure 7-6 shows the depth of drilling through magnetite mineralisation, with BIF units shown in grey and magnetite mineralisation domains within the BIFs in red.

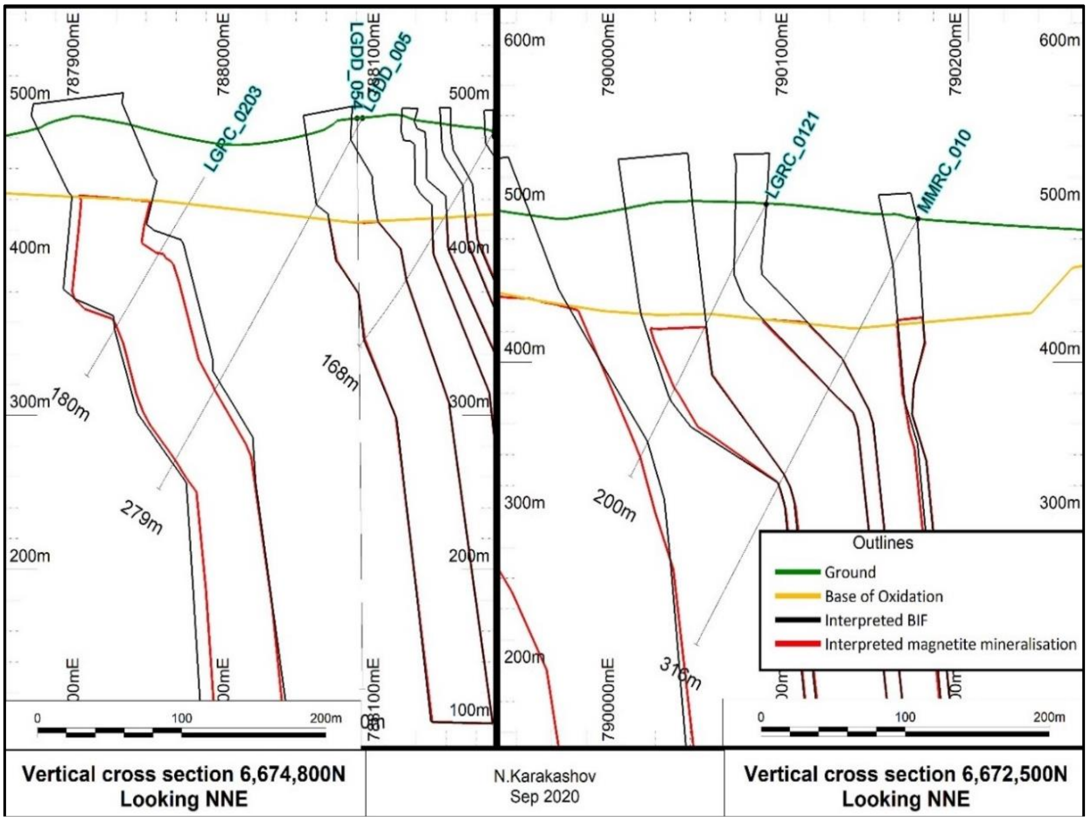


Figure 7-6: Cross sections through Moonshine North (left) and Moonshine (right)

Source: Karakashov (2020)

7.3. Weathering Profile

The rocks of the Lake Giles Iron Project have been logged into six different weathering classifications:

- Complete – All clay with no remnant rock texture
- Extreme – Largely clay with some remnant rock texture
- Strong – Rock texture moderately preserved, significant presence of fines, often weak
- Moderate – Rock texture fully preserved, all minerals show weathering
- Partial – Oxidation limited to the most unstable minerals only (e.g., sulphides); and
- Fresh – No oxidation of any minerals.

The majority of the hematite/goethite mineralisation grade (>50%Fe) material is located within the Strong and Moderate weathering classifications. The boundary between partial oxidation and fresh rock has variable depths within the Project area with down hole (-60º dip) depths ranging from 30 m to 100 m.

The magnetite is present in the fresh BIFs along with high quantities of silica. This is the primary unaltered form of BIFs at site and in general has not been subject to any significant later iron enrichment.

The base of the Complete oxidation weathering profile strongly plunges downward proximal to the BIF bodies, rapidly rising to a relatively shallow depth of 3–10 m in the mafic/ultramafic rocks were distant from BIF units. This shallow depth of weathering is only observed at a handful of locations. Majority of drillhole collars at the Project are situated close to BIF units, and the depth to the base of complete oxidation is logged to greater depths, compared to holes drilled distal to the BIF units.

Cross sections through the magnetite Mineral Resource are presented in Section 14.3 which demonstrate the variable depth of weathering with respect to proximity to the BIF units.

7.4. Water Table

The water table throughout the project area varies greatly in both level and salinity. The Snark area has been subject to a recent hydrogeological study (GRM, 2011) and the water table has been interpreted to between 50 to 65 metres below the surface, at an RL of 410 m to 425 m. With regards to salinity, the ground water in the area has a TDS value that typically ranges between 1,600 and 13,000 mg/L, which indicates a moderately brackish to saline classification (typical seawater is >35,000 mg/L).

8. Deposit Types

8.1. Mineralisation Styles

The tenements held by Macarthur are known to be prospective for iron as well as nickel and gold mineralisation. The iron mineralisation is related to the extensive BIF units that occur throughout the tenements. Aerial magnetic data shows that BIF units totalling at least 73 km of strike occur within the tenements, mostly under shallow cover.

Weathered and leached BIFs host the massive hematite iron mineralisation deposits located in the northwest of Australia, such as the Tom Price (Rio Tinto) and Mount Whaleback (BHP) deposits.

A BIF is defined as a rock composed of dark-coloured layers of iron-rich minerals that are interlayered with light-coloured, silica-rich material. These rocks were principally deposited worldwide from seawater as chemical sediments in marine basins or seas about two billion years ago.

The main minerals that form the layers in BIFs include quartz (silicon oxide), hematite (an iron oxide), siderite (an iron carbonate), and stilpnomelane (a potassium, iron, magnesium aluminosilicate). BIFs appear to have been deposited in areas of the ocean where seawater with high contents of dissolved iron and silica came into contact with water containing higher amounts of oxygen, which resulted in the precipitation of hematite and chert (microcrystalline quartz).

Most of the iron and silicon probably came from upwelling iron-rich, deep ocean currents derived from ocean floor volcanic systems. Because of their great thickness and the enormous areas that they cover, BIFs probably accumulated on wide continental shelves at water depths of over 200 m.

The process of iron deposition in the Proterozoic seas, 2.5–1.9 billion years ago, is thought to have involved a fine balance between the chemistry of the ancient atmosphere and oceans at a time when the oxygen content of the atmosphere was beginning to increase. It was the emergence of the earliest forms of life, tiny microbes (cyanobacteria) that produced oxygen through photosynthesis, that probably saw the composition of the early atmosphere begin to change.

An example of a typical BIF in outcrop is presented in Figure 8-1. The location of this outcrop is approximately 20 km to the north of the Lake Giles Iron Project; however, it is similar in nature to the BIFs located at the Project.

Figure 8-1: Example of layered BIF sequence surface expression, near LGRC_0038, Clark Hill South, showing alternating bands of increased iron oxide presence (dark layers) and more silica-rich bands (light layers)

Source: *Macarthur (2020)*

8.2. Conceptual Models

The mineralisation at Moonshine and Moonshine North deposits is associated with primary magnetite mineralization hosted by banded iron formation (BIF). The multiple BIF units steeply dip 75° to 85° to the west and strikes approximately 320° and 335° respectively the units have an average thickness of 15m, over a strike length of 17 km. The BIF units frequently outcrop.

9. Exploration

Macarthur took over the tenements now known as the Lake Giles Iron Project in late 2005 with the purchase of Internickel and its assets. Internickel was later renamed to Macarthur Iron Ore Pty Ltd in 2010. Macarthur immediately continued with the ongoing exploration program for nickel and gold.

In particular, anomalies generated by a 2004 helicopter electromagnetic survey (HoistEM) were visited and many were mapped and sampled, with emphasis on the search for nickel-bearing gossans.

Nine specific electromagnetic anomalies were identified and modelled, and five fixed loop transient electromagnetic surveys were then planned and undertaken by Outer-Rim Exploration Services from June to August 2006. The results were interpreted and reported by Southern Geoscience Consultants in September 2006. A number of anomalies were generated despite poor positioning of loops. No follow-up work was undertaken.

Iron mineralisation associated exploration activities (non-drilling) commissioned by Macarthur at the Project area since 2005 includes geological and geomorphological mapping and geophysics, including air and ground magnetic anomaly, ground gravity, rock chip, auger and regional soil sampling Table 9-1.

Table 9-1: Summary of Macarthur's Iron Exploration – 2005 to 2013

Period	Activity
2005 to 2006	Geological mapping and reconnaissance rock chip and auger sampling of exploration targets including pisolite and BIF iron targets.
June 2006	Auger sampling of pisolite iron targets, with approximately 229 holes drilled to around 4 m depth on a 100 m east-west by 500 m north-south pattern.
2008	Ground gravity survey.
2009	Ground magnetic anomaly survey at Moonshine and Moonshine North.
2009 to 2013	Lithological surface outcrop mapping.
2010	Geomorphological mapping of Lake Giles, covering all prospects. Aerial magnetic anomaly survey by Southern Geoscience Consultants.
2011	LiDAR topographic and imagery survey for the entire Lake Giles Project area.
2013	Regional soil sampling for entire Lake Giles Project area and Project area grid soil sampling.

The 2008 ground gravity survey by Haines Surveys (2008) covered a small portion of Moonshine and Moonshine North with 4,103 stations at intervals of approximately 200 m over 38 east-west trending lines spaced at 200 m to produce a Bouguer anomaly map used to aid in geological interpretations and targeting. Although the survey was targeting haematite mineralisation it has still proven useful in providing background information and support for the magnetite geological modelling.

In 2009, Resource Potentials performed a ground based magnetic anomaly survey in the Moonshine and Moonshine North prospects using 50 m line spacing for a total of 308-line km of data. The survey identified several prospective strongly anomalous magnetic bodies.

The survey suggested a depth extent of magnetite mineralisation of at least 200 m. The images from this survey have been extensively used to aid in geological modelling and to support the thick tabular steeply northeast dipping general shape of the magnetite-bearing BIF bodies.

Since 2009, exploration activity has focused on geological mapping and drilling of magnetite targets. Between 2009 and 2013, several outcrop mapping campaigns were undertaken by Macarthur staff and contractors covering the entire project area from Snark to Moonshine. Initially outcrop mapping concentrated on simply differentiating silica leached, haematite-goethite altered BIF and unaltered oxidised siliceous BIF to aid in targeting for haematite goethite enriched mineralisation. At the same time, a number of areas were mapped in much greater detail by CSA contractors, whereby Moonshine, Moonshine North and Clark Hill North were included. The detailed mapping performed in 2010 delineated a greater diversity of rock types especially over the BIF ridges and outlined the outcrops in fine detail, helping to establish the strike and serve as a good indicator of the width and continuity of the main magnetite bearing lodes at the Moonshine prospects.

This detailed mapping style was adopted for the later stages of Macarthur’s outcrop mapping, which re-mapped Snark and all remaining haematite-goethite prospects, increasing the geological confidence in the modelling in those areas.

Detailed mapping from 2009 through to 2012 outlined surface lithologies of interest, which were targeted in subsequent drilling programs. These mapping programs assisted in defining the continuity and thickness of individual mineralised domains, supporting the Mineral Resources that are the subject of this report.

Russel (2010) conducted regional geomorphological mapping of the Lake Giles area, with the specific aim to produce a list of secondary iron targets such as canga (iron-rich duricrusts that cap iron mineralisation), detrital iron deposits, channel iron deposits, and bedded iron deposits for follow-up exploration by Macarthur. The mapping area extended over the entire Project area from the northern extents of Snark to beyond the southern extents of Moonshine. Some of the generated targets were subsequently drilled to evaluate the regional potential for secondary iron deposits at Lake Giles. The mapping also proved useful for general geological modelling throughout the Project area and assisted in the targeting and placement of some holes as well as a better understanding of the weathering depth and profile, especially over the mafic and ultramafic lithologies surrounding the BIF bodies.

In June 2011, Outline Global Pty Ltd performed a 200 Hz LiDAR survey over the entire Lake Giles Project area, producing a 1 m resolution 0.5 m contour terrain model, as well as RGB composite and NIR imagery. The LiDAR survey was used for environmental assessment and targeting, vegetation mapping as well as infrastructure planning and accurate terrain surface modelling for Mineral Resource estimates and geological mapping.

In 2013, a regional soil sampling program was undertaken by Macarthur, which included several spacing patterns. A 1,000 m grid pattern covered the entire project area from Snark to Moonshine and beyond. Following this, several areas in Moonshine and Moonshine North were covered by a 100 m x 200 m (east x north) grid pattern. Although the soil sampling data was not directly used in the magnetite resource estimates, the results were nonetheless useful in geological modelling of the mafic and ultramafic rocks surrounding the BIF packages.

In addition to the above exploration data, some optical televiewer surveys were conducted by ABIMS between 2011 and 2013, as part of the gyro downhole surveys, immediately following drilling. Further information regarding televiewer data is presented in section 10.2.3 Drillhole Surveys.

A summary of exploration drilling methodology and results, used to support the Mineral Resource estimates discussed in this report, are presented in Section 10.

10. Drilling

10.1. Drilling Summary

The magnetite Mineral Resource estimate includes drilling and sampling completed from 2006 to 31 December 2019.

Drill collar plans are presented in Figure 10-1 to Figure 10-3 Figure 10-2 showing the locations of drillhole collars superimposed on Banded Iron Formation surface geology.

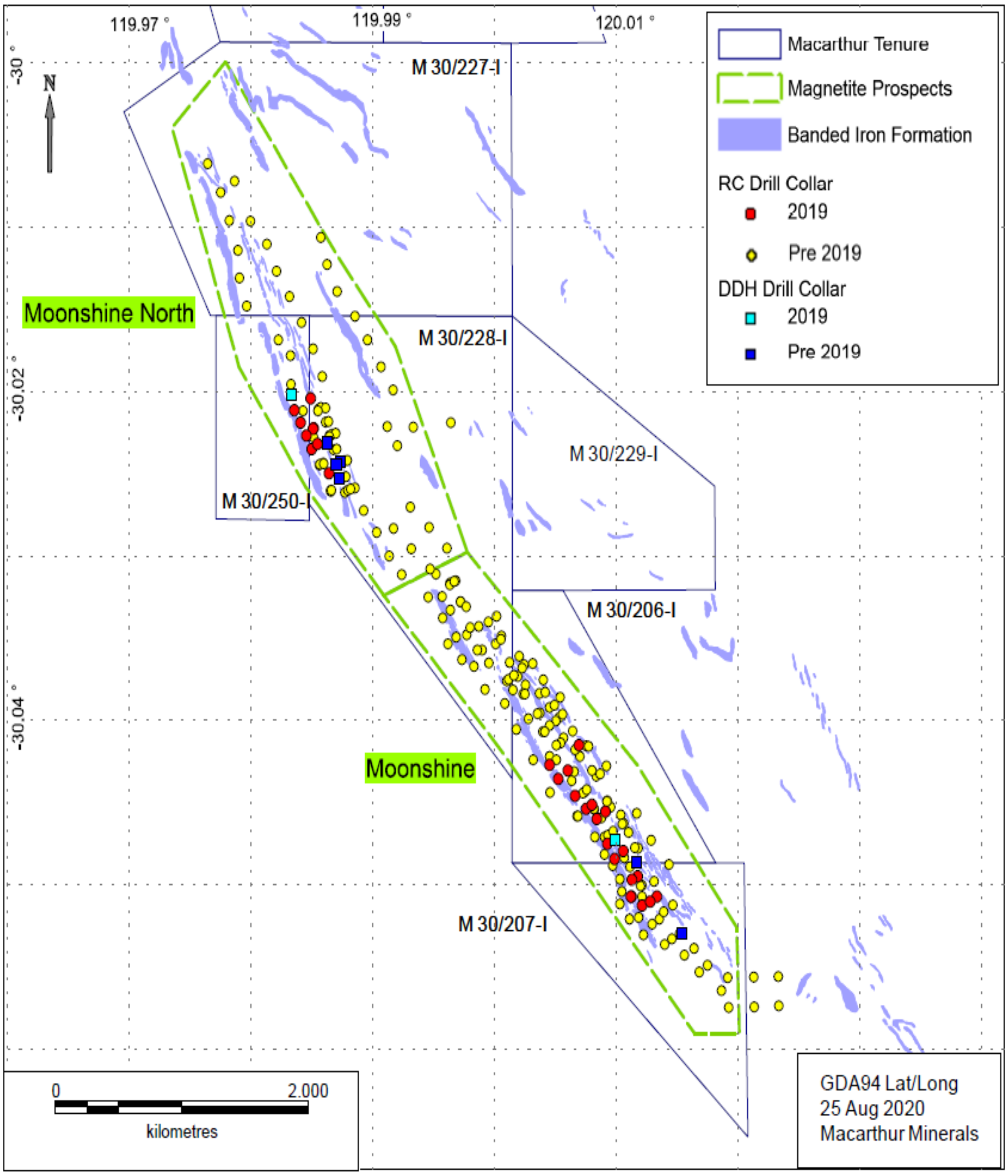


Figure 10-1: Drill collar plot, Moonshine and Moonshine North, showing drill collars by type and program, mapped BIF outcrop and tenure

Source: Macarthur (2020)

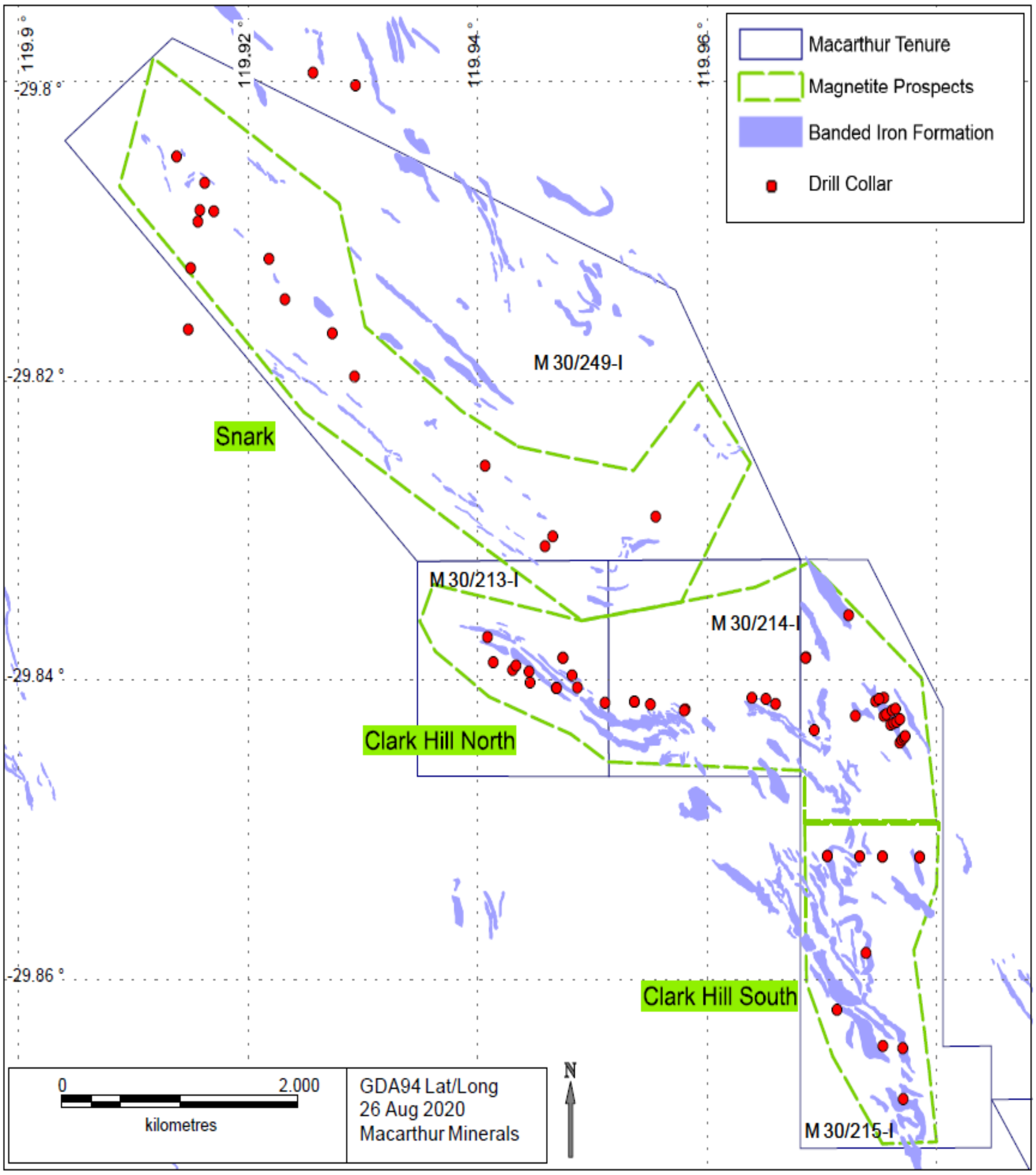


Figure 10-2: Drill collar plot, Snark, Clark Hill North and Clark Hill South, showing RC drill collars, mapped BIF outcrop and tenure

Source: Macarthur (2020)

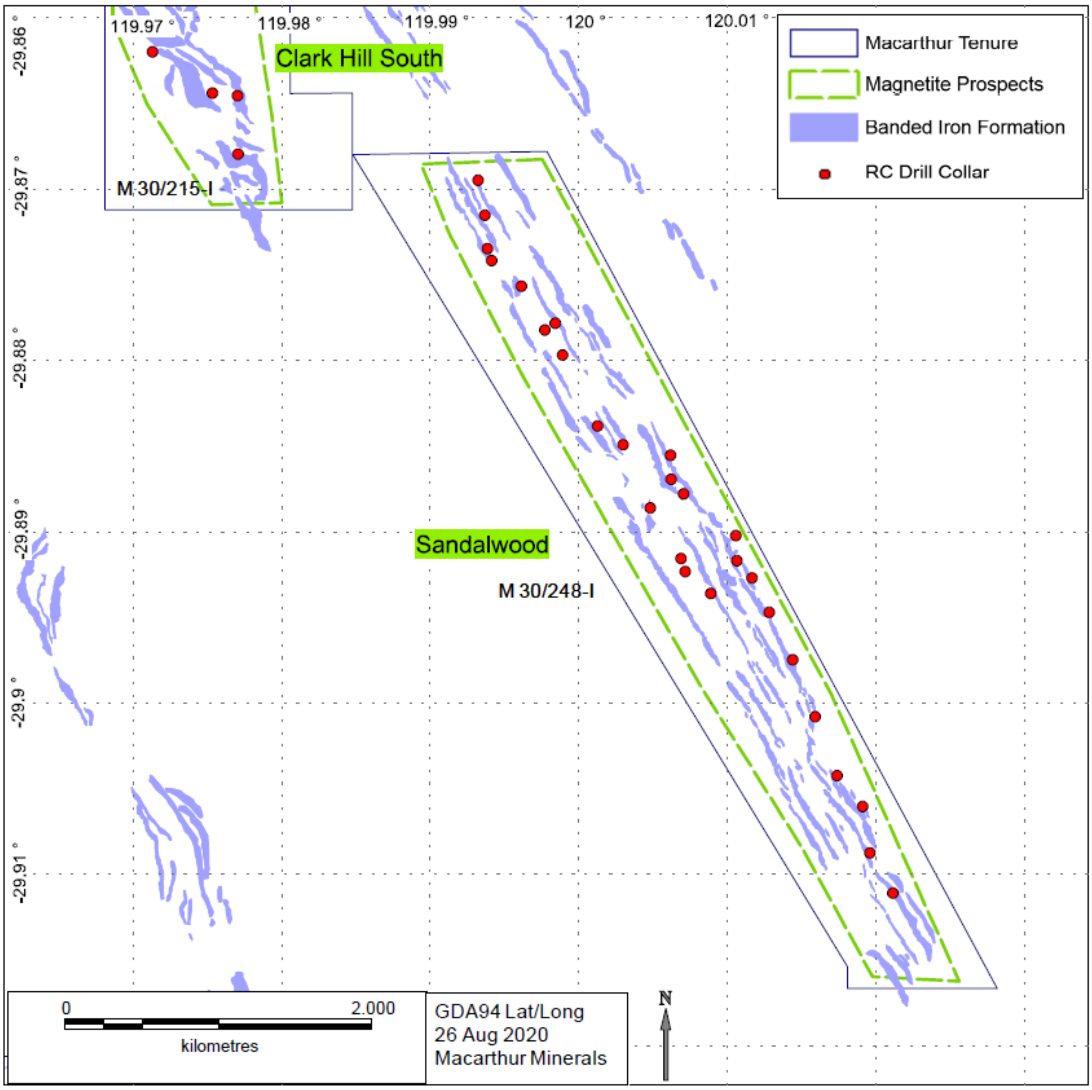


Figure 10-3: Drill collar plot, Sandalwood, showing RC drill collars, mapped BIF outcrop and tenure (the southern end of Figure 10-2 is to the north of the plot)

Source: Macarthur (2020)

Macarthur's drilling at the Snark, Clark Hill, Sandalwood, Moonshine and Moonshine North prospects totals 373 RC and diamond drillholes. These results do not include drilling at Macarthur's hematite project.

In the Snark, Clark Hill North, Moonshine and Moonshine North prospects, most of the drillholes are drilled perpendicular to strike of the BIF units, intersections approximate the true thickness of the BIF units. In Clark Hill South, the orientation of drillholes varies and is not always perpendicular to surface outcrops due to the structural complexity of the area in comparison to the other prospects, where the BIF ridges are relatively continuous and consistent in strike.

In Moonshine, most of the drillholes are oriented 080° azimuth, dipping -60° or 240°azimuth dipping -60°, with a minor number of drillholes having a 030° azimuth dipping -60° or a dip of -90°. At Moonshine North, the azimuths range from 240° to 280° but all dip -60° towards the west.

The drillhole spacing varies from 50 m to 300 m and does not transect the mineralisation on some transverses.

Table 10-1 presents a summary of all drilling, by deposit area, at the Lake Giles Iron Project.

Table 10-1: Summary of Lake Giles Iron Project drilling by deposit area

Deposit	Years	Diamond holes		RC holes		Total	
		No.	Metres	No.	Metres	No.	Metres
Clark Hill North	2006–2010	5	1,002	60	8,551	65	9,553
Clark Hill South	2006–2007	-	-	9	2,086	9	2,086
Sandalwood	2007–2010	-	-	38	6,933	38	6,933
Snark	2006–2007	-	-	16	3,007	16	3,007
Moonshine/Moonshine North	2008–2019	16	3,155	229	41,808	245	44,963
Total		**21**	**4,157**	**352**	**62,385**	**373**	**66,542**

10.2. Drilling Techniques and Procedures

10.2.1. Overview

Drilling and sampling procedures were consistent across all exploration prospects within each drilling program, with minor changes adopted across the years as different campaigns employed different practices.

Macarthur contracted Orbit Drilling Pty Ltd to carry out both the RC and diamond drilling for all prospects between 2006 and 2018, and then iDrilling Australia (previously named Orbit Drilling Pty Ltd) in 2019. Both firms are exploration drilling companies based in Perth, Western Australia. Two RC drill rigs were utilised, a Schramm T660 (Volvo 8x4 wheel rig) and a track mounted Schramm T450WS.

Macarthur has a number of procedures in place, which have been designed to reduce the risk of errors from drilling, sampling and assaying processes. These procedures are summarised below.

10.2.2. Drillhole Planning

Holes drilled prior to 2019 were planned and supervised by Macarthur geological staff. Infill drillholes drilled in 2019 at the Moonshine deposit were planned by the QPs of this report and supervised by Macarthur geological personnel. Holes were planned to intercept the host lithologies in the most representative way possible, with consideration given to local terrain, outcropping geology and results from previous drilling. During RC drilling, a Company geologist would supervise the work and log the geology to each metre interval (or at appropriate intervals during diamond drilling) and end the hole at a certain depth based on the outcome of the drilling and the estimates provided by the drillhole planning.

10.2.3. Drillhole Surveys

Planned drillhole collar positions were marked by GPS, and if clearing was required to provide a suitable drill site, then planned collar positions were re-marked after clearing. To assist with drill rig alignment, two sighter pegs were placed at appropriate distances from the collar position using a sighter compass.

In areas of high magnetic field deviation due to underlying magnetite bodies, a GPS azimuth method was used. All drill collars were surveyed with high accuracy Real Time Kinematic (RTK) GPS by surveyors from Minecomp Pty Ltd and are accurate to within 50 mm in three dimensions.

After the drill rig was set up on each hole, Macarthur staff checked the planned hole inclinations with a clinometer. Holes drilled prior to 2010 were downhole surveyed with a single-shot downhole camera lowered down the rod string, with surveys generally taken at 30 m intervals

All holes drilled after 2009 were surveyed with a GYRO tool. Surveys were conducted at sub-metre accuracy and composited into 5 m intervals before the results were entered into the drillhole database. For the 2019 drilling campaign, a drilling contractor supplied Reflex Sprint-IQ gyro tool was used with readings taken at a nominal spacing of 10 m.

10.3. Drillhole Logging

10.3.1. General

Diamond drill core and RC chip samples were geologically and geotechnically logged to a level of detail required to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. Not all drillholes penetrated the BIF host units, but all were used to guide the geological interpretations supporting the Mineral Resource estimates.

All drillholes were geologically logged, using Microsoft Excel™ spreadsheets pre-formatted for use by Macarthur geologists, with lithologies, oxidation, structure, alteration, and mineralogy among the geological categories logged. Geological logging of drill samples was qualitative in nature for all RC drilling and diamond core samples.

In addition to GYRO surveying, some holes were also surveyed using optical televiewer, producing photographical, as well as interpreted structural defect logs. The holes surveyed were drilled between 2011 and 2013, with some of the holes forming part of the Lake Giles Magnetite project (Snark and Moonshine North prospects) – LGDD_0007-0017 and LGDD_0054. The structural data and photographic logs were used as aids in geological modelling, most notably assisting in defining structural trends for BIF bodies.

10.3.2. Diamond Drill Core Logging

Diamond core drilling used mostly HQ diameter core with occasional PQ core depending on the mass of core required. Core orientation was performed using Reflex apparatus, which was unsuccessful for majority of core samples obtained from within the weathered rock profile.

The core from the diamond drillholes were geologically and geotechnically logged incorporating structural measurements, by contract geologists or Macarthur geologists. Figure 10-4 presents an example of diamond core from drillhole LGDD069 and shows the BIF host rock with magnetite mineralisation.

Figure 10-4: Diamond Core Sample from Drillhole LGDD-069, 98.29 m to 101.56 m

Note: Magnetite layers (dark) and chert (light) can be seen.
Source: Macarthur (2020)

The structural orientation of a planar feature is defined by the alpha angle, which is measured by the core axis, and the beta angle which requires a bottom or top of the core axis defined by the orientation line. Although both the alpha and beta angles are required to calculate an orientation of the structural feature, if the strike of the feature is known, some information about the dip can be inferred from the alpha angle.

Diamond core recoveries were recorded by measuring the length of drill core retrieved per metre of drill penetration.

Core photography was undertaken for all diamond drilling, with one photo per core tray, ensuring all labelling is clear and visible.

10.4. Representative Drill Sections

Representative cross sections of the Moonshine deposit, showing the geological interpretation and drilling, are shown in Figure 14-2 to Figure 14-4 in Section 14.3.

10.5. Density Determinations

From the 2019 drilling program at Moonshine and Moonshine North, a total of 624 diamond core billets were selected for the measurement of density, with 400 of the samples logged as the BIF host rock unit. Samples were selected from unmineralised and mineralised BIF, and fresh and weathered BIF. The oxidised BIF is competent, exhibiting few fractures, vugs or voids which would normally necessitate the need to coat the core samples with paraffin wax prior to immersion in water for weighing. Therefore, the geological staff determined that the core samples did not require wax coating.

For the Clark Hill deposit, density measurements were taken from 122 diamond core billets sampled from four diamond holes, with 63 of the samples located within the BIF host rock. Density measurements were taken using a conventional Archimedes technique.

Further discussion is provided in Section 14.3.10 and Section 14.4.9.

Density measurements were carried out in the field camp by Macarthur staff using a conventional “Archimedes” procedure, where the samples were weighed in air and then weighed in water. The difference between weight (air) and when the sample is weighed in water equates to the mass of the displaced water and hence the volume of the core sample.

The basic Archimedes formula used to calculate the density is:

$$Density = Weight\ (air)/(Weight\ (air) - weight\ (water))$$

Figure 10-5: Dry Core Samples Prior to Immersion and Weighing in Water

Source: Macarthur (2020)

11. Sample Preparation, Analysis and Security

11.1. Field Sample Preparation, Handling and Security

11.1.1. Sample Handling and Security

Sample collection, handling and dispatch was of a high standard, with good practices employed throughout the process. Security tags were used at all steps of sampling and dispatch through to delivery at the relevant sample preparation and analytical laboratories.

Sample preparation for drillhole samples have followed consistent methodologies since drilling of the Project commenced in 2006. On completion of each hole the field assistants collect the samples and secure them in poly-weave bags using a cable tie labelled with a unique ID, which the lab would check upon receipt as a way of being aware of tampering. The poly-weave bags were securely stored in the Project exploration camp compound, where Macarthur personnel were present on a continual basis during the course of the drilling programmes.

The samples were transported to the assay laboratory depot in Kalgoorlie in a large bulk bag to avoid loss of samples, prior to being dispatched to the assay laboratory in Perth using a local courier company.

11.1.2. Reverse Circulation Sampling

Drilling practices are focused on maximising sample recovery and minimising sample contamination. For RC drilling, at the end of each 6 m drill rod, the drilling paused and compressed air was blown through the rods to flush cuttings from the drillhole, the sample hoses, and the cyclone to minimise sample contamination, and to ensure that there were no blockages in the sample stream. The cyclone was regularly inspected and cleaned as necessary. Samples were collected over 1 m downhole intervals and a subsample was collected in a calico bag by splitting through an industry standard three-tier riffle splitter. The splitter was calibrated for 75% of the sample passing through the splitter to be captured in a residue bucket, whilst the remaining 25% of the sample was evenly distributed through the primary sample chute and the field duplicate chute (*Figure 11-1*). The calico bag subsamples were labelled with the drillhole number and depth range and placed on top of the remnant bulk sample, which was placed in individual piles on the ground alongside the drill collar (*Figure 11-2*). All primary 1 m samples were submitted to the assay laboratory. Sample recovery is estimated from the appearance and volume of the primary sample, contained within its calico bag, and the remnant bulk sample.

Sample quality from RC drilling at the Lake Giles Iron Project has been judged by Macarthur and the QPs to be very good, with consistent recoveries and sample quality, such as dryness of sample.

Figure 11-1: Three-tiered splitter on an RC drill rig, showing collection of primary sample and field duplicate (sample residue is collected in the bucket)

Source: Macarthur (2020)

Figure 11-2: Drill samples laid out prior to collection and dispatch to assay laboratory

Source: Macarthur (2020)

11.1.3. Diamond Drilling Sampling

Diamond core drilling used mostly HQ diameter core with occasional PQ core depending on the mass of core required.

After the core was logged and sample intervals marked out by the geologist, the diamond core was cut using an electric core saw Figure 11-3 for samples obtained from competent ground, or hand split when the core sample was unconsolidated, at either 1 m intervals or to geological contacts.

Figure 11-3: Diamond saw used for cutting diamond core, as used during the 2019 drilling program

Source: Macarthur (2020)

11.2. Laboratory Sample Preparation and Analyses

Samples from Sandalwood, Clark Hill North, Clark Hill South, and Snark were submitted to either Genalysis or Amdel Laboratories in Perth, Western Australia. Samples taken during the 2019 drilling program at Moonshine were dispatched to SGS Australia, located in Perth (Table 11-1).

Table 11-1: Independent Laboratories used in the Various Drill Programs

Laboratory	Location	Accreditation
Amdel Laboratories wholly owned by Bureau Veritas	6 Gauge Circuit, Canning Vale, Western Australia	ISO 9001 Quality Management System certification and NATA accreditation (Accreditation number 626)
Genalysis Laboratory Services wholly owned by the Intertek Group	15 Davison Street, Maddington, Western Australia	Accredited by NATA to operate in accordance with ISO/IEC 17025, which includes the management requirements of ISO 9001
SGS Australia Pty Ltd	431 Victoria Road, Malaga, Western Australia	Accredited with ISO 9001

All laboratories are accredited by the National Association of Testing Authorities (NATA) in accordance with ISO/IEC 17025, which includes the management requirements of ISO9001:2015. All the laboratories are independent of Macarthur.

All laboratories used over the course of the Project maintained sound security for all samples, from receipt of sample to storage of crush and pulp residue (limited storage time). Assay results were emailed to Macarthur.

Assays were performed on majority of single metre RC intervals and on selected diamond core intervals, averaging 1 m, while accounting for lithological boundaries. DTR analyses were performed on composited samples using lab bulk residues from the primary samples according to compositing instructions from Macarthur staff. The average composite length was 5 m, with geological staff grouping together intervals of similar character and setting boundaries at lithological changes, giving occasional composites between 2 m and 6 m in length.

Samples were delivered to the analytical laboratory where they were crushed to 3 mm, then pulverised to 105 µm (p95). The samples were subject to XRF analysis, with results provided for a suite of 25 elements, in addition to loss on ignition (LOI). Table 11-2 presents the elements or oxides analysed for head and concentrate grades, by analytical laboratory. Further detail for each laboratory is presented in Section 11.2.1 to 11.2.4.

Table 11-2: Laboratory analysis details

Analysis	Laboratory	Elements and oxides
Head grades	Genalysis	Au, Cu, Pb, Zn, Ni, Fe, Fe_2O_3, Co, K_2O, As, Ba, Cl, CaO, MgO, MnO, P, S, SiO_2, Al_2O_3, TiO_2, LOI
	Amdel	Ni, Fe, Al_2O_3, CaO, MgO, P, S, SiO_2, TiO_2, LOI
	SGS	Fe, Al_2O_3, SiO_2, Ba, Cr, Co, MnO, P, S, Pb, Cl, Sn, CaO, TiO_2, K_2O, Cu, As, Sr, MgO, Na_2O, Zn, V, Ni, Zr, H_2O, LOI
DTR and concentrate grades	Amdel	Fe, Fe_2+, SiO_2, Al_2O_3, TiO_2, CaO, MgO, P, S, Na_2O, K_2O, Cr, LOI
	SGS	Fe, Al_2O_3, CaO, Cr, K_2O, LOI, MgO, MnO, Na_2O, S, P, SiO_2, TiO_2, V, Ba, Co, Cu, Pb, Cl, Sn, As, Sr, Zn, Ni, Zr

Selected sample splits were ground to p98 45 µm and subjected to DTR testing with XRF analysis performed on head and concentrate material. A mass recovery estimate was calculated, which is the percentage of the sample that is considered recoverable by magnetic separation. The magnetite product is contained in this recovered fraction. A flowchart for this process at the Amdel laboratory can be seen in Figure 11-4 which is also considered to be representative for all XRF and DTR analysis procedures at the other analytical laboratories used.

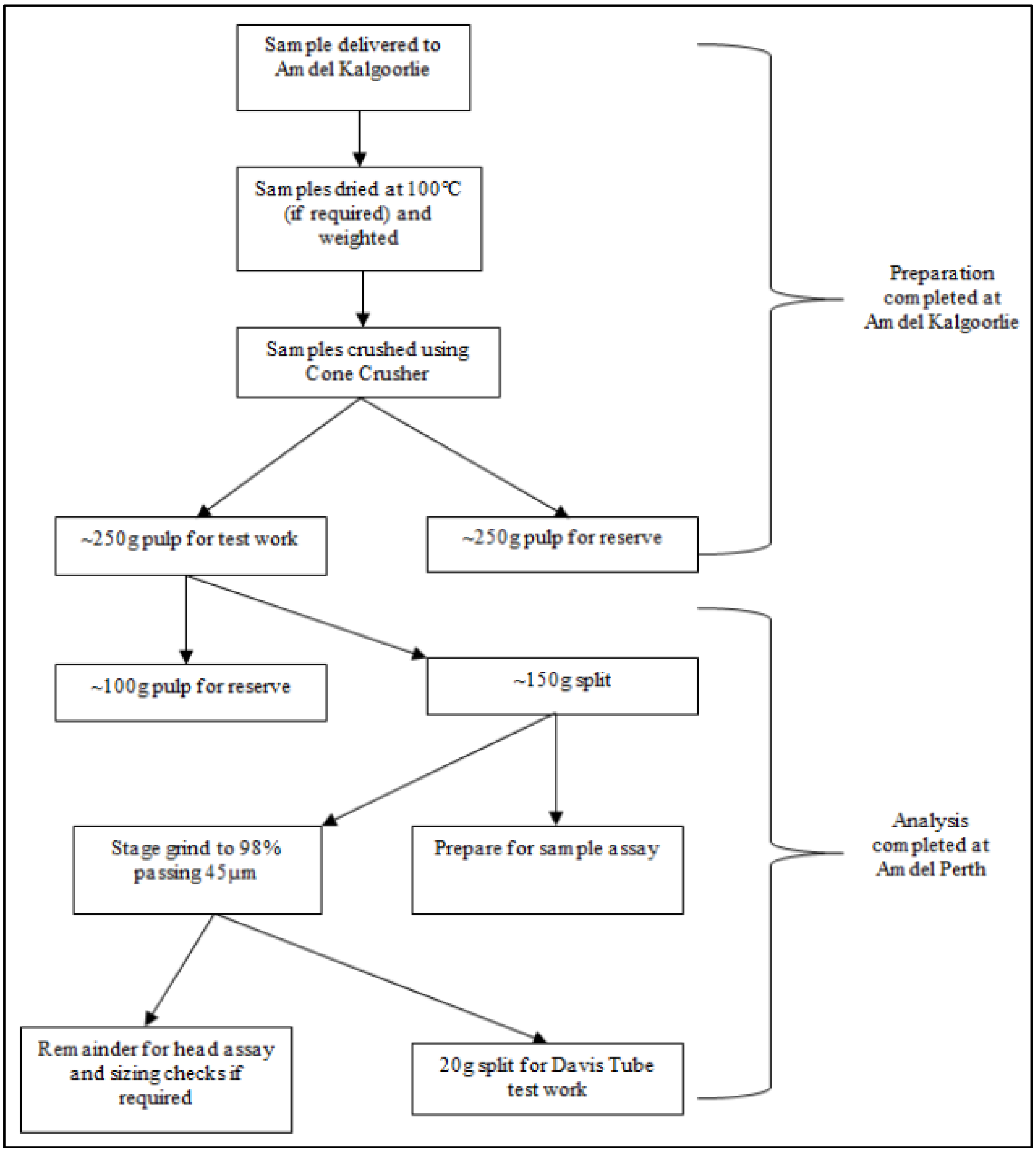


Figure 11-4: Flowchart of the analysis of sample at Amdel laboratory

Source: Macarthur (2020)

11.2.1. Genalysis Laboratory Services Pty Ltd

Head grade analyses of samples from 2006 were performed at Genalysis (Abbott et al., 2009b). These samples were sourced from the Sandalwood, Clark Hill North, Clark Hill South, and Snark prospects.

Genalysis' sample preparation procedure for the RC drill chips commenced with sorting and oven drying, followed by robotic sample preparation comprising crushing the entire sample to nominally 2 mm, and riffle splitting a 1 kg subsample, with the bulk residue retained. The 1 kg subsamples were pulverised to nominally 85% passing 75 microns and split into a 200 g subsample, and 800 g retained sample.

The samples were analysed by XRF in accordance with Genalysis procedure designated as “FUS1”. Subsamples of the pulverised material were fused with a suitable flux and poured into a mould to produce a homogenous glass disc. Grades of the elements of interest (Table 11-2) were determined by simultaneous XRF.

11.2.2. Amdel

Head grade analyses for samples taken between 2007 and 2009 were performed at Amdel (Abbott et al., 2009b). The RC drill chip samples were initially sorted and dried at 105°C before being crushed to minus 3.5 mm using a Rocklabs Boyd Crusher, and subsequently pulverised in a ring mill.

The samples were analysed by XRF in accordance with Amdel procedure designated as “XRF4”. Subsamples of the pulverised material were fluxed with a lithium-metaborate flux and cast into a 30 mm diameter disc. Grades of the elements of interest (Table 11-2) were determined by simultaneous XRF using a Philips PW-1480 XRF spectrometer.

CRMs are fused with each batch of samples and are analysed as per the drillhole samples. Amdel also performed LOI analyses on a separate pre-dried portion of the sample in electric furnace set to 1,000°C.

11.2.3. Amdel Davis Tube Recovery Analysis

All DTR analysis of samples collected between 2006 and 2009 was completed by Amdel using 150 g subsamples split from the jaw-crushed residue samples, which were further pulverised to 45 µ with a ring pulveriser. The pulverised material was repeatedly wet sieved at 45 µ, and the coarse fraction reground until the oversize component was less than 5 g. A 20 g subsample was collected for DTR testwork.

The Davis Tube magnetic concentration procedure used a 25 mm diameter tube with a stroke length of 38 mm and a stroke frequency of 60 cycles per minute, with a magnetic field strength of 3000 gauss. The magnetic concentrate material was analysed by XRF for a range of elements using a procedure consistent with Amdel’s XRF analysis of head grade samples.

11.2.4. SGS Australia

Head grade, DTR analyses and concentrate grade analyses of samples from the 2019 drilling programme were performed at SGS Australia Pty Ltd (SGS). Samples were weighed upon receipt at the lab, dried at 105°C before being coarse crushed to a nominal 6 mm size, then a 3 kg split was dry pulverised to 85% passing 75µ. The sample was then fused in a platinum crucible using lithium metaborate/tetraborate flux and the resultant glass bead irradiated with x-rays and the elements of interest quantified. LOI was determined by a LECO thermo gravimetric analyser (TGA) at temperatures of 105°C, 371°C, 650°C and 1,000°C.

The DTR analyses used a 40 mm diameter tube with a stroke length of 38 mm and a stroke frequency of 60 cycles per minute, with a magnetic field strength of 3000 gauss. The magnetic concentrate material was analysed by XRF for a range of elements as detailed in Table 11-2 using a procedure consistent with SGS’s XRF analysis of head grade samples.

11.3. Quality Assurance and Quality Control

11.3.1. Overview and Summary of Methodology

QAQC practices and processes have been implemented by Macarthur for the drilling programs since 2006.

CRMs (or standards) were used throughout the drilling programs to test analytical accuracy, at a rate of 1:50 with at least one standard inserted per drillhole. Field duplicates were collected at a rate of 1:25 prior to 2019 and 1:20 in 2019. Pulp duplicates from pre-2019 drilling was also re-analysed in 2019 to test for analytical accuracy. A selection of pulp samples was also sent to Genalysis Intertek for umpire analyses of head grade XRF results.

The analytical laboratories conducted their own QAQC analyses and results were provided to Macarthur. The QAQC procedures and results showed that acceptable levels of accuracy and precision were established over the life of the drilling programs at the Project.

11.3.2. Blanks

No blank standards were used during the 2006–2009 sampling programs.

Macarthur employed the use of CRM GIOP-119 (refer Figure 11-5) during the 2019 drilling campaign as a blank testing standard, with exceedingly low iron grade in comparison to expected grades encountered at the Lake Giles Iron Project. All 177 instances of the CRM test returned results within accepted ranges with no significant grade bias to report, as shown in Figure 11-5.

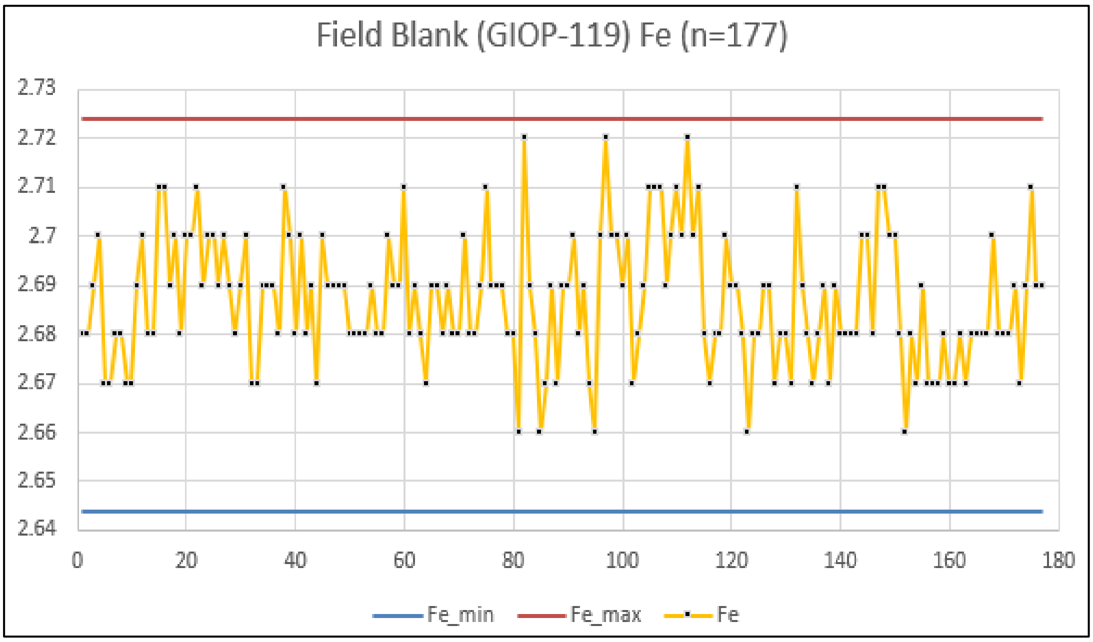


Figure 11-5: 2019 GIOP-119 blank testing, SGS

Source: Macarthur (2020)

11.3.3. Certified Reference Materials – Amdel 2007 to 2009 Drilling Programs

CRMs analysed at Amdel between 2007 and 2009 showed the majority of the assays falling within the expected ranges, as shown in Figure 11-6 and Figure 11-7. No CRM analyses were performed during the 2006 drill program.

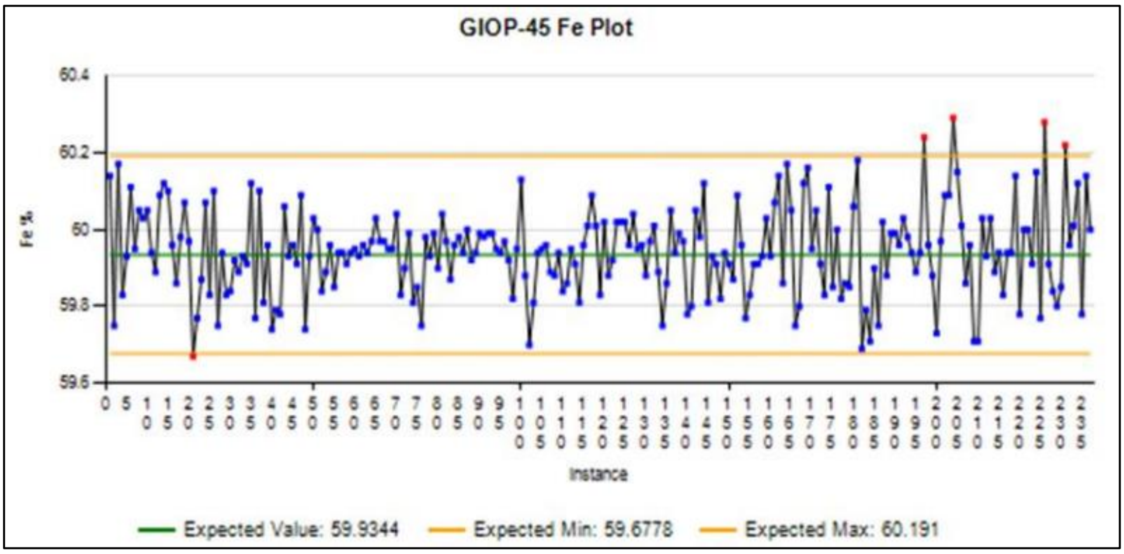


Figure 11-6: CRM performance chart for GIOP-45, Amdel (expected limits are ± 2 standard deviation)

Source: Macarthur (2020)

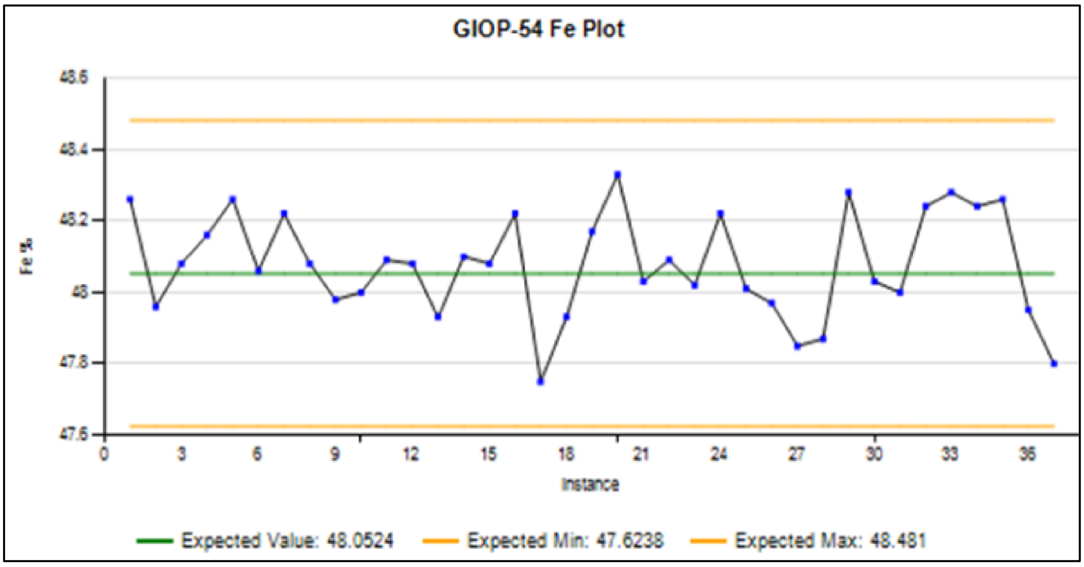


Figure 11-7: CRM performance chart for GIOP-54, Amdel (expected limits are ± 2 standard deviation)

Source: Macarthur (2020)

11.3.4. SGS Australia – 2019 Moonshine Drilling Program

The 2019 Moonshine drilling campaign used eight different CRMs supplied by Geostats Pty Ltd. A total of 369 CRM samples were assayed by SGS. A short summary of the overall results is presented in Table 11-3. Selected CRM performance charts are presented in **Figure *11-8*: CRM performance chart, GIOP-45 (Fe)**Figure 11-8 to Figure 11-10. In general, the 2019 drilling campaign's CRM testing was successful and within expected ranges for majority of samples tested.

Analysis of laboratory results showed that sample GIOP-102 showed some strong negative biases in iron and silica, a low negative bias in phosphorous and a strong positive bias in LOI, with the remaining elements in close range to expected averages.

Macarthur notes that an internet search for other projects using GIOP-102 also reported iron and silica analyses below the expected range. It is recommended that Macarthur discontinue use of this CRM.

Table 11-3: CRM summary data

CRM	No. tested	Fe			Al2O3		
		Expected mean	2σ error range	Mean	Expected mean	2σ error range	Mean
GIOP-102	57	25.60	0.18	25.28	2.051	0.102	2.04
GIOP-111	30	33.35	0.3	33.17	0.2213	0.0162	0.22
GIOP-118	40	71.51	0.26	71.47	0	0	0.01
GIOP-119	177	2.68	0.04	2.69	0.0264	0.02	0.01
GIOP-134	5	47.52	0.2	47.50	9.953	0.15	9.87
GIOP-135	17	53.51	0.16	53.46	7.322	0.104	7.26
GIOP-142	22	56.58	0.28	56.57	3.032	0.05	3.02
GIOP-45	21	59.93	0.26	59.92	2.024	0.062	2.01
GIOP-102	53.35	0.52	52.81	0.0758	0.0026	0.074	-0.194
GIOP-111	48.26	0.34	48.05	0.0674	0.0028	0.066	-1.069
GIOP-118	0.76	0.074	0.76	0.0058	0.0024	0.006	-3.857
GIOP-119	86.05	0.5	86.05	0.1225	0.003	0.120	0.634
GIOP-134	13.47	0.13	13.48	0.0577	0.002	0.057	4.452
GIOP-135	9.63	0.108	9.61	0.05917	0.00178	0.059	3.562
GIOP-142	6.70	0.062	6.71	0.0412	0.0024	0.041	8.368
GIOP-45	4.99	0.09	4.99	0.0505	0.0022	0.050	6.615

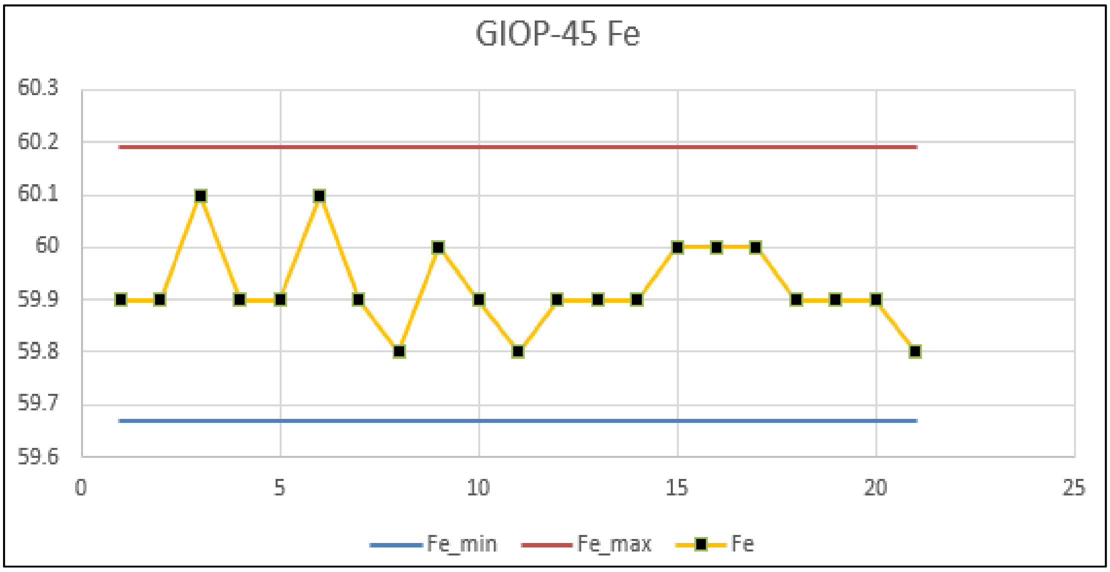


Figure 11-8: CRM performance chart, GIOP-45 (Fe)

Source: Macarthur (2020)

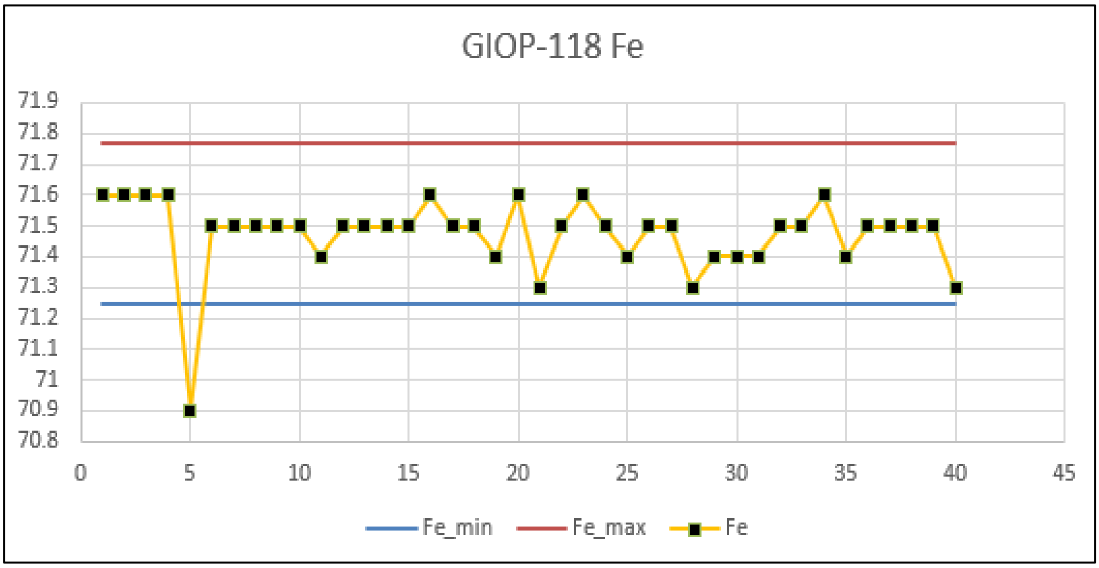


Figure 11-9: CRM performance chart, GIOP-118 (Fe)

Source: Macarthur (2020)

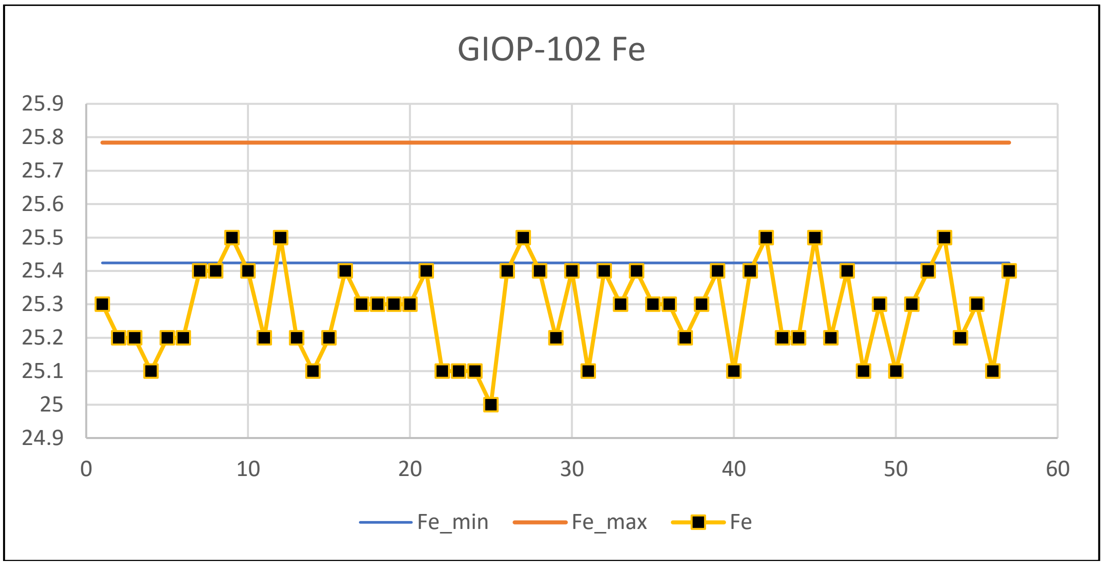


Figure 11-10: CRM performance chart, GIOP-102 (Fe)

Source: Macarthur (2020)

11.3.5. Field Duplicates

A number of field duplicates were tested as part of drilling programs at the Lake Giles Iron Project. Scatter plots for Fe (%) are presented in Figure 11-11 and Figure 11-12. These demonstrate a tight clustering around the 1:1 line, although there are outliers. These outliers may be due to misallocation of field duplicate samples (sample bags erroneously labelled) or sampling bias at the drill rig. A very high correlation coefficient (0.99) implies sampling at the drill rig was maintained at a high level of proficiency.

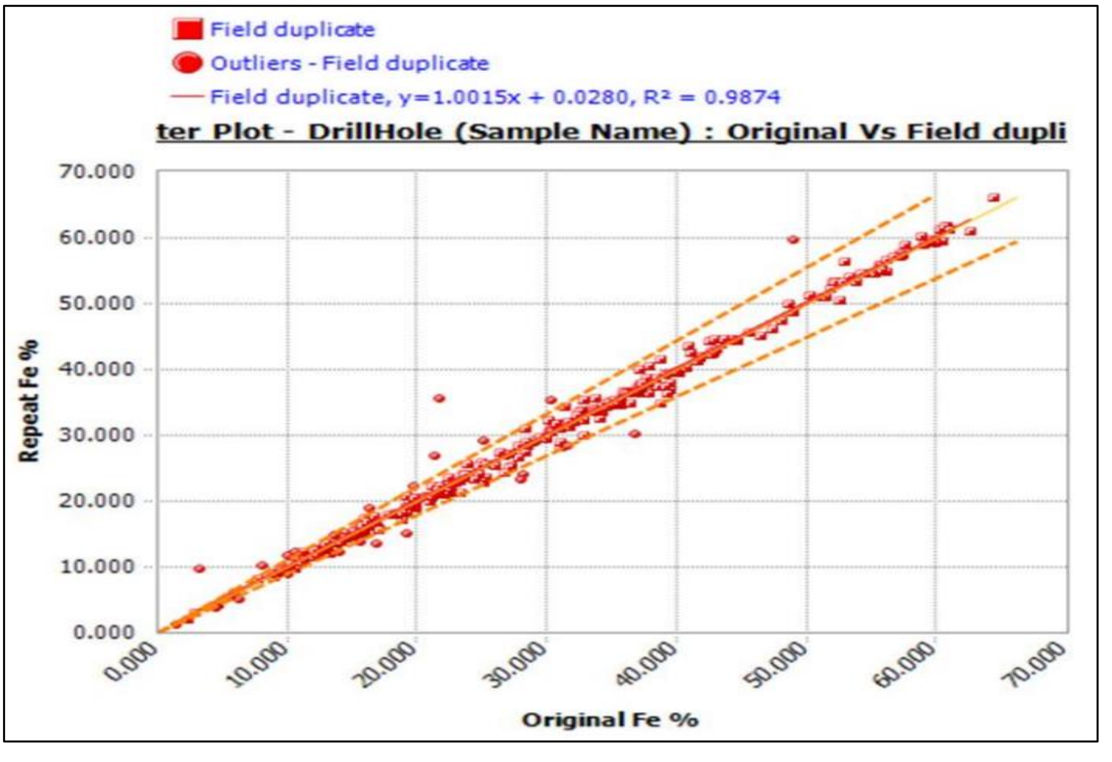


Figure 11-11: Field duplicate testing, Amdel

Source: Macarthur (2020)

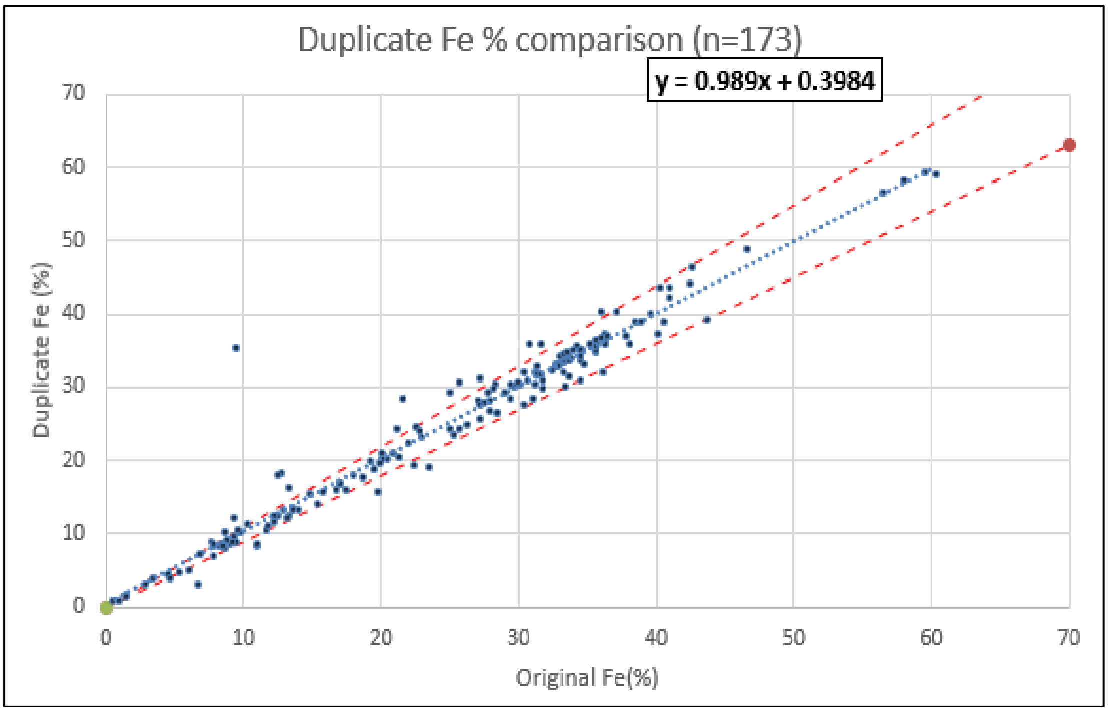


Figure 11-12: 2019 Field duplicate testing, SGS

Source: Macarthur (2020)

11.3.6. 2006–2013 Pulp Duplicates

During the 2019, Moonshine drilling campaign a total of 101 pulp residue samples from drilling conducted between 2006 and 2013 were submitted to SGS to compare the assays against the original head XRF assay values for consistency.

Samples were selected in order to represent a variety of holes and grades from both Moonshine and Moonshine North, especially around the central portions of the resource.

The samples chosen had been stored as pulps in boxed sealed packets in sealed sea containers at the Lake Giles sample storage compound. The packets were assigned new sample IDs and dispatched to SGS along with samples from the 2019 drilling program with appropriate security tags.

The selected samples included pulps tested at Amdel. Assays of the pre-2019 pulps have shown very consistent and repeatable results for all samples tested (n=101) with a sub-1% error range (resulting in 0.5% Fe grade difference at 50% Fe) and no significant grade bias shown by SGS assaying compared to Amdel. Scatterplots for Fe, SiO_2 and LOI are presented in Figure 11-13 to Figure 11-15.

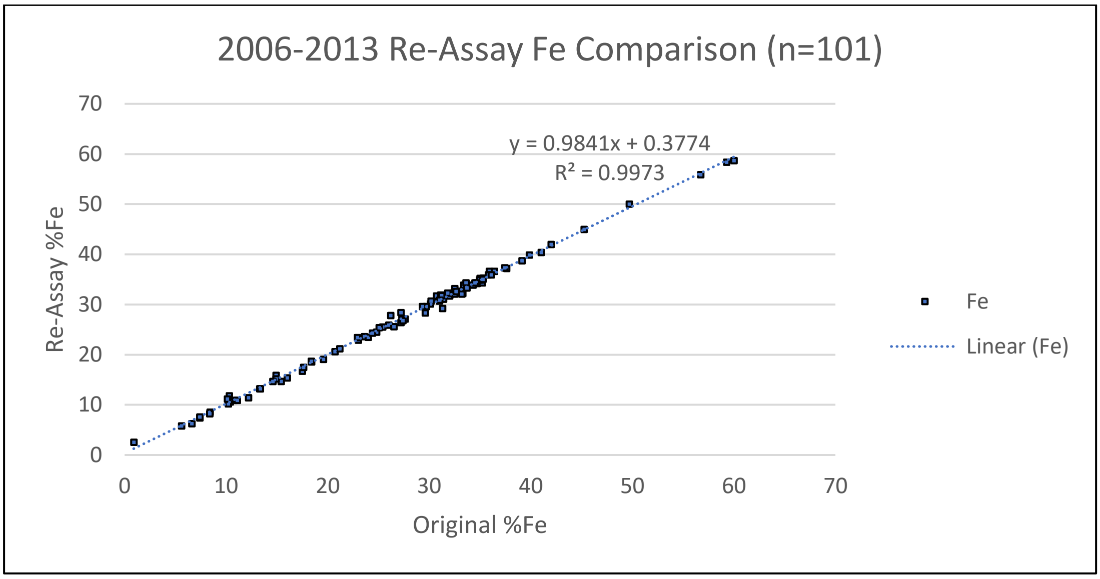


Figure 11-13: Scatterplot, Original (Amdel) Assays vs Pulp Repeats, Fe %

Source: Macarthur (2020)

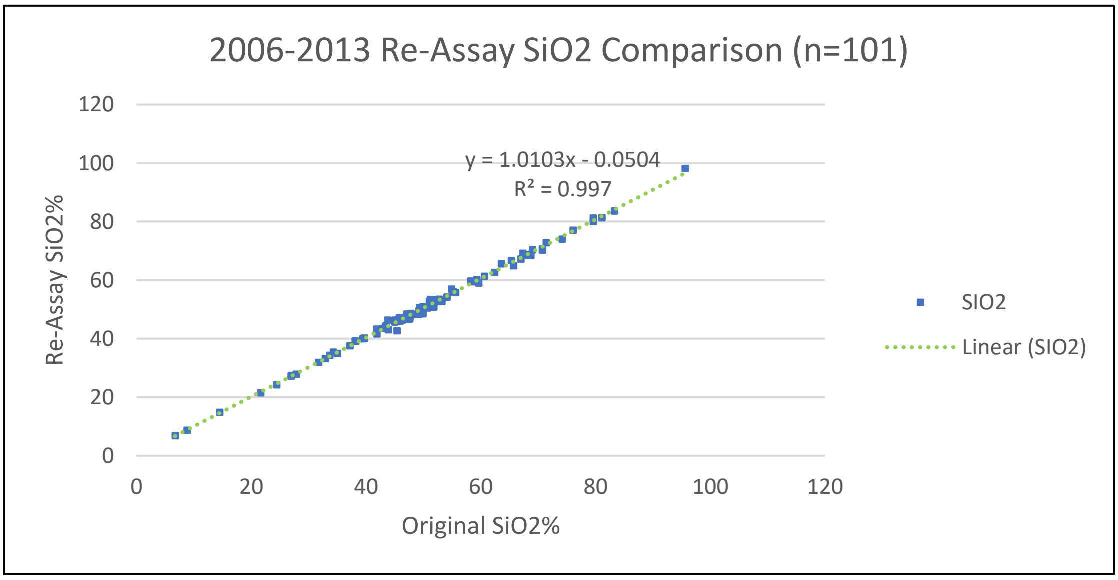


Figure 11-14: Scatterplot, Original (Amdel) Assays vs Pulp Repeats, SiO2 %

Source: Macarthur (2020)

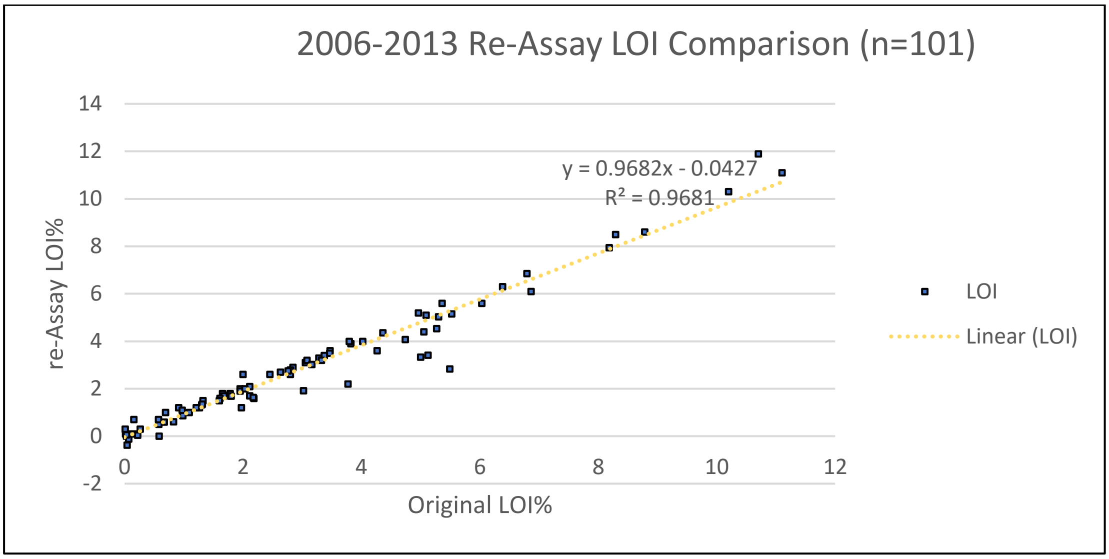


Figure 11-15: Scatterplot, Original (Amdel) Assays vs Pulp Repeats, LOI %

Source: Macarthur (2020)

11.3.7. Umpire Assay Results

The 2019 Moonshine drilling campaign sent a total of 148 samples to Intertek Laboratory in Perth for umpire testing of head XRF grades.

Samples for umpiring were selected by Macarthur and requested as pulps from the bulk samples held at SGS, then delivered to Intertek. Intertek assayed the samples using their FB1/XRF (lithium borate fusion) method for elements and TGA method for LOI.

Figure 11-16 shows a scatterplot of Fe % from the umpire analyses. Results from this and other elements showed no grade bias towards either laboratory and therefore support the use of the SGS sample analyses in the Mineral Resource estimate.

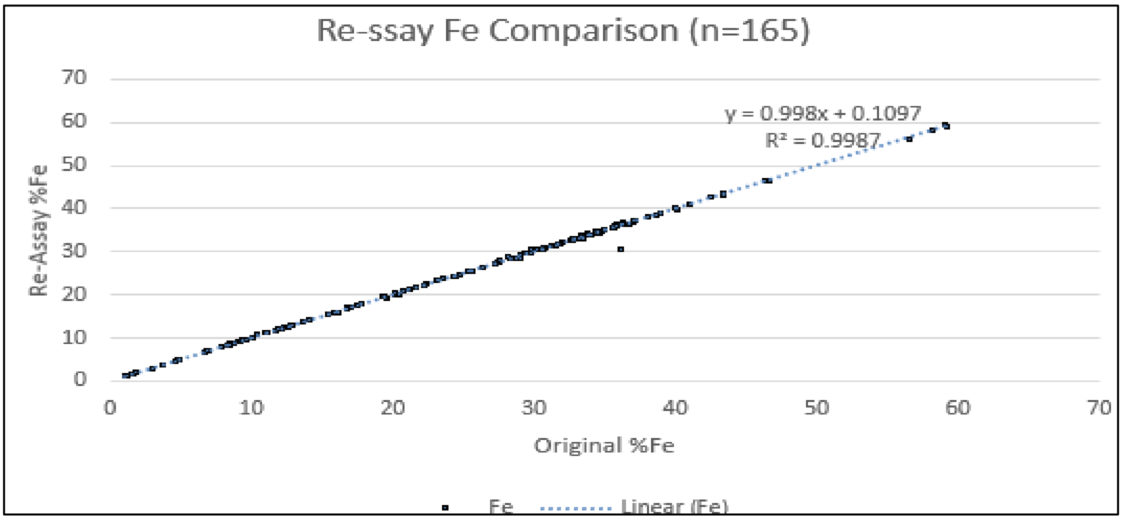


Figure 11-16: Scatterplot of Fe %, SGS vs Intertek Analyses, Moonshine 2019 Drill Program

Source: Macarthur (2020)

11.3.8. Laboratory Internal Testing

As part of their normal analytical operations, laboratories often perform internal duplicate testing of splits from the bulk 105 µm sample as a means of testing the XRF apparatus. Certain laboratories also include their own standards and blanks on a regular basis and include the results in the results being sent to the client. Analysis of all laboratory testing suites for all laboratories used over the lifetime of the Project have shown excellent consistency and have not raised any issues of concern for Macarthur.

11.4. Qualified Person's Opinion

The author is of the opinion that the sample preparation, sample security and analytical procedures are of industry standard and are adequate to support the Mineral Resource classification disclosed in this report.

12. Data Verification

12.1. Site Inspection

Mr Nikolay Karakashov, independent contract geologist to Macarthur, visited the property between 9 and 10 September 2020 in the company of Dr Dean Carter, General Manager, Macarthur. Mr Karakashov was also on site in 2021 (6-19th Sep and 29th Sep-10th Oct), to assist with and observe the geotechnical drilling program at Moonshine While on site, Mr Karakashov inspected the overall geology of the Project including outcropping magnetite mineralisation of the Moonshine, Moonshine North, Sandalwood, Clark Hill North, Clark Hill South, and Snark deposits. Representative drill core and RC chips of mineralised intervals from the deposits were inspected. Multiple drillhole locations were visited and collar coordinates for 28 drillholes were surveyed with a handheld Garmin GPS device, with an accuracy of ± 3 m on the GDA94 grid system. In all cases, the surveyed collar coordinates were confirmed. Some historical collar locations were only estimated, due to the extensive rehabilitation of the drill sites, as seen in Figure 12-1 and Figure 12-2. In all cases, the surveyed collar coordinates were confirmed.

Mr Karakashov also appraised the local infrastructure including the quality of access to the Project site, and the proximity of the Project to adjacent properties hosting advanced projects.

Table 12-1 shows the results of hole location checking, showing a good average error range.

There were no negative outcomes from the site inspection.

Table 12-1: Collar Coordinate Location Checking 9–10 September 2020

Hole ID	Measured east	Measured north	Database east	Database north	Deviation distance
LGRC_0027	787,769	6,693,914	787,765	6,693,913	1.3
LGRC_0032	787,584	6,692,501	787,583	6,692,500	0.8
LGRC_0082	791,371	6,687,998	791,368	6,687,997	1.4
LGRC_0102	789,185	6,691,476	789,182	6,691,470	6.3
LGRC_0103	789,174	6,691,687	789,170	6,691,686	1.2
LGRC_0104	789,134	6,691,917	789,133	6,691,913	4.2
LGRC_0105	790,119	6,672,306	790,123	6,672,306	0.3
LGRC_0113	789,376	6,673,092	789,378	6,673,100	8.3
LGRC_0199	790,755	6,671,360	790,756	6,671,362	2.3
LGRC_0203	787,982	6,674,757	787,980	6,674,758	0.8
LGRC_2148	790,085	6,672,302	790,086	6,672,302	0.6
LGRC_2152	790,346	6,671,769	790,345	6,671,765	4.4
LGRC_2165	787,888	6,674,858	787,894	6,674,856	1.3
LGRC_0225	787,945	6,675,127	787,949	6,675,123	3.7
LGRC_0236	787,967	6,675,138	787,971	6,675,134	4.1
LGRC_0266	791,039	6,671,116	791,042	6,671,116	0.3
LGRC_0271	787,737	6,675,301	787,738	6,675,296	5.0
LGRC_0273	787,645	6,675,605	787,647	6,675,600	4.2
LGRC_0368	791,601	6,687,199	791,598	6,687,199	0.3
LGRC_0431	787,950	6,674,752	787,948	6,674,749	2.9
LGRC_0084	791,104	6,688,759	791,100	6,688,755	5.2

LGRC_0088	790,753	6,689,573	790,750	6,689,568	4.8
LGDD_066	790,221	6,672,153	790,218	6,672,154	0.4
LGDD_071	787,936	6,674,893	787,942	6,674,887	5.6
LGWE_013	788,043	6,674,578	788,041	6,674,575	2.4
18MNRC001	788,030	6,674,942	788,035	6,674,937	4.7
LGWE_042	791,169	6,690,791	791,175	6,690,795	4.6
LGWE_043	791,107	6,690,742	791,105	6,690,738	3.4
				Average	3.0

Figure 12-1: Location of LGRC_0038 in Clark Hill South (showing drill cuttings) at estimated collar location

Figure 12-2: An example of rehabilitation extent at a historical drill site (LGRC_0021) in Clark Hill North

Mr Karakashov was involved in the 2010 and 2011 RC and diamond drilling campaigns and 2021 geotechnical drilling at the Moonshine and Moonshine North prospects. Mr Karakashov was satisfied with drilling, sampling and QAQC practices at the time. Sample quality was predominantly satisfactory, with any sample recovery issues dealt with immediately. Majority of the samples obtained within the mineralised domains were of good quality and consistency.

Planned drillholes orientations were also raised as an issue in 2011, due to the strongly varying magnetic field direction in the vicinity of major magnetite BIF ridges, sometimes causing deviation of the north direction by over 40°. Mr Karakashov proposed an alternative procedure for lining up drill rigs proximal to magnetite BIF ridges by pegging sighter pegs using a GPS device and taking a back-bearing located at least 200 m away from the drillhole to minimise GPS error to within 5°. A second sighter peg was then placed at the drill site. The procedure was adopted for all affected areas thereafter. Drilling orientation prior to this procedure did not pose an issue as all drillholes were later surveyed with a gyro tool, superseding any handheld compass orientations. Drilling also remained closely perpendicular to the BIF ridges with all drillhole orientations estimated on BIF outcrop orientation, as opposed to pre-planned cardinal directions.

12.2. Data Verification and Validation

12.2.1. Sample Dispatch, Handling and Data Collection

Sample collection, handling and dispatch was of a high standard, with good practices employed throughout the process. Security tags were used at all steps of sampling and dispatch through to delivery at the relevant testing labs.

Prior to 2019, sampling data was stringently collected at all steps of the process and logged on paper with subsequent validated data entry into a secure relational database package, maintained by Macarthur staff. The operational database then exported packages of data, which were validated and entered by CSA into Macarthur’s database, fully maintained and operated by CSA Global. Exports of the data were then supplied to Macarthur and checked by field staff, when relevant. A summary of these procedures can be seen in Figure 12-3.

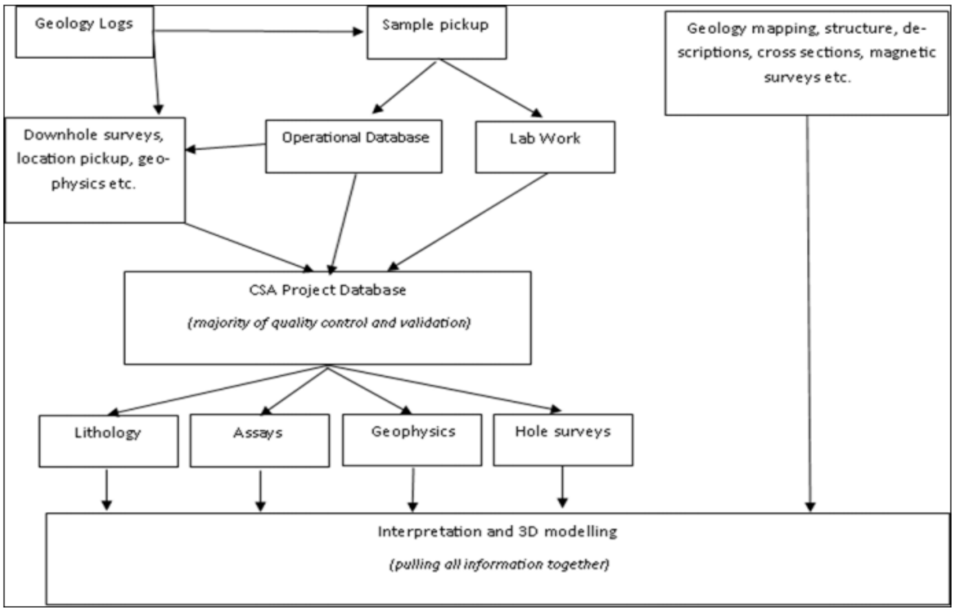


Figure 12-3: Diagrammatic summary of data management at Lake Giles Iron Project prior to 2019

Source: CSA Global (2020)

For the 2019 drilling campaign at Moonshine and Moonshine North, sampling, dispatch and data generation was done entirely by Macarthur staff and contractors. Detailed procedures for drilling, sampling and collection of data were provided by field supervisors and appear to have been followed to a satisfactory level. Drilling, logging and sampling data was provided in digital format as a series of spreadsheet templates, which were collated by Mr Karakashov. The field data then underwent stringent quality control, utilising a variety of industry standard techniques for verifying exploration data before being imported into a relational database, constructed and maintained by Mr Karakashov.

A small number of corrections to original field data (e.g., fixing typographical errors, incorrect dates of sample collection, incorrect sample ID assignments) were performed and fully logged into a separate document, for future reference. Any other data was entered directly into the database with no alteration. Copies of the original field data were also stored and reviewed by the Project geologists.

The Lake Giles Iron Project currently has two separate databases for exploration, one for pre-2019 data, and the other capturing data from the 2019 drilling program. The 2019 drilling and sampling data was stored in a unique database to manage QAQC protocols and correcting any errors in the database, without affecting the pre-2019 database which was validated and deemed fit for use to support the previous non-current Mineral Resource estimates for the Lake Giles Iron Project.

The two databases contain compatible data, allowing merging of the database tables at the Mineral Resource estimate stage, as discussed in Section 14.3.1. It is recommended that Macarthur merge both databases with associated database validation and data security procedures prior to future updates to the Mineral Resource estimates.

12.2.2. Laboratory Analyses and QAQC

Prior to 2019, all assays and QAQC data associated with the Lake Giles Iron Project was managed by CSA as part of their maintenance of the Project database. As such, all relevant industry standard quality controls and data aggregation methods were employed and applied on incoming data. Macarthur staff were also supplied with both raw data from lab dispatches, as well as exported assays, which were then widely used in internal geological modelling, as well as cross referencing to original field data (e.g., in cases of sample duplicate IDs or expected sudden changes in grade). Assay data was deemed of good quality and no major issues were raised at the time.

The 2019 drillhole database incorporates laboratory analyses and QAQC results from the 2019 drilling campaign, and also includes the 30 drillholes drilled up until 2010, as well as laboratory repeats from 26 holes drilled prior to 2019. QAQC data included CRMs, duplicate and blank testing, lab umpiring comparison, as well as internal laboratory tests as detailed in Section 0.

Data manipulation of primary data was minimal and aimed at converting non-numeric results into useable numeric values. Actions included:

- Converting all below detection limit (usually represented as negative) values to 0 or half of the lower detection limit
- Assigning values of -9999 for missing data, – 8888 for unreported data, – 5555 for insufficient sample quantity and – 1111 for over detection limit samples. The values could then be excluded from any estimations and analysis. This data manipulation step was applied to the pre-2019 drillhole database; and
- Conversion of non-assayed values to null.

Laboratory results from SGS and Intertek (as part of umpiring, discussed in Section 11.3.7) was sent directly to Macarthur and was subsequently verified before being included in the database. Several sample dispatches from SGS showed inconsistencies with some of the calculated values returning erroneous results. The batches in question were returned to SGS and promptly rectified, leaving no outstanding data issues relating to assay results.

Upon receipt of the entirety of QAQC data Macarthur noted several mismatches between expected and returned values for a handful of CRM samples. Due to their close match to other CRMs used in the program and the high apparent quality of testing on SGS's part, a decision was made to amend the database to reflect these changes and reassign the CRM IDs to the expected ones. A record of these changes was maintained in the relevant spreadsheet. It is likely the error was caused by inserting the incorrect CRM packet in the field by the field assistants. All remaining QAQC data received and verified by Macarthur was of adequate quality to support Mineral Resource estimates.

No drillholes were excluded from the Mineral Resource estimate.

12.2.3. Twin Drilling

A total of two diamond drillholes in Moonshine North partially twinned existing RC holes. Twinning was planned for the purpose of increasing geological confidence in creating metallurgical sample composites from the two prospects without sacrificing extra core, as well as verifying the consistency of downhole geology across short distances.

The twinned hole pairs were LGRC_0276 with LGDD_052 for the first 50 m and LGRC_0222 with LGDD_005 for the first 54 m. The twinned sets intersected the footwall and hanging wall contacts of the Moonshine North east 1 lode bearing BIF respectively and were located approximately 100 m apart along strike of the main BIF unit.

Limited assay data is available for the LGDD_052 twinned interval, as only a few samples were selected for assaying by the supervising geologist. The assays indicate close correlation associated with the rapid decrease in iron grade at the footwall contact. A simple comparison can be seen in Figure 12-4 to Figure 12-5.

No assay data is available for LGDD_005 as the core was composited into a larger metallurgical sample, with no metre-scale assaying being performed.

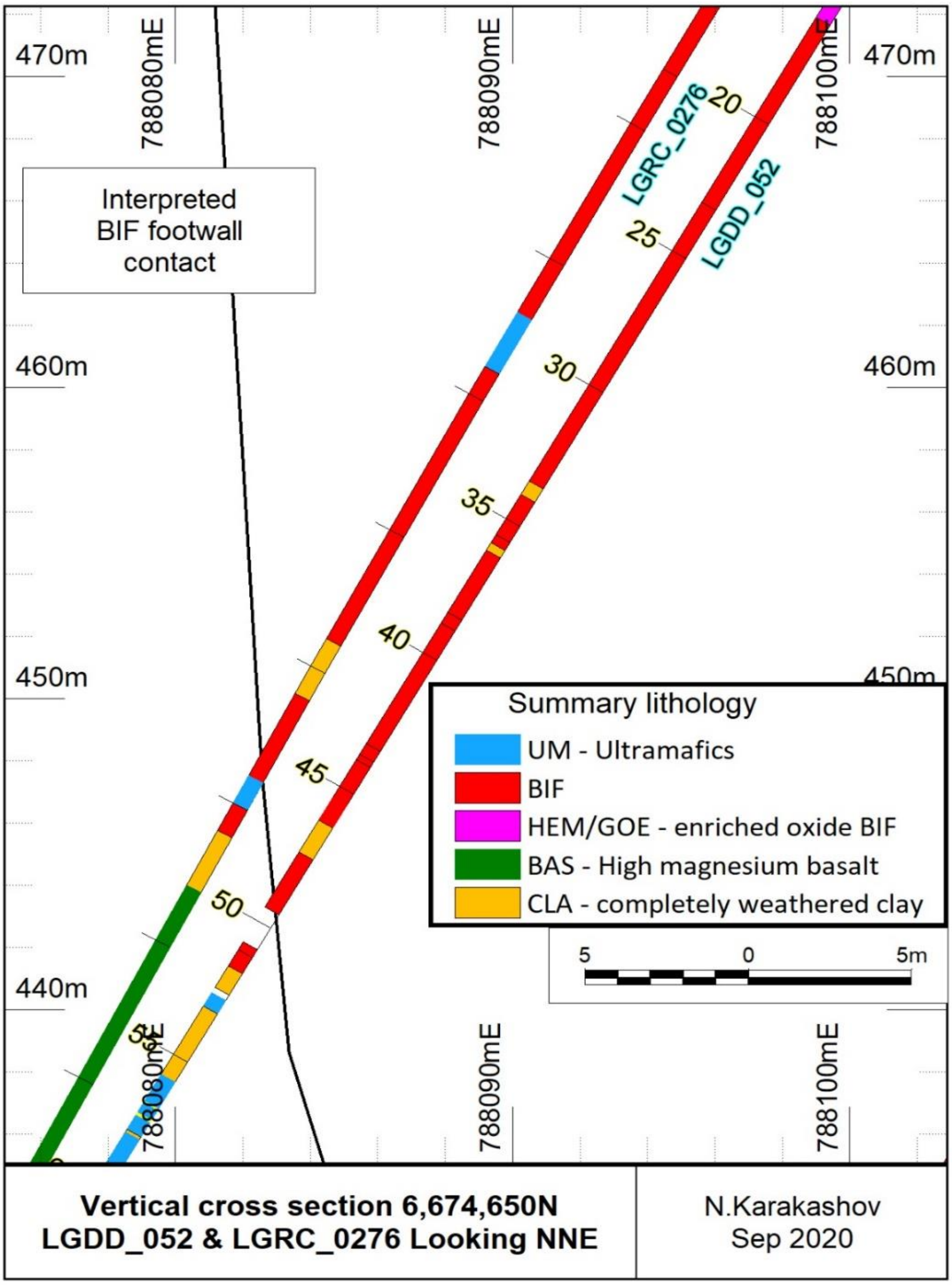


Figure 12-4: Lithological logging for LGDD_052/LGRC_276 Pair

Source: Karakashov (2020)

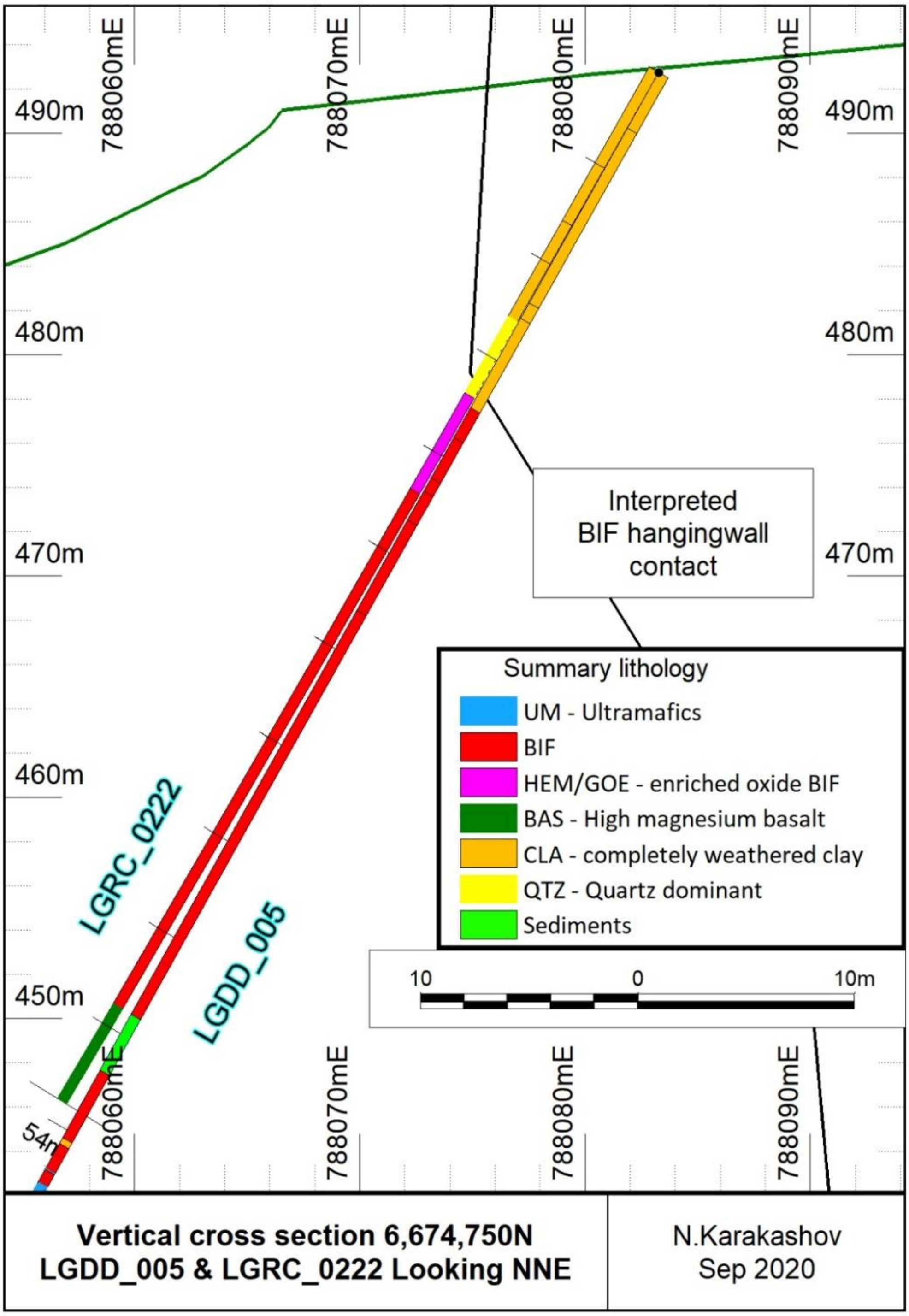


Figure 12-5: Lithological logging for LGDD_005/LGRC_222 Pair

Source: Karakashov (2020)

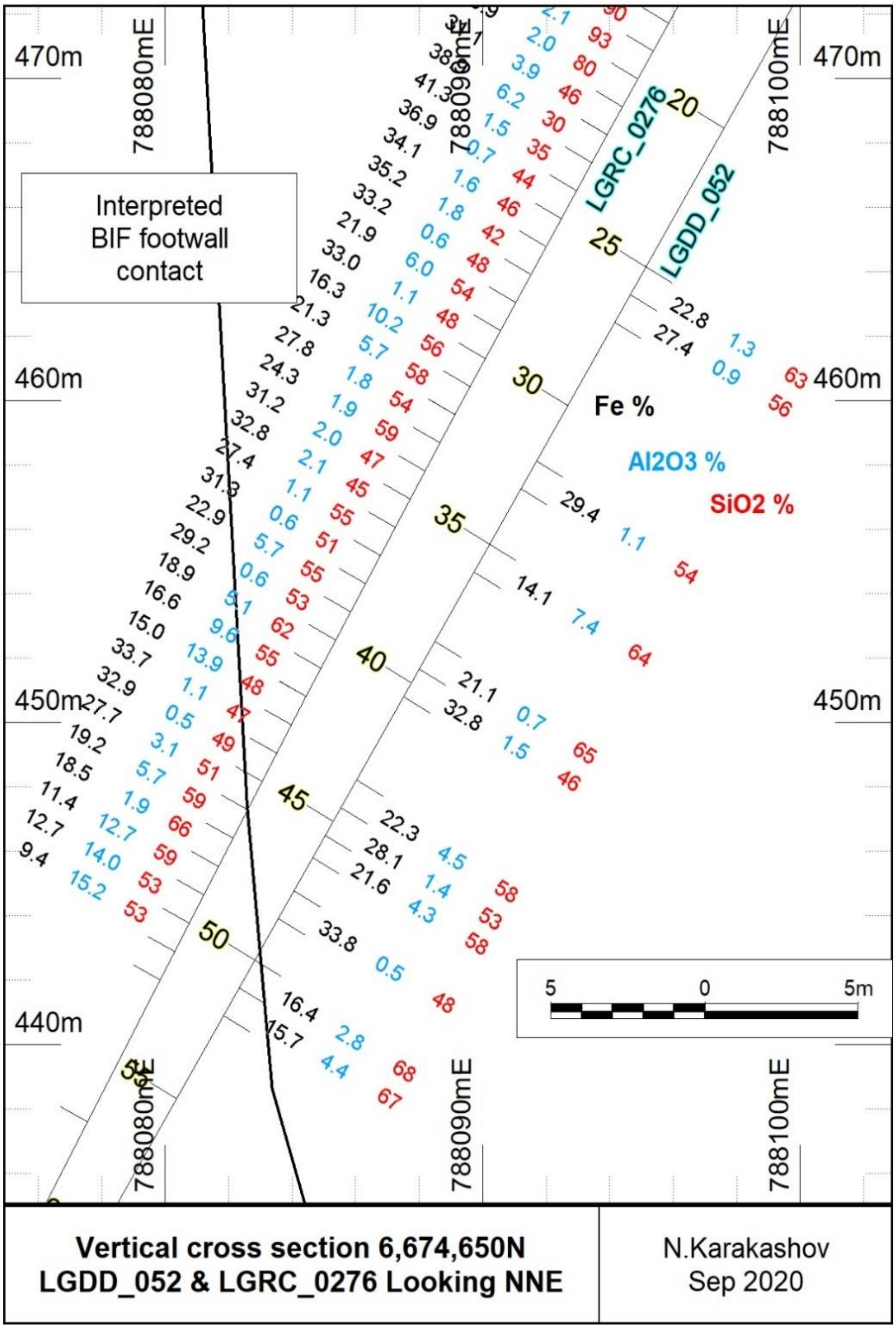


Figure 12-6: Assay comparison for LGDD_052/LGRC_276 Pair

Source: Karakashov (2020)

12.2.4. Audits and Reviews

No independent audits or reviews of the drillhole database and QAQC results have been carried out, apart from current and previous reviews of data conducted by the QPs at the time of reporting of Mineral Resources.

12.2.5. Opinion of Qualified Person

The QP is of the opinion that the drillhole and sample data is adequate for use in the Mineral Resource estimates disclosed in this report.

13. Mineral Processing and Metallurgical Testing

13.1. Previous Metallurgical Testwork

Engenium (2010) carried out preliminary studies based on samples from two RC holes (LGRC199 and LGRC203); one each from the Moonshine and Moonshine North deposits. The location of these holes is shown below.

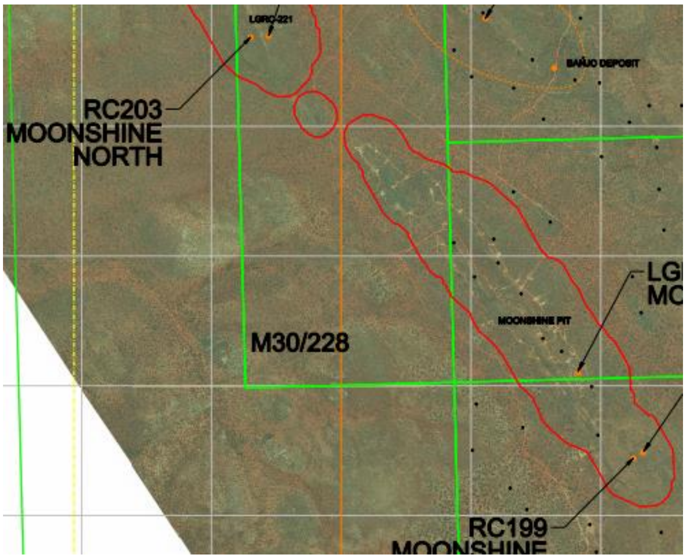


Figure 13-1: RC203 (Moonshine North) and RC199 (Moonshine)

The main conclusions were:

- The iron head grades from the metallurgical test samples and the DTR concentrate grade were higher than the bulk of the intervals used for the Mineral Resource estimate (Snowden, 2011).
- The Low Intensity Magnetic Separators (LIMS) test results yielded a poorer quality concentrate than was determined from the DTR preliminary analysis. The reason for this is unknown.
- DTR concentrate grades for silica from both holes were ~ 5%; however, the LIMS test did not achieve this grade in hole LGRC199.

13.2. Magnetite Metallurgical Testwork

13.2.1. Introduction

The full laboratory testwork report is detailed in document LGI01-EN-20000-A-R-0004. This report is summarised below.

The previous testwork encouraged the latest recent drilling programme to assess the deposits in some detail. The drilling programme collected HQ sized core and split the core for assays and Davis Tube testwork. Half core was available for testing. The drillhole locations are shown as green in Figure 13-2.

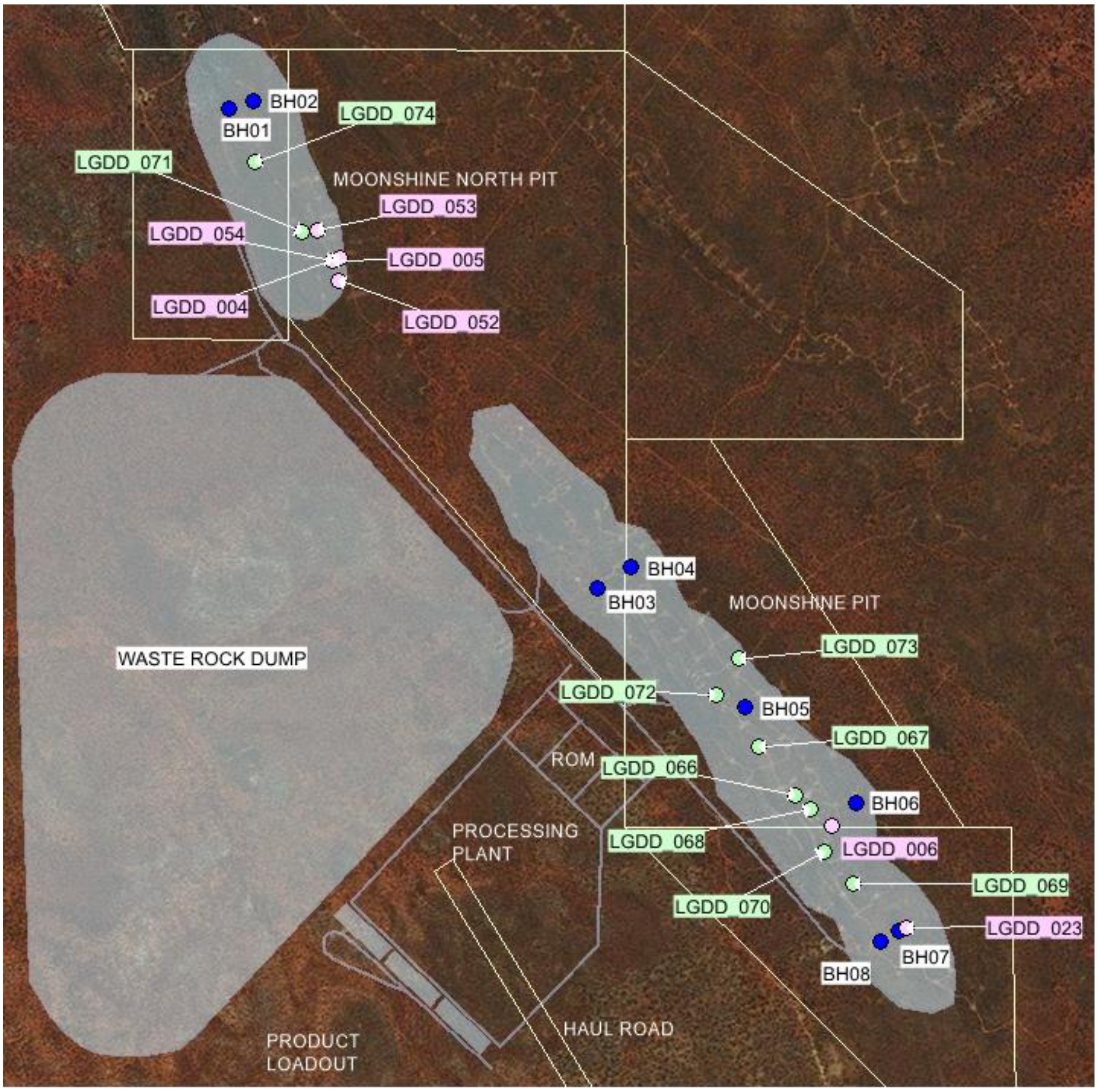


Figure 13-2: Magnetite testwork core drillhole locations (green) against preliminary site layout.

The testwork was performed at the Bureau Veritas (BV) laboratory in Canning Vale, Western Australia, an ISO9001 certified Laboratory. BV Staff laid out the core trays, for inspection by MIO and Engenium personnel, to select composites for testwork. Indicative photos of the core are overleaf as Figure 13-3.

As core was limited, sample selection focussed on maximising the inclusion of mineralised ore, whilst also including diluting intervals not rejectable by selective mining. The interval considered was a half bench height of six metres.

There was no geological model available to nominate any variation in the ore deposit structure. There was also no mine plan to guide a compositing strategy based on mining over time. The composite details are provided Table 13-1.

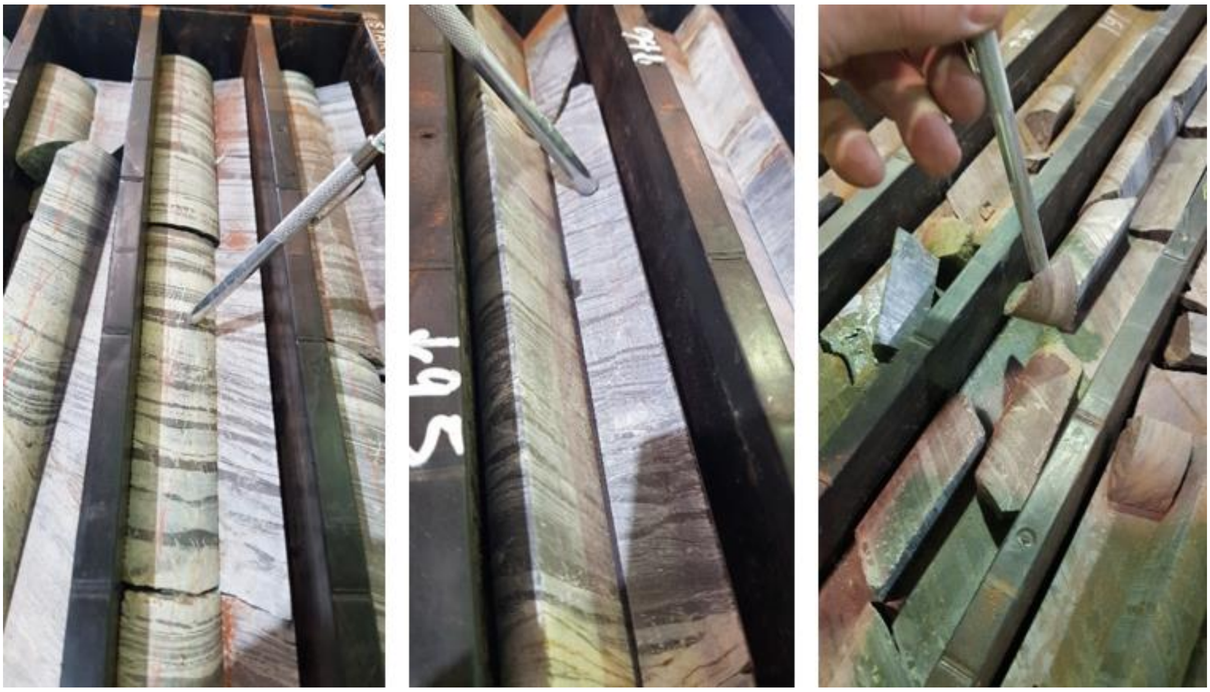


Figure 13-3: Indicative core photographs

Table 13-1: Testwork composite details

Prospect	Hole Identification	Core Selected at BV			Sample Mass (kg)
		Start	End	m	
Moonshine	LGDD_006	144.0	265.0	121.0	965
	LGDD_066	83.7	165.3	81.6	407
	LGDD_067	69.0	135.6	66.6	332
	LGDD_068	83.0	193.5	110.5	551
	LGDD_069	88.0	115.0	27.0	135
	LGDD_070	88.0	132.8	44.8	223
	LGDD_070	143.0	152.4	9.3	47
	LGDD_070	166.0	173.5	7.5	37
	LGDD_072	56.3	117.2	61.0	304
	LGDD_073	110.4	140.9	30.6	152
	LGDD_073	200.0	269.7	69.7	347
	LGDD_023	101.1	198.7	97.6	973
Total				**630**	**4473**
Moonshine North	LGDD_071	81.8	162.0	80.2	400
	LGDD_074	47.1	71.0	23.9	119
	LGDD_074	80.7	98.9	18.2	91
Total				**122**	**610**

13.2.2. Testwork

There were two test plans developed, one for magnetic separation and one for high pressure grinding rolls (HPGR) testwork. The test plans are detailed in the standalone metallurgical test work summary report

13.3. Head Assays

The composite head assays for Moonshine and Moonshine North are below.

Table 13-2: Composite head assays

Composite	Fe	SiO_2	Al_2O_3	P	S	LOI-1000
	%	%	%	%	%	%
Moonshine Actual	30.7	50.3	0.37	0.04	0.56	-0.12
Moonshine Expected	30.1					
Moonshine North Actual	32.8	47.4	1.03	0.05	1.21	0.68
Moonshine North Expected	32.7					

Note: Hole 23 assays not included in Moonshine calculations, as they were unavailable.

The Moonshine composite sample head assay is a reasonable comparison to the resource head assays.

The Moonshine North composite sample head assay is of higher grade in comparison to the resource head assays.

The testwork results, especially yield, would need reviewing with this in mind before being utilised as design inputs.

The high proportion of Inferred resources at Moonshine North is also a factor.

13.4. Comminution

The Bond Abrasive Index (Ai) as determined for each of the composites is shown Table 13-3.

Table 13-3: Bond abrasive index

Composite	Abrasive Index
Moonshine North	0.5792
Moonshine	0.5285

The Moonshine and Moonshine North Ai values are high and indicate the need for specialised design to combat wear in items such as chutes and bins.

Specific engineering solutions will be required which will be significant enough to affect capital expenditure, and the operating costs would be higher, due to increased maintenance requirements.

Each composite's Bond Ball Mill Work Index (BWI) was developed at 75 and 125 µm, below.

Table 13-4: Bond Ball Mill Work Index Composite 75 µm, 125 µm

Composite	Size (µm)	BWI (kWh/t)
Moonshine	75	13.5
Moonshine	150	13.5
Moonshine North	75	14.9
Moonshine North	150	14.9

The BWI classifies these samples as hard, an observation that confirms field reports of hard drilling and the test results above. However, given that the hard rock mining industry treats this type of ore, these values would present few problems to processing.

A BWI was determined for each composite's dry LIMS magnetic product at 106 µm.

Table 13-5: Bond Ball Mill Work Index – Dry LIMS Mags 106 µm

Composite	Size (µm)	BWI (kWh/t)
Moonshine	106	12.8
Moonshine North	106	13.3

The energy requirement reduced a minor amount in removing some host rock.

The major results from the SMC Tests are below.

Table 13-6: SMC test results

Parameter	Moonshine	Moonshine North
M_{ia} (kWhr/t)	19.9	19.1
M_{ic} (kWhr/t)	8.1	7.7
M_{ih} (kWhr/t)	15.6	14.9
t_a	0.30	0.30
A*b	37.6	38.7

The hardness of the ores is inversely proportional to the A*b values. The measured values, while classified as hard, are in the typical range of ores treated by the mining industry. The A*b values report in the top 66.7% and 63.9% of values ever reported and the t_a values report as similar typical values. The entire A*b database of many hundreds of values is shown below, with the Lake Giles values shown as vertical lines.

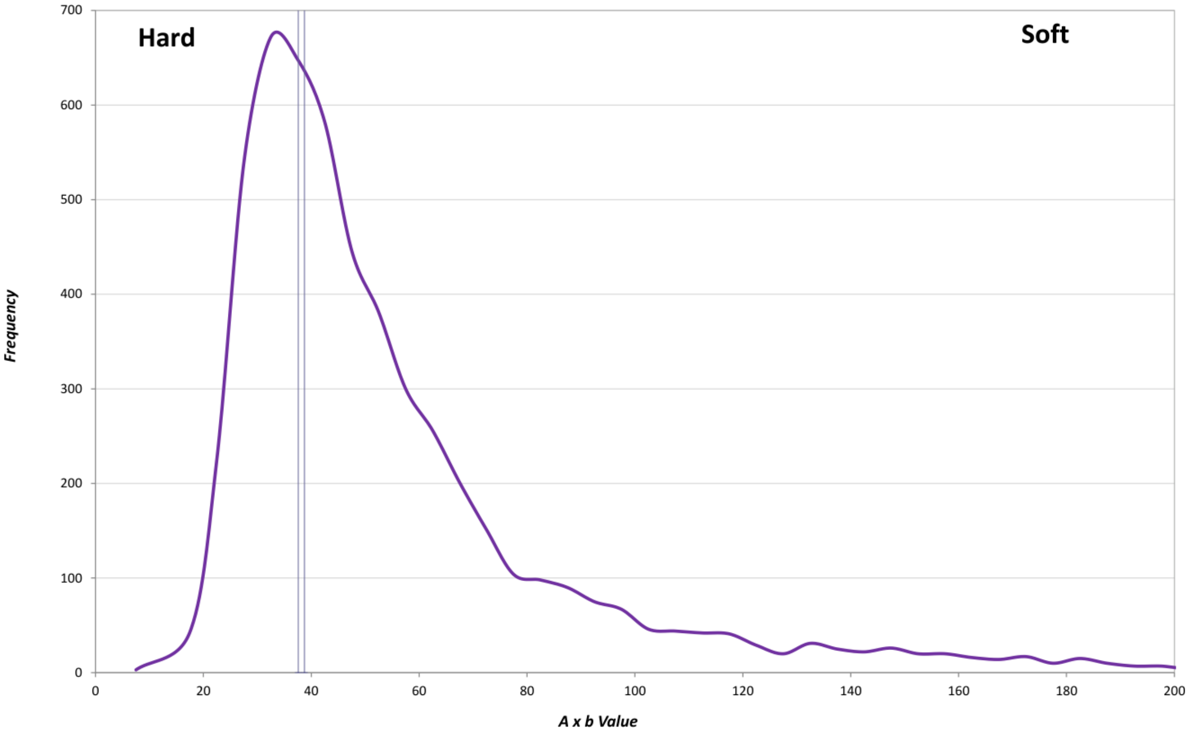


Figure 13-4: Lake Giles Ore A*b values relative to database

While the A x b values are in the mid 60's in percentile reporting (66.7% and 63.9%) the t_a values are both at 77.1%, again showing the abrasive nature of the hard ore.

13.5. In-Situ Specific Gravity

Determination of the in-situ Specific Gravity (SG) of the dry solids formed part of the SMC testwork.

Table 13-7: Specific gravity results

Composite	Sample Origin	SG
Moonshine	SMC Testwork Sample	3.29
Moonshine	Composite	3.46
Moonshine North	SMC Testwork Sample	3.24
Moonshine North	Composite	3.46

These SG values are within normal bounds to be reconciled with the field readings.

Concentrate Bulk Density

Determining the concentrate bulk density (BD) on a small sample provided a parameter for the bulk material calculations going forward. The results are below.

Table 13-8: Concentrate Bulk Density

Composite	Unconsolidated BD (t/m^3)	Consolidated BD (t/m^3)
Moonshine	1.88	2.39
Moonshine North	1.95	2.48

13.6. Asbestiform Analysis

A mineralogist examined the composite head samples for the presence of fibrous mineralisation. The mineralogist did not detect any asbestos like fibres in either sample.

13.7. Davis Tube Wash Investigation

A Davis Tube Wash (DTW) was performed on the each of the composite samples at a range of sizes between 12 µm and 218 µm, in order to determine the recovery and grades achievable. A summary of the results is below. The size fraction to achieve a 65% Fe concentrate is highlighted in each table.

Moonshine

Table 13-9: Moonshine Davis Tube Wash Results

P_{80} (µm)	Yield (%)	Fe (%)	SiO_2 (%)	S (%)
215	60.9	45.6	32.8	0.38
151	55.7	50.4	27.4	0.39
71	48.0	56.7	19.3	0.39
56	47.4	57.7	18.0	0.38
38	43.7	61.9	12.7	0.41
28	42.5	65.7	8.0	0.43
12	39.1	70.0	2.5	0.46

Moonshine North

Table 13-10: Moonshine North Davis Tube Wash Results

P_{80} (µm)	Yield (%)	Fe (%)	SiO_2 (%)	S (%)
218	56.9	48.6	28.3	0.95
142	49.8	53.9	22.4	0.87
74	43.8	59.8	14.6	0.77
55	42.6	62.5	11.4	0.79
40	40.9	65.0	8.5	0.78
24	38.3	67.3	5.4	0.74
12	37.3	70.2	1.9	0.83

The plant design was progressed to produce the final concentrate at a sizing P80 of 38 µm. It was expected that desilification flotation would reduce the silica to the required grade at this grind size, with no further comminution required.

However, the sulphur content, even with finer beneficiation, remained above the generally desired 0.1% for a saleable magnetite concentrate. Following confirmation that the testwork samples were likely representative of the main ore body, for sulphur, further testwork was prompted. This sulphide removal testwork is future work. Satmagan measurements for the head, magnetics and non-magnetics of each DTW test determined the magnetite grade and recovery. The following tables summarise the concentrate analyses.

Moonshine

Table 13-11: Moonshine Magnetite Grade and Recovery

P_{80} (µm)	Yield %	Magnetite Grade %	Magnetite Recovery %
15	39.1	90.4	99.2
25	42.5	86.4	98.9
38	43.7	81.0	98.9
53	47.4	75.5	98.4
75	48.0	74.0	98.4
150	55.7	64.8	98.5
212	60.9	58.2	98.8

Moonshine North

Table 13-12: Moonshine North Magnetite Grade and Recovery

P_{80} (µm)	Yield %	Magnetite Grade %	Magnetite Recovery %
15	37.3	82.4	98.7
25	38.3	80.1	98.4
38	40.9	78.0	98.4
53	42.6	74.9	98.4
75	43.8	69.8	98.2
150	49.8	61.6	98.1
212	56.9	54.5	98.5

These magnetite recoveries are favourable and show that the magnetic components of the ore recover well, at all sizes, and the reduction in size improves the grade. This is a conventional trend. It shows that the magnetic minerals process conventionally with minimal losses due to mineralogical reasons.

13.8. Dry Low Intensity Magnetic Separation Testwork

The dry low intensity magnetic separation (LIMS) testwork was performed at -6.3 mm, for each composite sample, on a laboratory scale dry LIMS unit at 1100 Gauss. This unit runs at a maximum of 91 rpm, resulting in a peripheral velocity of 1.5 m/s. Two tests were performed, one at 80% speed and the other at maximum velocity. Satmagan tests were performed on both speed samples and a full assay for the dry cobbing products for both composites from the maximum speed test. The results are as follows.

Table 13-13: Dry LIMS Testwork Velocity vs Yield and Recovery

Peripheral Velocity (m/s)	Mags Yield %	Non-Mags Yield %	Magnetite Recovery %
1.5	83.4	16.6	98.5
1.14	85.4	14.6	99.0

Based on these results, the bulk dry LIMS testwork was performed at the maximum drum speed, corresponding to 1.5 m/s peripheral velocity.

The plant flowsheet was modified to treat the +3 mm, -12 mm (3x12 mm) material in the HPGR circulating load, after the testwork programme had started. During subsequent testing, it was apparent that more crushed sample would be needed for sulphide flotation testwork. In preparing this material, BV was asked to closed-circuit crush the first HPGR test run to -3 mm, treating the 3x12 mm material in the dry LIMS unit.

As the first dry LIMS pass would be the most representative of a treated stream, its assay and yield along with the mass recoveries seen at each grind out stage were the basis for determining the mass balance dry LIMS recovery. The test assays are given below.

Table 13-14: Closed Circuit HPGR Testwork Assays

Stream	Yield	Fe	Fe_3O_4	SiO_2	P	LOI $_{650\text{-}1000}$	LOI $_{1000}$
	%	%	%	%	%	%	%
HPGR -3.0 mm	100	33.0	36.5	48.2	0.047	0.29	-0.23
HPGR First Pass Mags	15	34.4	35.0	47.0	0.050	0.17	-0.45
HPGR Non-Mags	85	18.2	15.0	65.1	0.030	0.83	1.17

The overall dry LIMS reject to non-magnetics decreases in treating the reduced size range, as the liberated gangue in the -3 mm fines is not subject to treatment.

13.9. Low Intensity Magnetic Separation Test Programme

As LIMS is the primary beneficiation process, both the Moonshine and the Moonshine North composites were tested. The work followed the original three-stage wet LIMS flow sheet, as the flowsheet changes trailed the testwork programme start. The flowsheet was subsequently modified to two-stage wet LIMS following the dry LIMS (i.e., Rougher LIMS and Cleaner LIMS only) but the programme had progressed to allow this change. The operational conditions for each stage are summarised Table 13-5.

Table 13-15: LIMS Testwork Conditions

Process	Feed Size P80	Flux	Passes
Stage	**(μm)**	**(G)**	**(number)**
Dry LIMS	6,000	1,100	Single
Rougher LIMS	212	1,100	Single
Cleaner LIMS	106	1,100	Double
Re-Cleaner LIMS	38	1,100	Double

The composites performances are discussed below.

Moonshine

The Moonshine sample LIMS testwork gave a higher yield but at a lower grade than was expected from previous work. This makes the subsequent desilification stage critical.

The high recoveries of the Fe_3O_4 and FeO assays show the separation is very good. Unfortunately, this assay includes all the ferrous ions, including minerals based on iron sulphide as well as the magnetite, so may be over-emphasising the result.

Table 13-16: Moonshine LIMS Testwork Results Summary

	Magnetic Stream %							
Process Stage	**Yield**	**Fe_3O_4**	**FeO**	**Fe**	**SiO_2**	**Al_2O_3**	**P**	**S**
Feed	100.0	35.0	3.0	31.3	50.2	0.39	0.048	0.52
Dry LIMS	83.9	41.1	3.6	35.2	45.9	0.19	0.052	0.36
Rougher LIMS	66.5	49.7	19.7	42.5	37.0	0.14	0.043	0.26
Cleaner LIMS	51.8	62.6	23.8	52.3	24.7	0.10	0.030	0.22
Re-Cleaner LIMS	43.6	77.8	27.0	61.3	13.6	0.05	0.020	0.19
Final Recovery								
Re-Cleaner LIMS	43.6	97.0	95.5	85.3	11.8	5.6	18.3	16.1

While the sulphur assay reduces with progressive treatment, it does not reach the usual <0.1% expectation for iron ore products.

Some marketing advice will be needed to address this, and a desulphurisation stage added if required. This will be the purpose of a small study outside of this study programme.

The non-magnetic streams were assayed, including Satmagan Magnetite assays to ensure that there were no excessive losses in the testwork. Results are presented in Table 13-17.

The magnetite content is low, so no excessive valuable mineral losses were found.

Table 13-17: Moonshine LIMS Testwork Non-Magnetic Assays

	Non-Magnetic Stream %						
Process Stage	**Yield**	**Fe_3O_4**	**Fe**	**SiO_2**	**Al_2O_3**	**P**	**S**
Dry LIMS	16.1	3.0	11.2	72.4	1.42	0.025	1.4
Rougher LIMS	17.3	1.4	6.8	79.1	0.29	0.074	0.58
Cleaner LIMS	14.7	2.1	6.7	80.5	0.29	0.089	0.39
Re-Cleaner LIMS	8.2	1.5	6.3	83.0	0.29	0.094	0.36
Wet Tail	40.2	1.68	6.67	80.4	0.29	0.08	0.47
Total Tails	56.4	2.05	7.98	78.1	0.61	0.07	0.72

Moonshine North

The Moonshine North LIMS testwork gave a lower yield, at a higher grade, than the Moonshine LIMS testwork but this is expected to be a function of the improved head grade. The sample performed as expected; that is the sample would not be able to meet design concentrate product grades by LIMS treatment alone. Again, this makes the desilification stage, to follow, critical.

While the sulphur assay reduces with progressive treatment, the Moonshine North composite LIMS testwork resulted in a concentrate of poorer sulphur grade than Moonshine. Moonshine North will need to be included in the desulphurisation study as described above.

Table 13-18: Moonshine North LIMS Testwork Results Summary

	Magnetic Stream							
Process Stage	**Yield**	**Fe_3O_4**	**FeO**	**Fe**	**SiO_2**	**Al_2O_3**	**P**	**S**
Feed	100.0	30.4	2.8	33.0	46.5	1.03	0.049	1.33
Dry LIMS	74.8	39.5	3.7	38.6	41.3	0.25	0.056	1.08
Rougher LIMS	57.6	50.2	21.1	46.6	31.5	0.20	0.049	0.81
Cleaner LIMS	45.8	63.2	24.9	55.59	20.0	0.16	0.036	0.69
Re-Cleaner LIMS	37.7	74.2	28.3	64.29	9.4	0.07	0.022	0.60
Final Recovery								
Re-Cleaner LIMS	37.7	92.2	93.6	73.4	7.6	2.6	17.0	17.1

The non-magnetic streams were assayed, including Satmagan Magnetite assays to ensure that there were no excessive losses in the testwork. These are below.

Table 13-19: Moonshine North LIMS Testwork Non-Magnetic Assays

	Non-Magnetic Stream						
Process Stage	**Yield**	**Fe_3O_4**	**Fe**	**SiO_2**	**Al_2O_3**	**P**	**S**
Dry LIMS	25.2	3.1	16.6	62.2	3.34	0.028	2.08
Rougher LIMS	17.2	1.2	10.3	77.2	0.47	0.087	1.35
Cleaner LIMS	11.8	1.2	10.6	77.3	0.42	0.102	1.11
Re-Cleaner LIMS	8.1	0.7	11.2	76.7	0.43	0.118	1.11
Wet Tail	37.1	1.11	10.59	77.1	0.45	0.10	1.22
Total Tails	62.3	1.90	13.02	71.1	1.62	0.07	1.57

The various tails streams magnetite content is low, so show no excessive losses.

If a magnetite concentrate is composed of clean magnetite and quartz material, plotting iron vs silica assays over a range of assays will show the magnetite intercept when silica is zero as 72.4% Fe (the iron content of pure magnetite). The graph then should slope downwards with a gradient of -0.724 to meet the barren magnetite intercept at 100% silica. To assess the purity of the concentrate in this bulk testwork, the iron: silica graph was developed, as below.

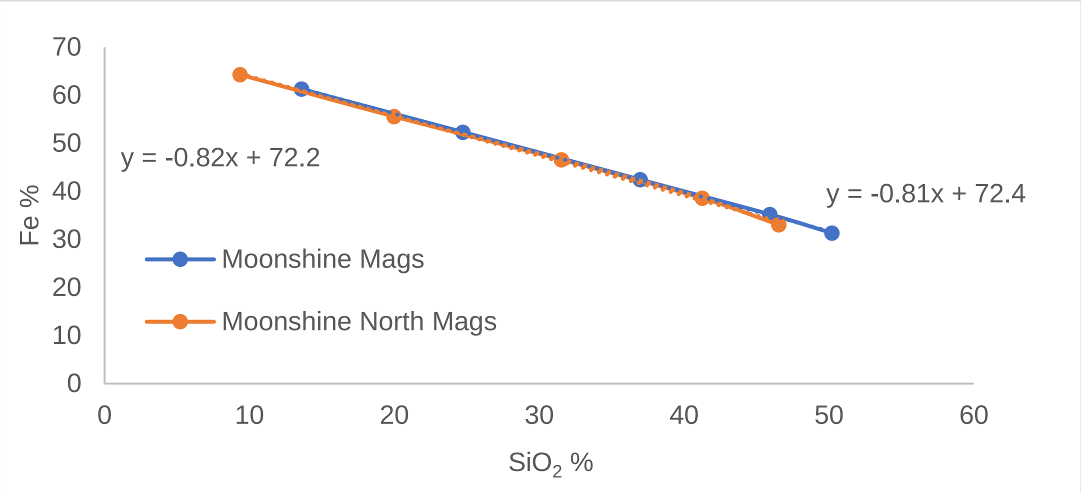


Figure 13-5: LIMS Testwork Magnetics Iron vs Silica

The equation of the line shows that the magnetite is quite pure as the intercepts are over 72% Fe. The slope is steeper than theoretical, showing the presence of other gangue minerals in the concentrate, not just quartz.

13.10. HPGR Testwork

Based on the hard ore, as well as previous industry experience, the project flowsheet utilised HPGR for the final stage of coarse comminution.

13.10.1. Open Circuit HPGR Tests

Due to sample constraints, no HPGR testwork was performed on the Moonshine North sample. The measured and inferred tonnage of the Moonshine North ore body is currently much lower than that of the Moonshine deposit, so the design intent is that the comminution circuit must be able to treat the Moonshine material.

A series of open circuit tests determined an optimal HPGR operating condition, which was used for subsequent closed circuit testwork.

Table 13-20: Open Circuit HPGR Testwork Results

Parameter / Test Ref	A1	A2	A3
Specific pressure (N/mm²)	4.1	3.4	2.6
Total Throughput (t/h)	38.6	38.4	40.9
% -2.8 mm in centre sample	51.4	45.87	41.29
-2.8 mm generated (dt/h)	19.8	17.6	16.9
Specific throughput (t/h)/(m³/s)	259.3	258.1	274.6

The size distributions of each run are graphed below.

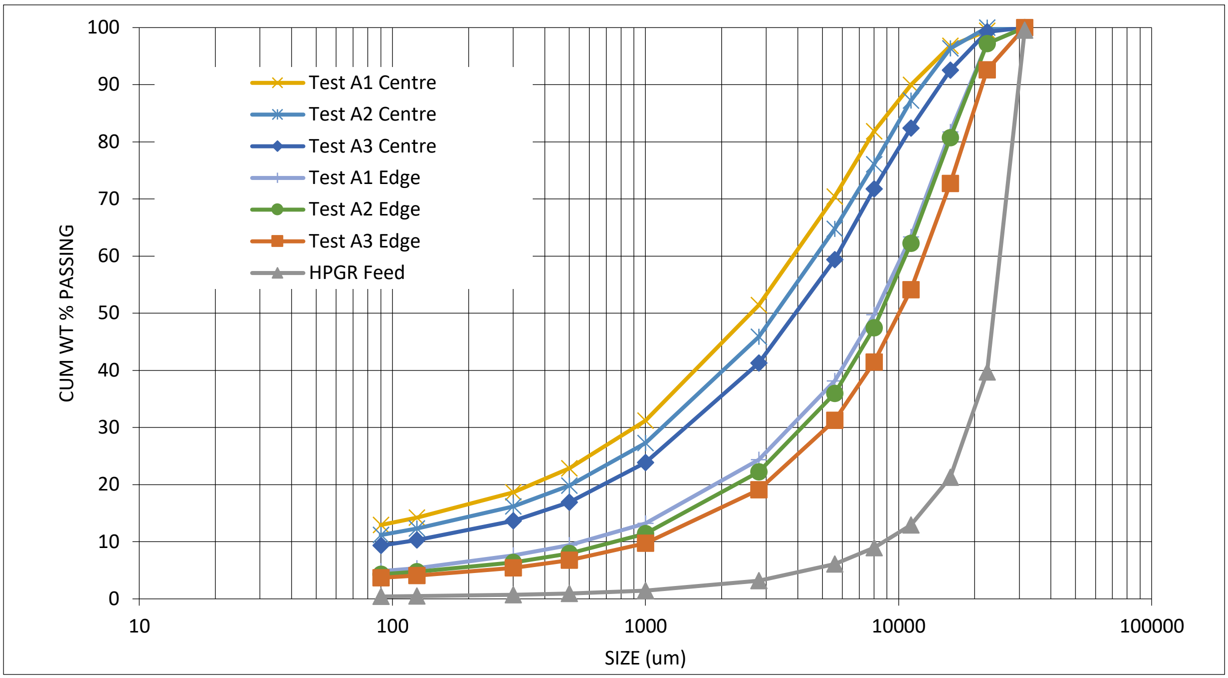


Figure 13-6: Open Circuit HPGR Tests Edge and Centre Size Distributions

The material did not produce much flake material, probably a function of the dry ore. It did crush down to approximately – 16 mm particles Figure 13-7.

Figure 13-7: Open circuit HPGR test products

The data was sent to Weir Enduron, the owners of the laboratory HPGR,for comment. They checked their model and determined the following outputs for each case, including estimated performance.

Table 13-21: HPGR Testwork Output Parameters

Parameter / Test Ref	A1	A2	A3
Press force (N/mm^2)	4.1	3.4	2.6
Specific power input (kWh/t)	2.1	1.9	1.5
Cut point (mm)		2.8	
Assumed screen efficiency (%)		90	
Predicted recirculating load (%)	116	142	169
Predicted power input (kWh/t of product)	4.54	4.60	4.04

Based on previous experience, the Weir Enduron performance recommendation was to maximise the operating Press Force, to minimise recirculating load on the screen. This maximises production efficiency. Thus 4.1 N/mm2 was used going forward.

13.10.2. Closed Circuit HPGR Testwork

As desilification testwork an required additional sample, the opportunity arose to further crush one open circuit sample in a closed circuit with a 3 mm screen. A Dry LIMS pass on the 3 x 12 mm material was added to the programme before each subsequent crushing pass, in line with the updated plant flowsheet.

This was not a locked cycle test as no new feed was added at each grind out stage, hence only subjective analysis can be used on the sequential grind-out data. Although this was not a locked cycle test, it was an opportune test to provide information going forward.

The grind-out information was assessed to estimate the expected Dry LIMS recovery for design. A more robust determination would result from a locked cycle test in the next stage of testwork.

The results are tabulated below.

Table 13-22: Closed Circuit HPGR Testwork Summary

Parameter	Test A1	Test B1	Test B2	Test B3
	Mass (%)	**Mass (%)**	**Mass (%)**	**Mass (%)**
Stage HPGR Mag concentrate yield		74.1	87.0	96.5
Stage Recycle (+12.0mm)		25.9	13.0	3.5
Screen Feed				
Mass (+12.0 mm)	14.4	6.7	1.8	2.3
Mass (+3.0 mm)	48.8	49.0	52.9	54.4
Mass (-3.0 mm)	36.8	44.2	45.4	43.3
Dry LIMS performance				
Stage Yield - Cons	85.0	93.8	96.3	97.3
Stage Yield - Tails	15.0	6.2	3.7	2.7
HPGR Operational points				
Rolls speed (rpm)	17.9	17.9	17.9	17.9
Moisture content (%)	1.2	1.2	0.5	1.2
Specific pressure (N/mm^2)	4.1	4.2	3.6	3.2
Specific throughput (t/h)/(m^3/s)	259	289	224	225
Net specific power (kWh/t)	2.1	1.6	1.7	1.6
Measured gap (average, mm)	17.0	18.6	16.0	14.1

13.11. Desilification

The study considered two methods of silica reduction for the final concentrate. These were:

- magnetic flotation, and
- reverse flotation.

The data from these tests were used to confirm the applicability of the technology. As only sighter tests, based on previous test programmes, were performed, an optimisation programme is required before plant design can be finalised.

Magnetic Flotation

Magnetic flotation is an emerging technology that uses a magnetic field to flocculate magnetic particles, so they settle against an upwards stream, to be collected in the unit underflow as a concentrate. The overflow is the tailings material that should contain the non-magnetic particles in the feed stream.

The programme used one test per sample, using a regime found to be successful on a recent Western Australian project. This would indicate if a magnetic flotation processing stage could be of value in future optimisation studies.

The performance results are provided below.

Table 13-23: Magnetic Flotation Testwork Results

	Mass %	Fe %	Fe_3O_4 %	SiO_2 %	Al_2O_3 %	S %	LOI $_{450}$ %	LOI $_{450\text{-}650}$ %	LOI $_{650\text{-}1000}$ %	LOI $_{1000}$ %
Moonshine										
Cons	94.7	63.8	80.7	10.7	0.04	0.184	-0.73	-1.48	-0.68	-2.89
Tails	5.3	24.8	15.4	60.1	0.42	0.746	0.01	0.51	0.29	0.81
Calculated Head	100.0	61.7	77.2	13.4	0.06	0.214	-0.69	-1.37	-0.63	-2.69
Recovery / Distribution										
Cons	94.7	97.9	98.9	76.0	62.8	81.4	-	-	-	-
Tails	5.3	2.1	1.1	24.0	37.2	18.6	-	-	-	-
Moonshine North										
Cons	95.7	65.8	77.9	7.57	0.07	0.581	-0.77	-1.15	-0.55	-2.47
Tails	4.3	15.7	17.5	71.9	0.33	0.287	0.12	0.04	0.48	0.64
Calculated Head	100.0	63.6	75.3	10.3	0.08	0.568	-0.73	-1.10	-0.51	-2.34
Recovery / Distribution										
Cons	95.7	98.9	99.0	70.1	82.5	97.8	-	-	-	-
Tails	4.3	1.1	1.0	29.9	17.5	2.2	-	-	-	-

The results of the magnetic flotation testwork showed poor removal of the silica particles, not achieving the desired 5% SiO2 grade specification. Due to the poor performance, this mode of separation was not considered further.

13.11.1. Reverse Flotation

Flotation is a physical separation that uses chemicals to make one species of the mineralisation (either ore or gangue) hydrophobic so it will attach to air bubbles and flow upwards to overflow out of a designed tank. Conventional flotation removes valuable material in the overflow stream, while reverse flotation removes gangue material from the overflow whilst the valuable concentrate stream flows out the bottom discharge of the tank.

Again, the programme used one test per composite to assess the potential for silica removal using reverse flotation. These were sighter tests only, to check the applicability of the technology. Further testwork would be required to provide definitive design data.

A test regime was taken from a successful South Australian project. A preliminary test showed an excessive recovery to the overflow fraction, so the reagent regime was modified to allow for a less aggressive separation. The final test conditions are shown overleaf.

Table 13-24: Reverse Flotation Testwork Conditions

Operation	Conditioning Time	Flotation Time	pH	Eh	Gem Gel	F2835-2	NaOH	MIBC
	(min)	(min)		(mV)	(g/t)	(g/t)	(g/t)	(# Drops)
Moonshine								
Condition 1	5		9.01	49	240			
Condition 2	3		8.95	40		40	2	
RoCon1		1	9.07	31			6	2
RoCon2		2	8.98	30			2	
Condition 3	1		9.02	28		10	2	
RoCon3		2	8.97	36			2	1
Condition 4	1		8.94	28		10		
RoCon4		2	8.96	31			4	
Condition 5	1		9.04	6		20	2	
RoCon5		2	8.98	21			2	
Condition 6	1		9.08	-3		20	2	
RoCon6		2	9.01	9			2	
Total	**12**	**11**			**240**	**100**	**26**	
Moonshine North								
Condition 1	5		8.95	0	120		20	
Condition 2	3		9.00	-5		20	2	
RoCon1		3	8.98	-3			10	3
Condition 3	1		8.98	6		10	2	
RoCon2		3	8.98	0			10	
Condition 4	1		9.05	6		10	4	
RoCon3		2	8.98	0			4	
Condition 5	1		9.01	7		10	4	
RoCon4		1	8.94	7				
Condition 6	1		9.00	9		10	4	
RoCon5		1	9.03	6			4	
Total	**12**	**10**			**120**	**60**	**64**	

The results are below with the applicable stream, producing the desired less than 5% silica concentrate, highlighted.

Table 13-25: Moonshine Reverse Flotation Testwork Results

Moonshine Stream		Mass	Fe	Fe_3O_4	SiO_2	S	LOI 1000
		%	%	%	%	%	%
Froth Product 1		3.81	23.7	28.6	65.3	0.112	-0.83
Froth Product 2		4.80	24.6	29.9	64.7	0.145	-0.83
Froth Product 3		5.37	33.7	41.1	51.7	0.184	-1.16
Froth Product 4		5.10	46.0	56.8	33.8	0.211	-1.68
Froth Product 5		6.41	56.9	72.1	17.4	0.342	-2.08
Froth Product 6		5.25	62.5	78.9	11.0	0.303	-2.42
Product		69.30	69.8	91.3	2.5	0.173	-3.12
Cumulative	Time (min)	Cumulative Mass %					
Feed	0	100.0	61.5	79.6	13.5	0.190	-2.64
Cell after Froth 1	1	96.2	63.0	81.6	11.5	0.193	-2.71
Cell after Froth 2	3	91.4	65.0	84.4	8.7	0.195	-2.81
Cell after Froth 3	5	86.0	67.0	87.1	6.0	0.196	-2.91
Cell after Froth 4	7	80.9	68.3	89.0	4.2	0.195	-2.99
Cell after Froth 5	9	74.5	69.3	90.4	3.1	0.182	-3.07
Product	11	69.3	69.8	91.3	2.5	0.173	-3.12

The Moonshine reverse flotation show that the required product grade was achieved in 7 minutes.

This is an encouraging result for a first sighter type test.

Table 13-26: Moonshine North Reverse Flotation Testwork Results

Moonshine North Stream		Mass	Fe	Fe_3O_4	SiO_2	S	LOI 1000
		%	%	%	%	%	%
Froth Product 1		8.45	33.5	36.9	50.2	0.979	-0.33
Froth Product 2		5.84	47.4	57.7	31.1	0.719	-1.09
Froth Product 3		3.31	55.6	65.2	20.3	0.672	-1.49
Froth Product 4		2.77	60.1	61.2	14.0	0.577	-1.77
Froth Product 5		3.14	62.8	68.4	10.6	0.515	-1.99
Product		76.5	69.3	84.0	2.6	0.528	-2.67
Cumulative	Time (min)	Cumulative Mass %					
Feed	0	100	64.1	76.7	9.42	0.583	-2.29
Cell after Froth 1	3	91.6	66.9	80.4	5.65	0.546	-2.48
Cell after Froth 2	6	85.7	68.2	81.9	3.92	0.535	-2.57
Cell after Froth 3	8	82.4	68.7	82.6	3.26	0.529	-2.61
Cell after Froth 4	9	79.6	69.0	83.3	2.89	0.527	-2.64
Product	10	76.5	69.3	84.0	2.57	0.528	-2.67

The test showed that the Moonshine North composite ore responded to reverse flotation to produce a suitable concentrate after 6 minutes of flotation test.

The overall performance profile of both samples is shown in Figure 13-8.

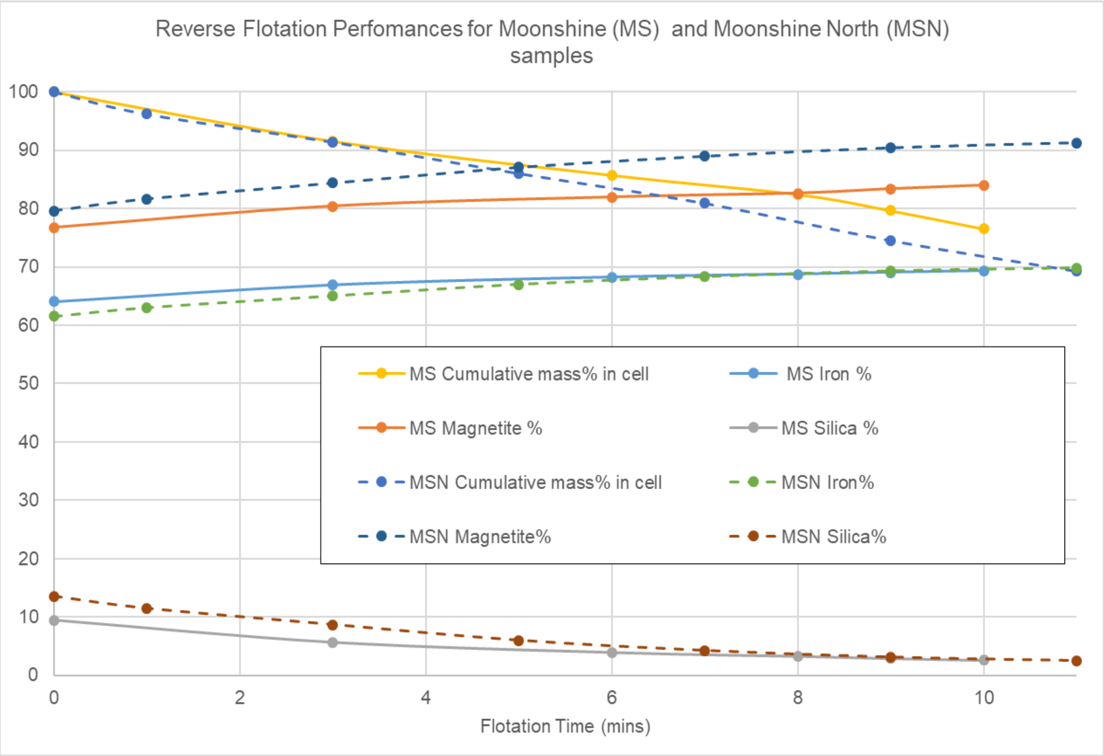


Figure 13-8: Reverse Flotation Tests Performance Profile

For both Moonshine and Moonshine North, the reverse flotation testwork results show good performance and illustrate that reverse flotation is a viable process route for desilification. Further testwork is required in future work, to optimise reagent selection and dosing, however, the study can progress with knowledge that there is a methodology to reduce the magnetite concentrate silica grade to project requirements.

13.12. Desulphurisation

During the testwork programme, it became apparent that the sulphide mineralisation was higher in the sample collected than expected and the DTW work showed poor sulphur removal. A survey of the resource model confirmed that this sulphide mineralisation was common, and the sample proved to be sufficiently representative to require investigation of the consequences of high sulphur assay in the concentrate. This investigation checked if the minerals in the concentrate responded to a conventional sulphide flotation as used in the gold and base metals industries.

High sulphur assays in magnetite concentrates are usually attributed to the presence of pyrrhotite. Pyrrhotite is an iron sulphide mineral found in igneous and metamorphic rocks. It has a chemical formula of $Fe_{(1-x)}S$, where x varies from 0 to 0.2, which indicates that it is deficient in iron. This allows pyrrhotite to be slightly to strongly magnetic. When x is close to zero, the mineral is not very magnetic, but as x increases, making the mineral more deficient, so does the magnetic susceptibility of the resulting mineral. After magnetite, pyrrhotite is the second most magnetic mineral.

To check this hypothesis quickly, a sighter sulphide orientated flotation test was performed on each LIMS cleaner concentrate. This was selected as the particle size distribution of this stream was similar to that of the flotation feeds for various sulphide flotation operations.

The LIMS concentrates were given:

- a polishing grind, with 1000 gpt Copper Sulphate added to the mill:
- 5 minutes of conditioning time with 120 gpt Potassium Amyl Xanthate (PAX)
- pH was maintained at ~8.6
- EV controlled to be mildly oxidising at +120 mV: and
- frother addition performed as required.

The Moonshine sighter sulphide flotation gave a poor result, with only 20% sulphide removal in 15% of the mass pull. This is insufficient for a viable design path.

The Moonshine North sighter sulphide flotation was marginally better than the Moonshine test, removing 25% of the sulphide mineralisation in 10% of the material.

Overall, these results show poor flotation of the sulphide mineralisation, and this was not encouraging enough to go forward as an indication of process success.

Further information on the mineralogy is required, as this may be the result of a number of factors, such as mineralisation locking, non-liberation of sulphides or association of sulphides with magnetite. Thus, the project team initiated a mineralogical programme, based on Quantitative Electron Microscope Scanning (QEMSCAN).

Table 13-27: Sighter Sulphide Flotation Testwork Results

	ASSAYS				DISTRIBUTIONS		
Products Recovered	**Mass %**	**Fe %**	**SiO_2 %**	**S %**	**Fe %**	**SiO_2 %**	**S %**
MOONSHINE							
Rougher Concentrate 1	2.63	51.57	23.82	0.324	2.20	4.61	4.22
Rougher Concentrate 2	1.67	54.76	20.46	0.309	1.48	2.51	2.55
Rougher Concentrate 3	5.32	57.98	16.99	0.305	5.00	6.65	8.04
Rougher Concentrate 4	2.16	62.69	11.15	0.268	2.19	1.77	2.87
Rougher Concentrate 5	1.68	63.30	10.82	0.254	1.72	1.34	2.11
Rougher Tail	86.5	62.25	13.06	0.187	87.4	83.13	80.2
Calculated Head	100.0	61.64	13.60	0.202	100.0	100.0	100.0
Assayed Head		61.28	13.60	0.193			
MOONSHINE NORTH							
Rougher Concentrate 1	3.10	53.76	20.00	2.400	2.59	6.61	12.36

Rougher Concentrate 2	1.74	57.22	16.55	1.720	1.55	3.07	4.98
Rougher Concentrate 3	1.40	60.37	13.07	1.360	1.32	1.95	3.17
Rougher Concentrate 4	1.84	62.24	11.14	1.100	1.79	2.19	3.37
Rougher Concentrate 5	1.19	63.00	10.41	0.974	1.16	1.32	1.92
Rougher Tail	90.7	64.79	8.76	0.492	91.6	84.85	74.2
Calculated Head	100.0	64.19	9.37	0.602	100.0	100.0	100.0
Assayed Head		64.29	9.35	0.603			

13.12.1. QEMSCAN Study

The QEMSCAN system integrates a Scanning Electron Microscope (SEM) hardware with QEMSCAN software. It uses two-dimensional particle mapping analysis to provide a visual representation of mineralogical associations.

The mineralogical elements are determined, and their associations indicate the degree of liberation and combination with each other by size. This gives liberation matrices, so showing the associations and allowing assessment of the ore.

Selected core samples were prepared for examination at BV and analysed using QEMSCAN at CSIRO located in Bentley, Western Australia.

The BV lab for developed seven samples at the survey. Two were samples of the DTR Head for each deposit sample and the other five were intervals selected by a Macarthur geologist as having high sulphur assays. The mineralogical QEMSCAN for each sample showed the following mineralogical make up.

The samples were ground to -106 µm and then deslimed using small cyclone. The first pass scan showed the following mineral distribution.

Table 13-28: QEMSCAN First Pass Mineral Distribution

Sample	MS Head DTR	MSN Head DTR	LGDD – 066 111 – 115 m	LGDD – 066 115– 119 m	LGDD – 068 107.1– 111 m	LGDD – 068 116 – 119 m	LGDD – 073 116 – 120 m
Mineral or Phase	**Mass %**	**Mass %**	**Mass %**	**Mass %**	**Mass %**	**Mass %**	**Mass %**
Fe Oxide (Magnetite)	40.84	49.87	47.77	44.94	55.21	45.28	49.20
Pyrite	2.06	4.13	1.36	4.60	0.99	5.02	0.41
Quartz	39.40	30.80	37.94	36.93	38.10	40.65	29.99
Talc	3.06	1.83	4.02	2.59	3.60	4.81	0.80
Pyroxene	5.56	6.76	0.84	0.47	0.65	2.07	16.30
Amphibole	2.42	0.88	2.87	6.24	0.09	0.36	0.23
Clays	0.72	2.92	2.84	0.04	0.02	0.03	0.29
Micas	0.31	0.23	0.01	0.00	0.00	0.10	0.02
Calcite	1.47	0.50	0.28	2.45	0.00	0.02	0.92
Dolomite/Ankerite	2.25	0.15	0.02	0.01	0.01	0.24	0.02
Phosphates	0.20	0.46	0.64	0.32	0.23	0.17	0.21
Rutile/Anatase	0.06	0.06	0.12	0.05	0.07	0.04	0.04

Feldspars	0.04	0.02	0.01	0.00	0.00	0.04	0.01
Wollastonite	0.05	0.01	0.01	0.03	0.00	0.03	0.07
Others	1.57	1.39	1.28	1.34	0.97	1.12	1.50

Note: MS – Moonshine, MSN – Moonshine North

The 'pyrite' reported mass includes any pyrrhotite, at this stage.

The images from the first scan can be found in the testwork report appendices but the form is as below. Hole 73 is not shown as it has little pyrite.

Moonshine Head	Moonshine North Head	LGDD – 066 111 – 115m	LGDD – 066 115 – 19m	LGDD – 068 107.1 – 111m	LGDD – 068 116 – 119m

Iron Oxide	Quartz	Pyrite	Grunerite	Amphibole	Talc

Figure 13-9: First Pass QEMSCAN Images

To determine particle sizing, QEMSCAN software measures the area of each particle and determines the diameter of a circle of equivalent area. This diameter is the size reported. As Pyrite is the mineral of interest, the pyrite grain size distribution for each sample is shown in Figure 13-10.

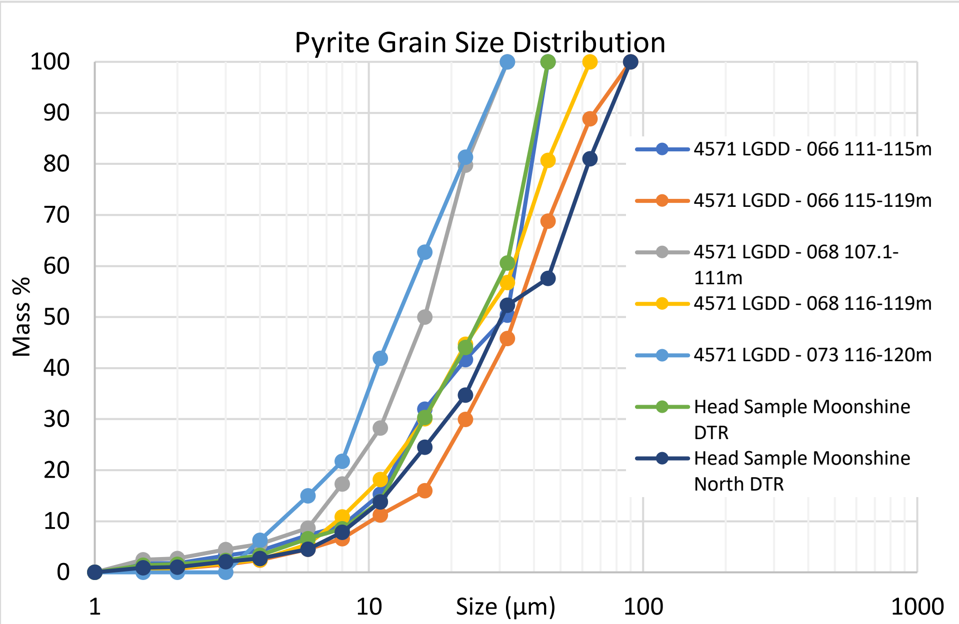


Figure 13-10: QEMSCAN Sample Pyrite Grain Size Distributions

The plot shows most of the samples to have a P80 between 40 and 60 µm.

The size relationship between the three items of interest varies so the particle, magnetite and pyrite 80% passing sizes are below.

Table 13-29: QEMSCAN Sample Feed, Magnetite and Pyrite P80

P80 µm	Moonshine DTR Head	Moonshine North DTR Head	DD 066 111 – 115	DD 066 115-119	DD 068 107.1 – 111	DD 068 116 – 119	DD 073 116 – 120
Feed	52.9	53.6	31.9	41.4	39.5	46.4	138.4
Magnetite	42.4	44.1	30.3	36.0	41.7	44.2	96.4
Pyrite	54.4	62.6	56.3	55.6	22.6	44.6	22.0

The drill hole 73 sample is aberrant and any results coming from this hole would need critical analysis.

The critical item in separating the pyrite from the other minerals is the extent that the mineral liberates from the particles it needs separation.

In a QEMSCAN analysis, liberation is determined by measuring the length of the boundaries of the mineral in question and the item is attached to it. If the entire mineral boundary attaches to the background material, it is determined to be 100% liberated. If half attaches to the background and half to another mineral it is determined to be 50% liberated.

Table 13-30: QEMSCAN Pyrite Liberation Matrix

Pyrite Liberation	MS Head	MSN Head	DD – 066 111-115m	DD – 066 115-119m	DD – 068 107.1-111m	DD – 068 116-119m	DD – 073 116-120m
Liberation Class	**Mass %**	**Mass %**	**Mass %**	**Mass %**	**Mass %**	**Mass %**	**Mass %**
<= 10%	3.25	2.48	6.74	5.38	5.04	1.84	32.17
<= 20%	0.52	0.16	1.27	0.51	20.13	0.91	15.66
<= 30%	3.89	0.84	2.07	0.09	0.62	0.89	1.59
<= 40%	9.27	1.71	0.02	0.13	0.04	0.03	21.30
<= 50%	0.51	0.87	0.18	1.11	2.38	0.05	1.36
<= 60%	2.67	5.77	0.00	0.00	0.25	0.04	4.73
<= 70%	0.00	15.87	0.14	0.00	0.04	0.04	0.00
<= 80%	0.02	12.21	10.21	0.56	0.11	0.00	0.38
<= 90%	5.71	9.09	0.00	0.30	6.65	0.21	0.00
< 100%	32.16	29.28	50.29	48.43	21.26	29.46	1.85
100%	42.01	21.72	29.09	43.49	43.47	66.52	20.97
<90%	74.17	51.00	79.38	91.92	64.73	95.99	22.82

Note:

Hole 73, again, shows aberrant with only some 20% liberation. XRD analysis of this sample showed it to be unrepresentative and required caution in mining. The analysis showed a significant amount of Grunerite present in the sample and, based on the morphology visible in the SEM; it is likely that crushing this material is likely to result in fibres. Material such as this may also be difficult to process due to the presence of amphiboles crosscutting magnetite requiring in a finer grind to liberate the magnetite and potential issue with safety and thickening of process streams due to the high abundance of fibrous minerals. There will be no more data reported on the Hole 73 sample in this report, to maintain clarity.

While the QEMSCAN data above was of value, the low concentration of sulphides made analysis of the sulphides difficult and did not supply the expected data. Namely showing how the sulphide mineralisation is associated with other minerals and how it would be expected to achieve a sulphide barren stream if beneficiated.

The samples images were re-examined and the grains with sulphide and magnetite mineralisation extracted. The assumption was that free pyrite and pyrite associated with gangue (non-magnetic) mineralisation will ultimately pass to a tailings stream so need not be considered a separate beneficiating species.

Filtering out the non-sulphide particles shows a better example of the distribution of the magnetite/sulphide material and even shows minor pyrrhotite, albeit all associated with other minerals. The images showed many grains that contain both magnetite and sulphide material. The issue is estimating what goes where when it is treated.

The QEMSCAN operator analysed the images to estimate the sulphur content after reverse flotation. Filtering the data set to remove all particles expected to float and determining the mineral content of the remainder should give some estimation of the underflow of a reverse flotation cell. The filtering assumed that particles with a free surface over 20% of iron sulphide (pyrite and pyrrhotite) would respond to flotation.

The scan data of the sample after removing the material with >20% area are tabulated in Table 13-31.

The samples with significant amounts of pyrite show good liberation at this -150 µm top size. For example:

- Sample LGDD – 068 116-119 m contains 5.0 % pyrite and 96% is >90% liberated
- Sample LGDD – 066 115-119 m contains 4.6 % pyrite with 92% is >90% liberated; and
- The Moonshine North Head Sample shows 4.1% pyrite but only about 50% liberated, even though the PSD is similar to other samples.

Examination of the mineral locking shows that the main locking minerals are magnetite and quartz. It is also shown that the pyrite is a relatively large, liberated particle in most samples.

Table 13-31: QEMSCAN Analysis Predicted Sulphur Grade Removing >20% Iron Sulphide Free Surface Particles

	MS Head	MSN Head	DD-066 111-115m	DD-066 115-119m	DD-068 107.3-111m	DD-068 116-119m
Sulphur Grade %	0.0412	0.1874	0.0217	0.0283	0.0073	0.0502
Mineral %						
Pyrite	0.0321	0.1552	0.0361	0.0449	0.0258	0.0855
Pyrrhotite	0.0634	0.2771	0.0056	0.0110	0.0035	0.0119

These results show that all the samples, except the Moonshine North Head sample, have the potential to leave a tailings grade less than the desired 0.1% sulphur, given that reverse flotation can remove all the particles exceeding 20% free surface.

Due to the disseminated form of the pyrrhotite, it actually results in being more abundant than the pyrite in Moonshine North.

A programme of sulphide reverse flotation is to be developed for future assessment of this issue if it becomes required.

13.13. Tailings Dewatering Testwork

While not included in the current flowsheet, tailings dewatering, and dry stacking is commonly used technique to reduce water requirements of water critical projects.

Using a blended tailings sample from each deposit, developed by mixing the various wet tailings streams from LIMS testwork, testwork was performed by the Outotec laboratory to determine parameters and discharge conditions for the design. The results are summarised in Table 13-12.

Table 13-32: Tailings Thickener and Filtration Testwork Results

Parameter	Unit	Moonshine Tailings	Moonshine North Tailings
Thickening			
Solids Loading	t/hr m^2	1.5	1.5
Flocculant dosage	gpt	20	10
Flocculant		Magnafloc 155	Magnafloc 155
Overflow Clarity	mg/L	140	130
Underflow density	% solids w/w	63	64
Diameter @ 691 dtph	m	25	25
Filtration (Fed by thickener underflow from above)			
Pressure Filter, MITO S-171 cloth			
Filtration Rate	kgDS/m^2 h	305	358
Cake moisture	% solids w/w	11.4	11.8
Filtrate clarity	ppm	280	110
Horizontal Vacuum Filter, S-71 cloth			
Filtration Rate	kgDS/m^2 h	394	746
Cake moisture	% solids w/w	16.1	17.0

13.14. Comminution Review

CMD Consulting Pty Ltd (CMD) reviewed Engenium's PFD's and Metso Outotec (MO) simulations and calculations that define the comminution circuit for the Lake Giles Iron Project.

CMD concurs with the circuit concept of primary crushing, followed by secondary crushing in a closed circuit with a dry classification screen, followed by an HPGR in a closed circuit with a wet classification screen, two ball mills and two Vertimills®.

For this review, CMD has used the same ore properties that Engenium has used. Comminution data is very limited, so the circuit design and equipment sizing should be considered preliminary. There are many haematite and magnetite resources in the area, however, the design of this circuit is restricted to the Moonshine ore deposit only.

The power modelling technique was used to size the equipment. The performance of the LIMS is based on information provided by the Engenium team.

JKSimMet models were used to add more definitions such as stream size distributions and mass balances. In all cases, the circuit conditions are tuned to achieve the targeted grind size T80's after each major unit process. A final magnetite concentrate of 3 Mtpa with a P80 of 38 µm is derived from processing a ROM head feed of ~10 Mtpa.

Key results/findings and comments are as follows:

- There is in general very little presentation of circuit-specific energy requirements and unit process-specific energy requirements in relation to design ore properties

- Comminution tests are limited to a few tests suited to concept studies
- CMD's analysis is in line with the results generated by Engenium and Metso-Outotec
- CMD's review of the comminution work concluded that the following equipment would be suitable for the operation
- Single Gyratory crusher 50x60" MkII
- Single MP1250 secondary crusher
- Single HPGR 2.4 m Ø x 1.65 m wide with 2 x 3.3 MW motors
- Two ball mills 7 MW each 20' x 33.5' EGL Ball mills (Shell supported TK mills), and
- Two 2,237 kW VTM3000 units.

Conclusions and Recommendations

The results were assessed as they were made available, and this changed some test parameters during the programme. The results are tabulated in Table 13-33.

Table 13-33: Testwork Summary

Testwork	Unit	Moonshine	Moonshine North
Head Assays			
Assay Fe Grade	%	30.7	32.8
SiO_2 Grade	%	50.3	47.4
Al_2O_3 Grade	%	0.37	1.03
P Grade	%	0.04	0.05
S Grade	%	0.56	1.21
LOI Grade	%	-0.12	0.68
In Situ SG		3.46	3.46
Concentrate BD Unconsolidated	t/m^3	1.88	1.95
Concentrate BD Consolidated	t/m^3	2.39	2.48
Abrasion Index		0.58	0.53
BWI @ 75 µm	kWh/t	13.5	14.9
BWI @ 125 µm	kWh/t	13.5	14.9
SMC A*b		37.6	38.7
DTR @ 38 µm Fe Grade	%	65.0	65.7
SiO_2 Grade	%	12.7	8.5
Mass recovery	%	40.9	43.7
HPGR			
Press Force	N/mm^2	4.1	
Total Throughput	t/h	38.6	
-2.8 mm in centre sample	%	51.4	
-2.8 mm generated	dtph	19.8	
Specific throughput	$(t/h)/(m^3/s)$	259.3	
Specific power input	kWh/t	2.1	
Predicted recirculating load	%	116	
Predicted power input (of product)	kWh/t	4.54	
Magnetic Separation			
Coarse Cobbing at -6 mm			
Mass Recovery	%	83.9	74.8
Fe Grade	%	32.5	38.6
SiO_2 Grade	%	45.9	41.3
S Grade	%	0.36	1.08

BBWi Of CC Product @ 125 µm	kWh/t	12.8	13.3
Single Stage LIMS @ 212 µm			
Mass Recovery	%	66.5	57.6
Fe Grade	%	42.5	46.6
SiO_2 Grade	%	37.0	31.5
Al_2O_3 Grade	%	0.14	0.20
P Grade	%	0.043	0.049
S Grade	%	0.52	0.81
2 stage LIMS @ 106 µm			
Mass Recovery	%	51.8	45.8
Fe Grade	%	52.3	55.6
SiO_2 Grade	%	24.7	20.0
Al_2O_3 Grade	%	0.10	0.16
P Grade	%	0.030	0.036
S Grade	%	0.22	0.69
2 stage LIMS @ 38 µm			
Mass Recovery	%	43.6	37.7
Fe Grade	%	61.3	64.3
SiO_2 Grade	%	13.6	9.4
Al_2O_3 Grade	%	0.05	0.07
P Grade	%	0.020	0.022
S Grade	%	0.19	0.60
Reverse Flotation			
Mass Recovery	%	35.3	32.3
Fe Grade	%	68.3	68.2
SiO_2 Grade	%	4.2	3.9
Al_2O_3 Grade	%	0.04	0.07
P Grade	%	0.018	0.019
S Grade	%	0.19	0.54
Tailings Thickening			
Solids Loading	t/m2 hr	1.5	1.5
Flocculant dosage	gpt	20	10
Flocculant		Magnafloc 155	Magnafloc 155
Overflow clarity	mg/L	140	130
Underflow density	% solids w/w	63	64
Diameter @ 691 dtph	m	25	25
Tailings Filtration			
Pressure Filter			
Filtration Rate	kgDS/m2 h	305	358
Cake moisture	% solids w/w	11.4	11.8
Filtrate clarity	ppm	280	110
Vacuum Filter			
Filtration Rate	kgDS/m2 h	394	746
Cake moisture	% solids w/w	16.1	17.0

A discussion of the results alongside the resource model led to the project product being defined as below. If this specification is found to be unsuitable, due to the high sulphur content, further work will be needed to address the issue.

Table 13-34: Project Product Specification

Fe %	Al_2O_3 %	SiO_2 %	P %	LOI %	S %
66.1	0.10	4.9	0.02	-2.7	0.6

13.15. Recommendations

The following recommendations arise from the completed metallurgical testwork and analyses.

Further drilling should be performed, in order to produce representative composites based on the ore types in the deposit, in sufficient quantities to allow the performance of a comminution and pilot plant programme.

The number of drill holes should be determined by addressing any ore types made evident in the geological modelling so that sufficient sample of each ore type to make feed for a significant flotation programme as well as for a Pilot Plant programme using a master composite. This would be a number of tonnes of the sample.

For the comminution programme CMD recommend that assuming a payback period of 7 years, at 10 Mtpa would require at least 70 samples, each sample representing 1 Mt of ore.

The plant will need to be designed to treat a highly abrasive ore.

The removal of material during dry LIMS processing is small compared to industry benchmarks, so an assessment of the benefit of the dry LIMS processing should be included during a value engineering stage.

The final size for the grinding circuit will be 80% passing 38 µm.

Further bench-scale reverse flotation work will be required to optimise reagent selection, dosing and recovery profile. A scale-up factor will be needed in sizing the flotation cells, expected to be in the range of 2 – 2.5 times the laboratory retention times. This should be vendor advised.

Further assessment of the sulphide mineralisation, in order to determine a mechanism to address desulphurisation and provide a path going forward.

Further recommendations from the CMD report include:

Algorithms that correlate ore properties with geological data such as RQD and fracture frequency could be an economical way of defining the ore over time.

Forecast modelling is recommended to better manage the operating conditions of the circuit if and when the ore blends change.

Metsso-Outotec will need to provide process guarantees for the Vertimills and show methods for the design and scale-up procedures.

14. Mineral Resource Estimates

14.1. Summary

The Moonshine and Moonshine North Mineral Resources are material updates to the previously reported Mineral Resource (Snowden, 2011), based upon an infill drill program (21 RC holes for 3,322 m and nine diamond holes for 1,676.5 m), a geological re-interpretation, and a significant increase in the number of DTR and density results.

The Sandalwood, Clark Hill North, Clark Hill South, and Snark Mineral Resources were all reported in 2009 (Allen, 2009) and in 2010 (Macarthur, 2010). No further exploration activities have occurred since then; however, the supporting geological model for the Clark Hill South Mineral Resource was re-interpreted and re-estimated following the QP's review of the previous Mineral Resource estimate. In addition, the Mineral Resources are now reported within the existing tenure, resulting in a minor tonnage no longer reported.

3D modelling methods and parameters were used in accordance with best industry practices. Datamine mining software was used for establishing the 3D block models and subsequent grade estimates. Geological interpretations of the iron mineralisation were derived from the drillhole logs and assays. Statistical and grade continuity analyses were completed in order to characterise the mineralisation and were subsequently used to develop grade interpolation parameters. Grade was interpolated into the block models using ordinary kriging. Densities were calculated for each block based upon an iron-density algorithm.

The block models were classified in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014).

14.2. Software

The Mineral Resource estimates were prepared using Datamine Studio, with the geological interpretations carried out using Micromine software. Geostatistical analyses were conducted using "Supervisor" (Snowden Industries) and "GeoAccess Professional" (Widenbar and Associates) packages.

14.3. Moonshine and Moonshine North

14.3.1. Drillhole Database

The drillhole data was provided in two separate databases, as per the following:

- 2019 drilling program, maintained by Macarthur
- Pre-2019 drilling data, previously maintained by CSA, and subsequently maintained by Macarthur with all security protocols maintained.

The databases were provided in Microsoft Access format with tables containing, at a minimum, collar, survey, assay, lithological and weathering data. Both databases were separately imported into Datamine and the imported data validated for the following items:

- Overlapping sample data (assays, surveys, specific gravity, lithology logs)
- Missing or absent data
- Negative assay grades; and
- Excessive drillhole deviation over short intervals.

A few minor issues were noted and reported to Macarthur, who corrected the relevant database table. Assay data presenting as negative values from the pre-2019 drilling database were treated as missing samples, as per advice provided by Macarthur, and the assay grades set to absent. The assays for manganese (head and concentrate assays) in the pre-2019 assay data were provided in elemental state, and the QP re-calculated the assays into their oxide constituents, to match the equivalent assays as provided int the 2019 database assay table.

The following assay re-calculations were preformed:

- MNO = MN * 1.2912; and
- MNOCON = MNCON * 1.2912.

Some assays in the pre-2019 database assay table were provided in ppm format, and were converted to percentage to match the 2019 database table settings, as per the following formula, for both head and concentrate assays:

- CR = CR_PPM / 10000; and
- V = V_PPM / 10000.

All assay fields were set to their appropriate oxidation state during importation of data from the laboratory certificates into the database.

A single drillhole file was created in Datamine after merging the relevant tables from the two databases, capturing collar, survey, assay, geology, DTR and density data. Drillholes were flagged according to drillhole type and year of drilling, to allow relevant statistical assessment of the data to occur.

Drillhole statistics are presented in Table 10-1. The database was provided to CSA on 4 June 2020, with no additional data provided thereafter.

A drillhole collar plot for Moonshine and Moonshine North is presented in Figure 10-1 in Section 10.1.

14.3.2. Topography

A LiDAR topographic survey was flown in June 2011. The data was re-sampled from 1 m to 2 m and exported as a wireframe surface in dxf format. The choice of a coarser contour interval has not resulted in any noticeable difference to resource volumes at the “outcropping” surface of the BIF strata.

The dxf file was imported into Datamine and saved as a wireframe surface. The surface was validated against several drill collars, representing different geographical locations of the resource, to ensure matching elevation levels between drillhole survey and topographic survey. The topographic DTM covers an area significantly larger than the mineralisation footprint and the area was trimmed to cover the deposit footprint. The topographic survey is considered adequate to support the Mineral Resource estimates.

14.3.3. Geological Interpretation

All geological models, for lithology, weathering and mineralisation, were interpreted and prepared by Macarthur. Discussion is provided in Section 7.

The outcropping geology of the project area is comprised of a combination of unaltered silica-rich BIFs and altered, enriched haematite/goethite BIFs. Weathering has resulted in the leaching of majority of the silica from the BIFs, thus producing a rock with elevated iron and decreased silica grades, near surface. These enriched bands vary from 10 m to 150 m in true thickness and are steeply dipping at 70–90°.

The main zones of mineralisation are interpreted as a series of thick tabular units, closely following the shape of the host BIF unit, with moderate to minimal structural deformation. More intense deformation is modelled at the south edge of the Moonshine prospect with several synclinal structures and possible shearing related to recumbent folds, which increase the apparent thickness of the zones of mineralisation.

Depth and consistency of mineralisation has been confirmed to in excess of 250 m below surface as demonstrated by results from several drillholes, confirming a consistent easterly dip of the hanging wall for the majority of the Moonshine and Moonshine North prospects.

A review of a log probability distribution of all DTR results (Figure 14-2) reveals a minor inflection at approximately 15% DTR and coupled with geological logging of magnetite mineralisation from drill cuttings, lead to a cut-off of 15% DTR to be selected for modelling of the mineralisation domains.

The interpreted mineralisation domains are confined to the fresh rock weathering domain, truncated at the base of oxidation, and by demonstrated levels of confidence at depth as determined by depth of drilling. Small pockets of internal waste, including quartz veins, mafic dykes, and shale horizons, which have DTR <15%, are included in the mineralisation domains due to their small thickness, typically 1–3 m, in comparison to the overall width of the mineralisation, making them unsuitable to selectively exclude.

The footwall of the mineralisation at Moonshine and, to a lesser extent, Moonshine North may sometimes be constrained by the thickness of the siliceous footwall (>60% SiO2), which make up the footwall of the western lodes, with thicknesses up to 80 m, as observed in drill samples and in outcrop. This siliceous footwall is modelled as part of the primary BIF package and is demonstrated by consistent unit thickness and strike extent over 100 m. The siliceous footwall is excluded from the mineralisation domains and Mineral Resource due to the low DTR results, with high amount of silica remaining in the magnetic fractions. Figure 14-2 and Figure 14-3 show representative cross sections through the Moonshine deposit, with BIF and mineralisation domain boundaries indicated.

The sectional interpretation was completed on 200 m ± 100 m oblique sections for the Moonshine deposits, with sectional spacing reduced to 50–100 m in areas where infill drilling occurred during 2019. The mineralised envelopes for Moonshine and Moonshine North were projected down to the 100 mRL, although Mineral Resources were not always reported to these depths of mineralisation.

Wireframe solids were created, linking the sectional polygons along strike. The wireframes were imported into Datamine where they were given unique file names and verified to check for crossing facets and open triangles. A total of eight mineralisation domains define the Moonshine deposit and eight domains define the Moonshine North deposit.

The domains vary in strike extent, depth extent and thickness. Magnetite mineralisation supporting the Mineral Resource is confined to below the base of oxidation surface. Mineralisation is recorded in the oxide zone, above the base of oxidation, but is not regarded as part of the magnetite Mineral Resource.

A representative cross section through the Moonshine deposit is presented in Figure 14-4 showing the host BIF unit and mineralisation domain (where DTR >15%), with drillholes, as modelled in support of the Mineral Resource. Moonshine North exhibits similar geometry of the host geological units to Moonshine, as shown in Figure 14-4. Table 14-1**Error! Reference source not found.** presents the resource model variables and codes associated with the key geological features.

Table 14-1: List of Geological Models and Datamine Filenames

Feature	Deposit	Wireframe (*tr/pt)	Datamine variable	Code
Weathering	All	Box2_	WEATH	10 (above)
		Box2_		30 (below)
Mineralisation	Moonshine	W1	MINZON	1001
		E1 to E7		2001 to 2007
	Moonshine North	NW1 to NW3		3001 to 3003
		NE1 to NE5		4001 to 4005
Lithology	All	bif	LITH	1

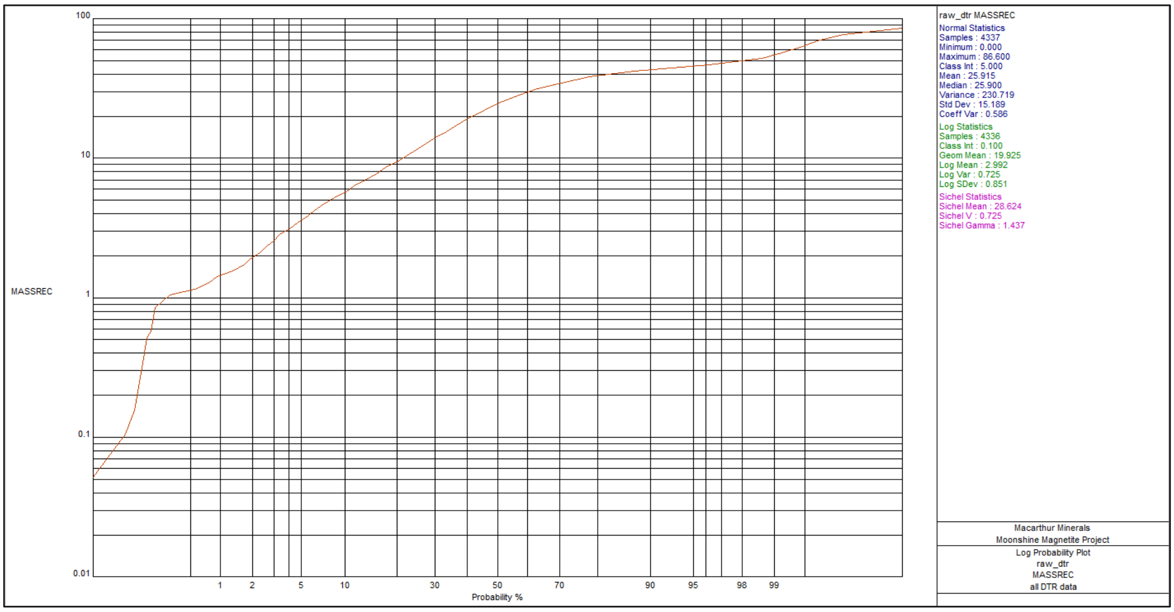


Figure 14-1: Log Probability Plot, DTR (%) All Sample Data

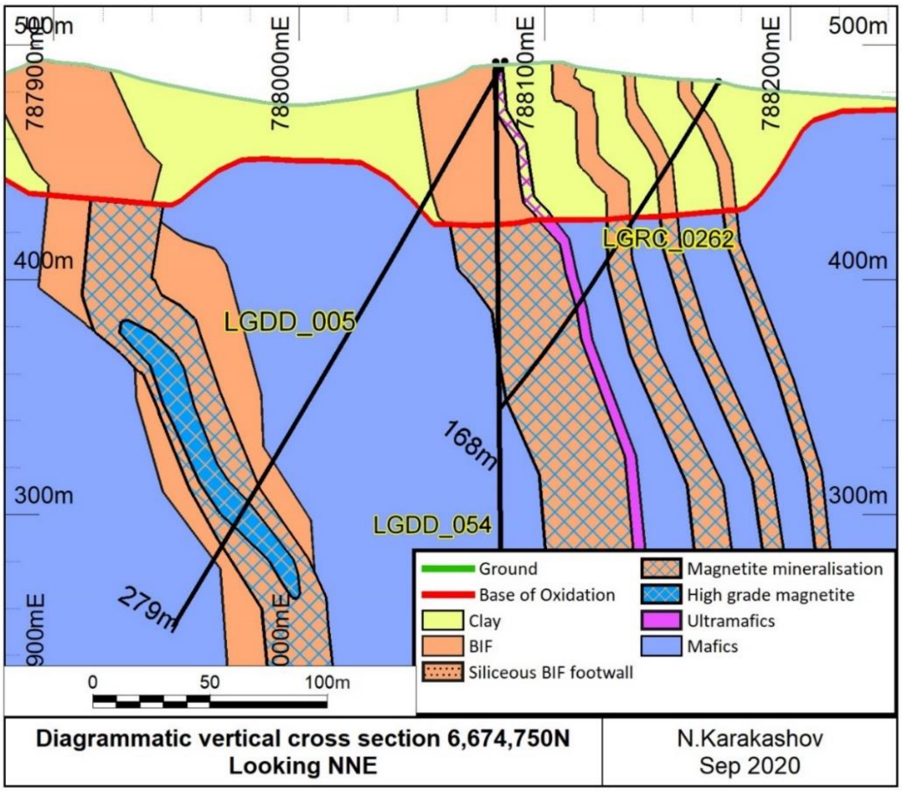


Figure 14-2: Cross Section showing relationship of high-grade magnetite pockets and bulk magnetite mineralisation, Moonshine (variability of depth of weathering is demonstrated)

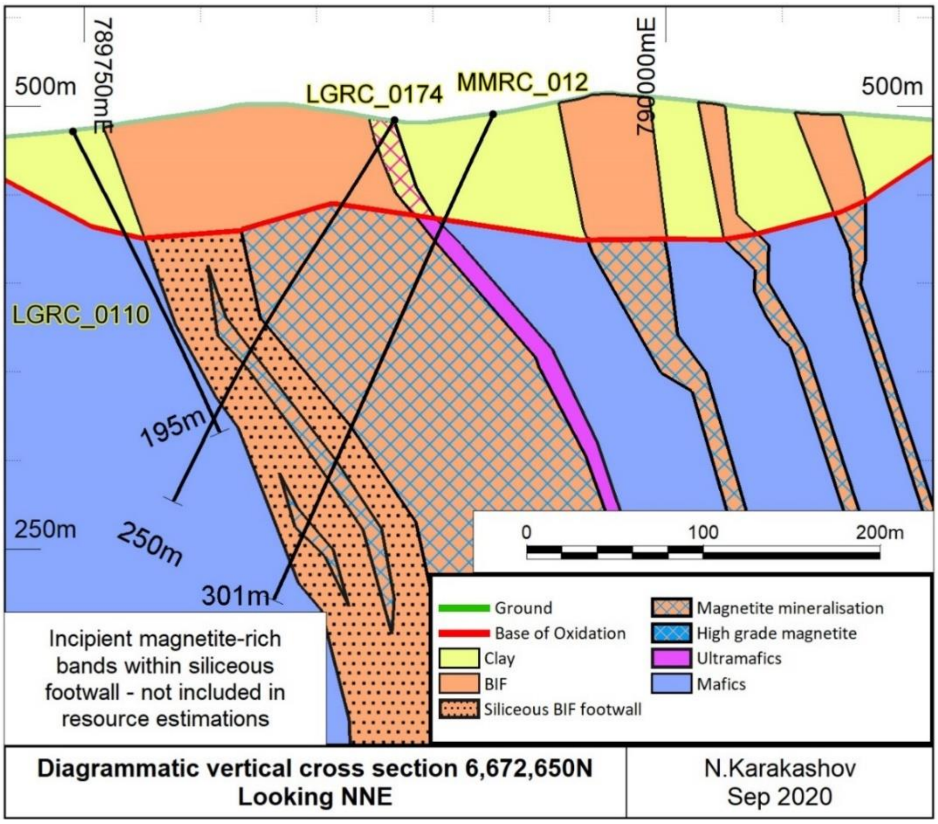


Figure 14-3: Cross section showing a typical profile through Moonshine, with a pronounced siliceous footwall

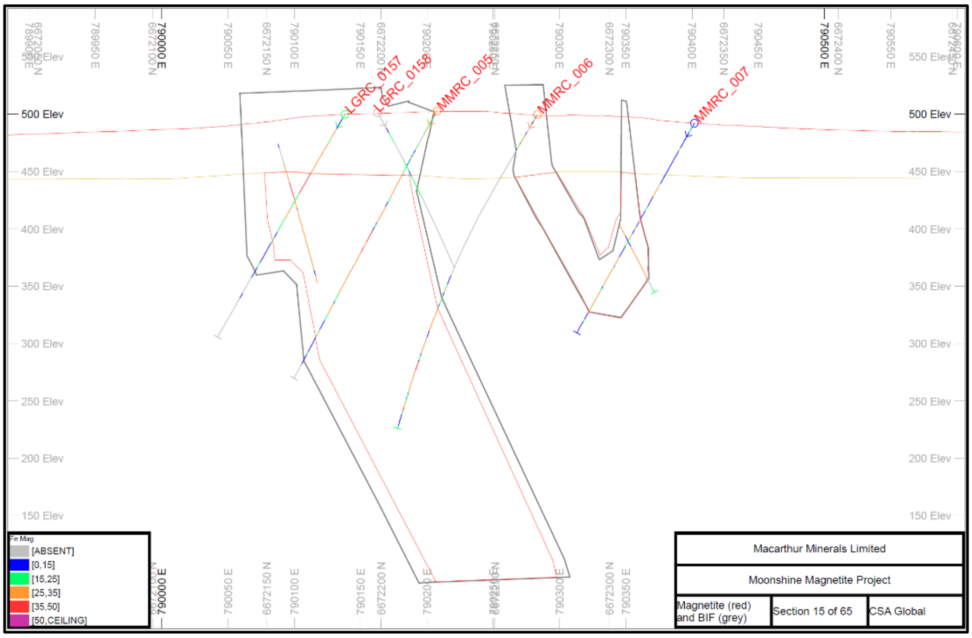


Figure 14-4: Cross section through Moonshine, showing host BIF (grey) and mineralisation envelope (where DTR>15%, red)

Note: Also shown are "base of oxidation" (yellow surface) and topographic surface (red). Drillholes shown with traces coloured by Fe %. View to north-northwest. Date of figure is July 2020.

14.3.4. Sample Coding by Domain

Drillhole samples within the Datamine drillhole files were flagged with unique codes according to the geological, mineralisation and weathering domain within which they were located.

14.3.5. Sample Compositing

An analysis of sample lengths for the domained sample data indicate a range of sample lengths of between <1 m to 6 m lengths. RC samples were sampled at between 1 m and 6 m intervals over the life of the Project, with a decision made to select a composite length of 5 m.

14.3.6. Statistical Analyses

Summary Statistics

Statistical summaries for key grades are presented in Tables 14-2 to 14-5 for Moonshine and Moonshine North. Histograms for the head and concentrate grades for iron, phosphorous and silica are presented in Figure 14-3 to Figure 14-4. A histogram for Mass Recovery (DTR) results is presented in Figure 14-8. The histograms show the result of separating the magnetite from the whole sample using the Davis Tube method, with iron grades significantly higher in the product, and a corresponding material decrease in silica and phosphorous grades. The higher silica and phosphorous grades are associated with silica and non-magnetic minerals caught in the gangue material, such as the siliceous bands in the host BIF rock.

Table 14-2: Summary Statistics, Head Grades, Moonshine Values in %)

Statistic	Al_2O_3	CaO	Fe	LOI	MgO	MnO	P	S	SiO_2
Number	2,089	2,089	2,089	2,089	2,089	2,050	2,084	2,089	2,089
Minimum	0	0.03	1.6	-1.05	0.05	0.012	0.007	0.001	28.6
Maximum	14.93	22.10	42.20	17.00	31.24	2.20	0.276	12.30	88.53
Mean	1.19	2.38	27.29	1.51	2.83	0.20	0.046	0.80	52.07
Standard deviation	2.12	1.74	7.54	2.21	3.56	0.19	0.017	1.41	8.38
Variance	4.49	3.03	56.90	4.88	12.64	0.03	0.000	2.00	70.27
Coefficient of variation	1.79	0.73	0.28	1.47	1.26	0.94	0.361	1.77	0.16

Table 14-3: Summary Statistics, Concentrate Grades, Moonshine (Values in %)

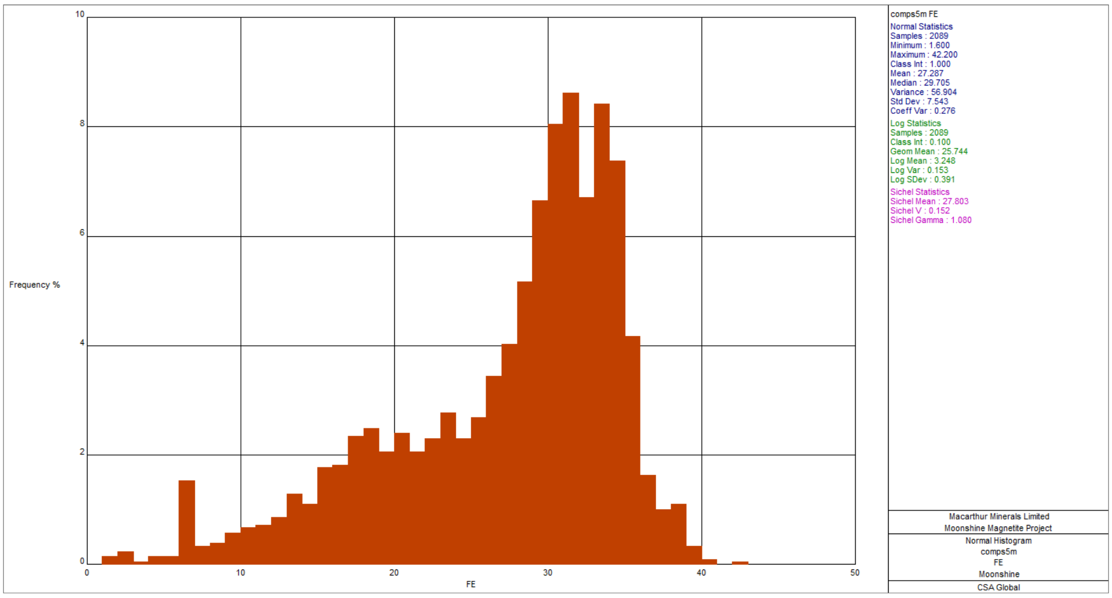
Statistic	Al_2O_3	CaO	Fe	LOI	MgO	MnO	P	S	SiO_2	DTR
Number	1,766	1,765	1,766	1,735	1,765	1,727	1,765	1,765	1,766	1,870
Minimum	0.001	0.001	48.3	0	0.001	0.001	0.003	0.001	1.448	0.001
Maximum	1.50	1.98	70.98	6.60	2.42	0.39	0.08	19.20	33.10	78.24
Mean	0.10	0.20	66.34	0.11	0.27	0.05	0.02	1.17	6.66	29.74
Standard deviation	0.16	0.15	3.15	0.59	0.15	0.04	0.01	2.44	3.85	13.18
Variance	0.03	0.02	9.91	0.35	0.02	0.00	0.00	5.95	14.80	173.68
Coefficient of variation	1.55	0.74	0.05	5.52	0.58	0.74	0.65	2.09	0.58	0.44

***Table 14-4*:** **Summary Statistics, Head Grades, Moonshine North (Values in %)**

Statistic	Al_2O_3	CaO	Fe	LOI	MgO	MnO	P	S	SiO_2
Number	659	659	659	650	659	659	659	659	650
Minimum	0.00	0.02	3.89	-1.07	0.06	0.02	0.007	0.00	2.80
Maximum	19.45	22.14	61.80	16.20	20.20	1.01	0.298	11.06	79.57
Mean	1.35	2.63	30.56	2.21	2.57	0.20	0.054	0.84	46.33
Standard deviation	2.46	2.61	8.41	2.80	1.75	0.17	0.020	1.55	12.55
Variance	6.06	6.82	70.79	7.84	3.05	0.03	0.000	2.41	157.49
Coefficient of variation	1.82	0.99	0.28	1.27	0.68	0.87	0.374	1.86	0.27

Table 14-5: **Summary Statistics, Concentrate Grades, Moonshine North (Values in %)**

Statistic	Al_2O_3	CaO	Fe	LOI	MgO	MnO	P	S	SiO_2	DTR
Number	455	455	455	438	455	455	455	455	455	462
Minimum	0.00	0.00	29.50	0.00	0.03	0.00	0.002	0.00	0.87	0.00
Maximum	3.90	7.07	71.44	2.91	4.00	0.35	0.773	10.40	51.50	82.04
Mean	0.18	0.35	64.56	0.10	0.43	0.05	0.056	0.71	8.45	30.21
Standard deviation	0.38	0.54	6.06	0.39	0.41	0.06	0.112	1.34	7.05	11.75
Variance	0.14	0.29	36.72	0.15	0.16	0.00	0.013	1.79	49.69	137.99
Coefficient of variation	2.11	1.55	0.09	4.07	0.95	1.06	2.007	1.89	0.83	0.39



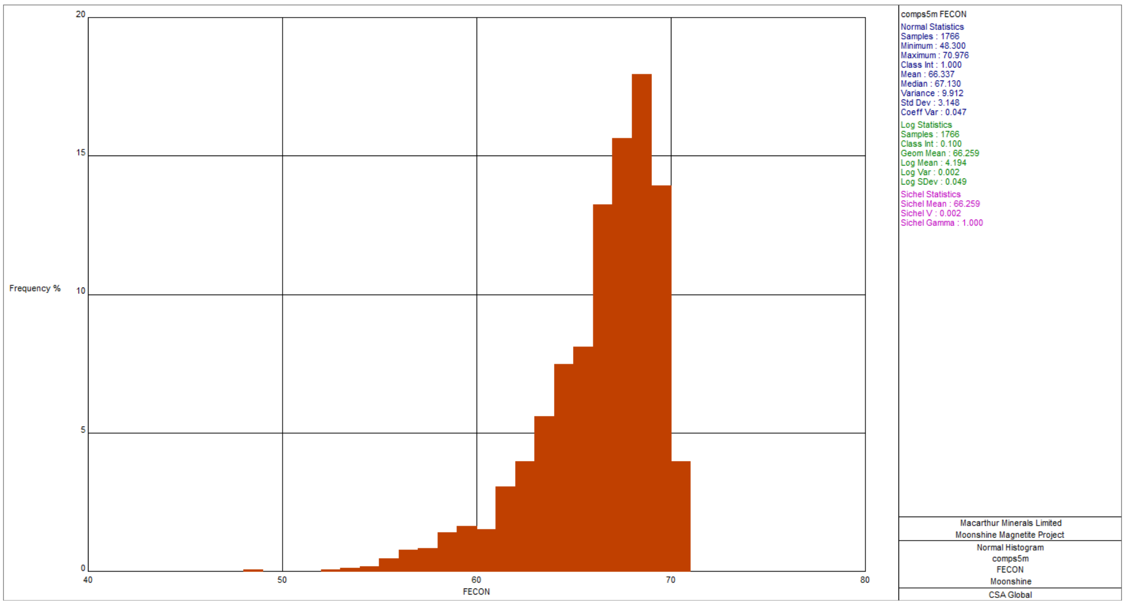


Figure 14-5: Histograms of Fe (Head) and Fe (Concentrate), from composited samples within mineralisation domains in Moonshine (values in %)

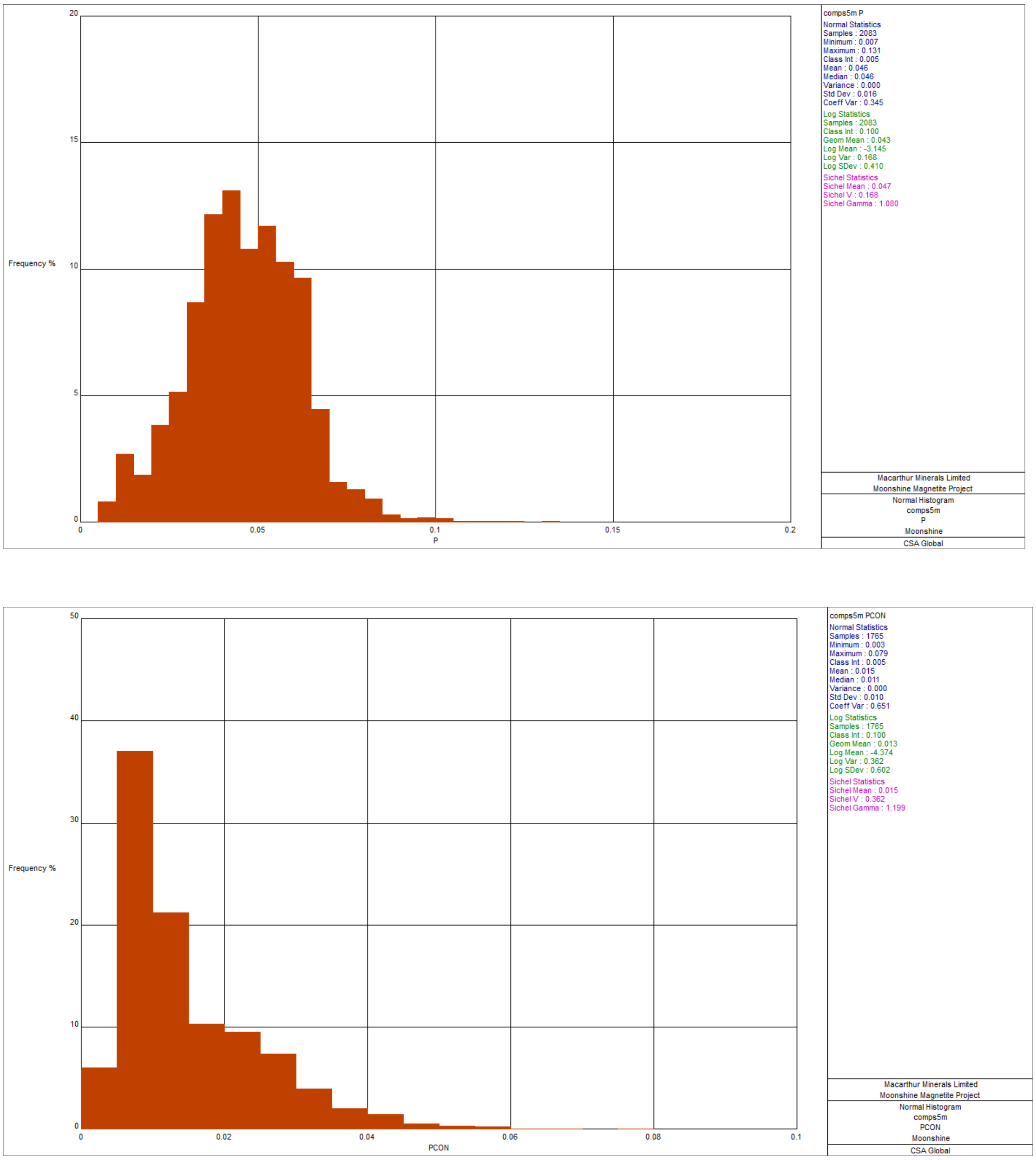
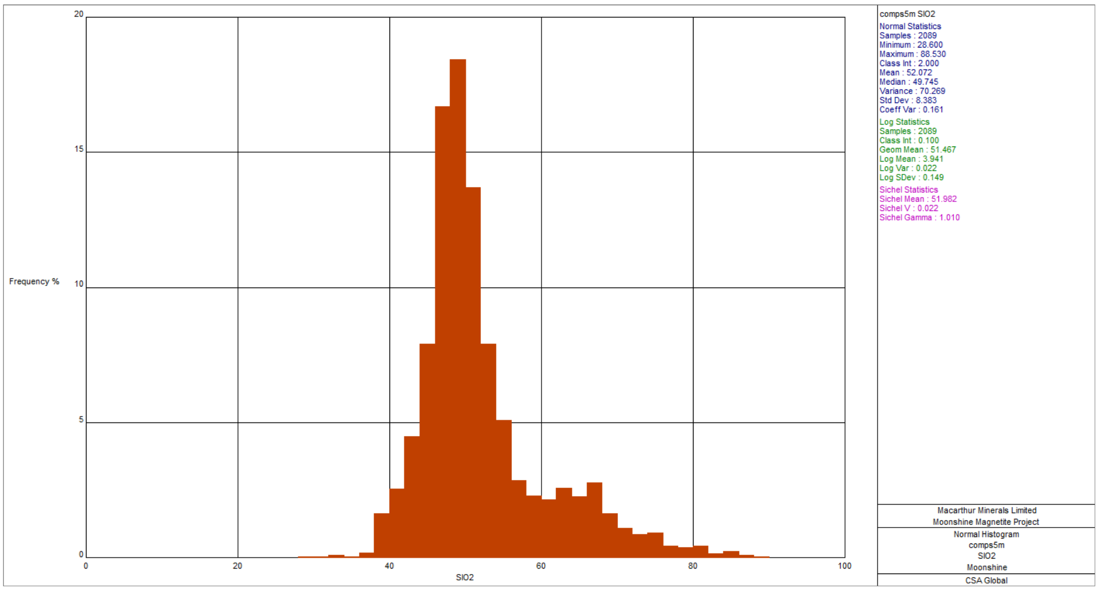


Figure 14-6: Histograms of P (Head) and P (Concentrate), from composited samples within mineralisation domains in Moonshine (values in %)



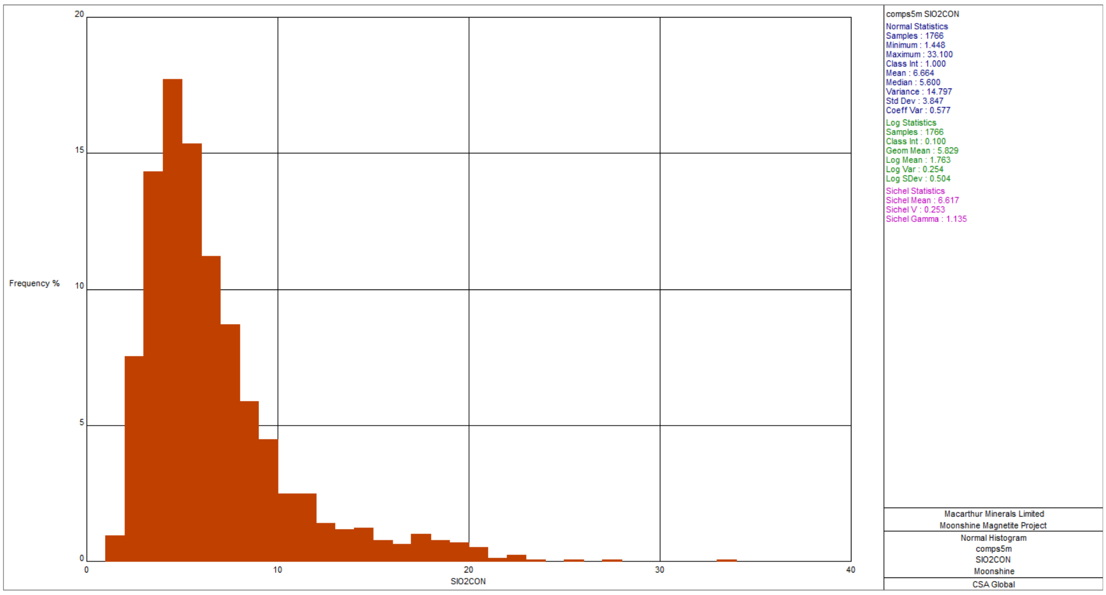


Figure 14-7: Histograms of SiO2 (Head) and SiO2 (Concentrate), from composited samples within mineralisation domains in Moonshine (values in %)

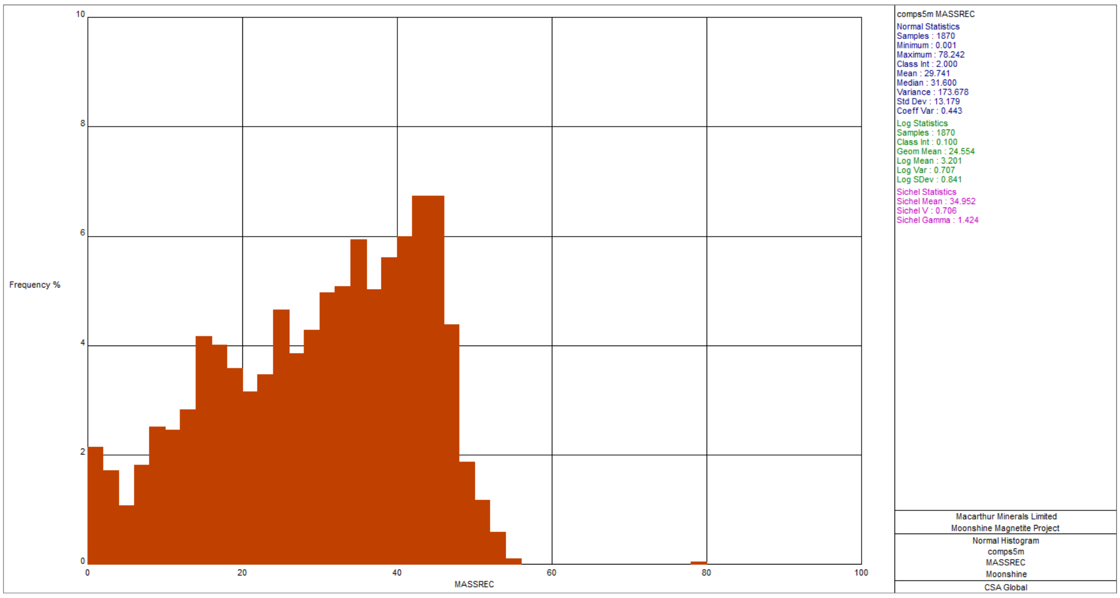


Figure 14-8: Histogram of mass recovery (DTR), from composited samples within mineralisation domains in Moonshine (values in %)

14.3.7. Mass Balance

An analysis of mass data is required to ensure the assayed grade values sum 100%, within a tight tolerance. This has been achieved with a few outliers noted. An example is provided in Figure 14-9, which shows a histogram of the mass balance data for the most populated domain in Moonshine. Most data are between 98% and 102%, with a mean value of 99.5%.

The QP is satisfied that the assay data is of suitable quality, with regards to Mass Balance, to be included in the Mineral Resource estimate.

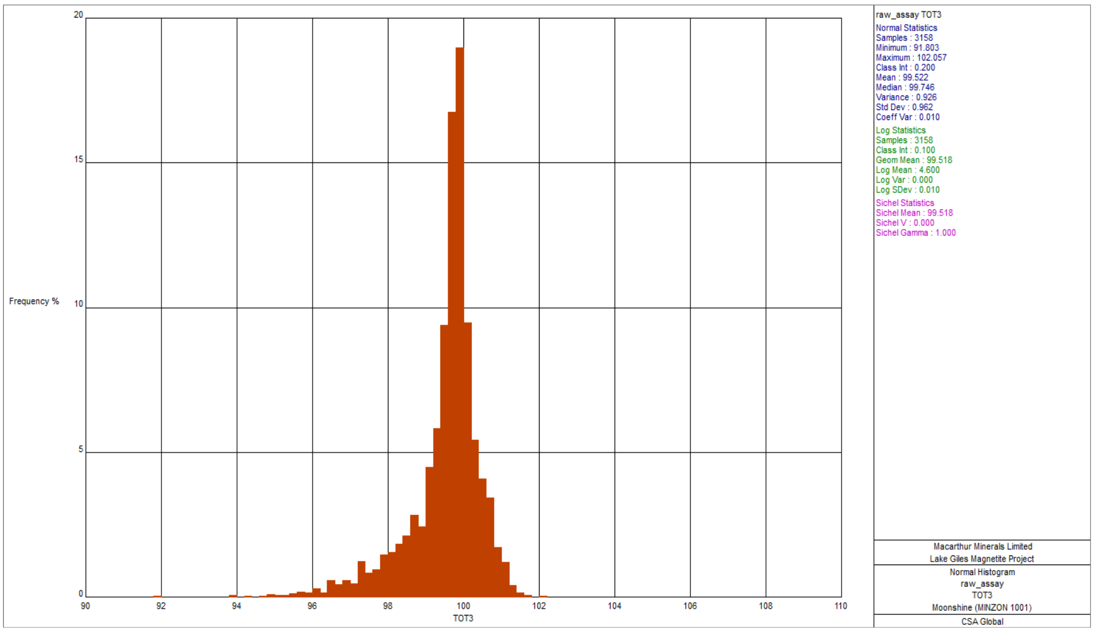


Figure 14-9: Mass Balance for Main Moonshine Domain

14.3.8. Top Cutting of Grades

A review of grade outliers was undertaken to ensure that extreme grades are treated appropriately during grade interpolation. Although extreme grade outliers within the assayed data are real, they are potentially not representative of the volume they inform during estimation. If these values are not cut, they have the potential to result in significant grade over-estimation on a local basis.

Top cuts were determined for selected composited head and concentrate assay grades using the following method:

- A statistical review was undertaken of the grades on a domain-by-domain basis, using the MINZON domain variable
- Log probability and log histograms of the population statistics by domain were reviewed
- Where population breaks at the highest percentile bins are noted, a top cut was selected, and top cut statistics were tabulated; and
- The samples with grades > top cut grades were reviewed in Datamine to determine if they were clustered with other data or located in isolation.

In all cases, the top cut value was equivalent to greater than the 99.5th percentile of data. No bottom cutting of grades was used.

14.3.9. Variography

A variogram is a graph of the variability between pairs of samples against the distance between them in a specific direction. A model is calculated for a particular variogram, which provides parameters known as the nugget, sills and ranges.

The nugget effect is the variability between the closest spaced samples available, which is usually two adjacent samples from the same drillhole.

The nugget value is where the variogram model cuts the Y-axis of the variogram and is usually referred to as a percentage of the total sill. The type of variogram that produces such a variogram is termed a downhole variogram.

As another explanation, the nugget effect is the theoretical variance in grade that would be obtained if a duplicate sample was taken at exactly the same point in space. The nugget effect is an important measure of the reliability/variability of the assay value of samples and is one of the parameters used to determine the weight assigned to individual samples when estimating block grades. A sample population with a low nugget means that more reliability can be placed on nearby individual samples to estimate the grade of a block, such as may be achieved with an “inverse distance weighted” estimate with a high power. Conversely, a grade estimation from a sample population with a very high nugget might require the average grade from a large number of samples be applied as the grade for each block.

The sill is the population variance within a domain and is often normalised to 1.0. The range is the distance at which samples are no longer spatially correlated and can be considered as the point where the variogram model approaches or cuts the sill. This is a subjective decision for which the resource estimator or geostatistician will call on their experience from other projects for the same commodity. More than one sill is often modelled; the first sill (and short range) defines a range of influence up to which the variance between samples may rise very rapidly with increasing distance. Beyond this short range the variability may increase less rapidly with distance until the sill is reached. The short range is often a useful measurement for planning grade control drilling patterns during mining.

Variograms were modelled for selected head and concentrate top cut and composited sample assays located within the most populated mineralisation domain in Moonshine (MINZON 1001). All variograms were modelled capturing a shallow to moderate plunge to the southeast, in the plane of mineralisation. Results are presented in Table 14-6. Selected variogram models are presented in Figure 14-10 to Figure 14-14.

Table 14-6: Variogram Sills and Ranges

Grade variable	Axes	Direction	Nugget	Sill 1	Range 1 (m)	Sill 2	Range 2 (m)	Sill 3	Range 3 (m)
Fe (Head)	1	-29→134	0.04	0.6	83	0.36	266	-	-
	2	-59→337			19		48		-
	3	10→050			38		47		-
SiO_2 (Head)	1	-29→134	0.13	0.48	110	0.39	380	-	-
	2	-59→337			9		26		-
	3	10→050			15		38		-
Al_2O_3 (Head)	1	-20→136	0.15	0.47	78	0.23	184	0.15	435
	2	-68→346			26		64		80
	3	10→050			35		56		62
MgO (Head)	1	-10→138	0.13	0.32	99	0.45	259	0.1	446
	2	-76→005			6		9		17
	3	10→050			41		47		61
P (Head)	1	-29→134	0.14	0.58	99	0.28	570	-	-
	2	-59→337			83		165		-
	3	10→050			25		37		-
S (Head)	1	-10→138	0.11	0.59	92	0.3	359	-	-
	2	-76→005			23		72		-
	3	10→050			37		60		-
LOI (Head)	1	-29→134	0.12	0.6	66	0.28	200	-	-
	2	-59→337			40		73		-
	3	10→050			26		71		-
Fe (Concentrate)	1	-20→136	0.12	0.5	66	0.38	215	-	-
	2	-68→346			11		21		-
	3	10→050			20		65		-
SiO_2 (Concentrate)	1	-29→134	0.13	0.6	91	0.27	316	-	-
	2	-59→337			25		62		-
	3	10→050			40		59		-
Al_2O_3 (Concentrate)	1	-20→136	0.23	0.47	67	0.3	213	-	-
	2	-68→346			30		100		-
	3	10→050			39		48		-
MgO (Concentrate)	1	-20→136	0.16	0.51	85	0.32	219	-	-
	2	-68→346			33		136		-
	3	10→050			46		133		-
P (Concentrate)	1	-59→123	0.08	0.6	53	0.32	211	-	-
	2	-29→326			58		203		-
	3	10→050			47		66		-
S (Concentrate)	1	-59→123	0.08	0.59	44	0.33	138	-	-
	2	-29→326			86		227		-
	3	10→050			42		92		-
	1	-76→095	0.13	0.2	57	0.66	234	-	-

LOI (Concentrate)	2	-10→322			23		45		-
	3	10→050			70		106		-
Mass recovery	1	-39→132	0.1	0.45	28	0.45	190	-	-
	2	-49→332			25		85		-
	3	10→050			24		46		-

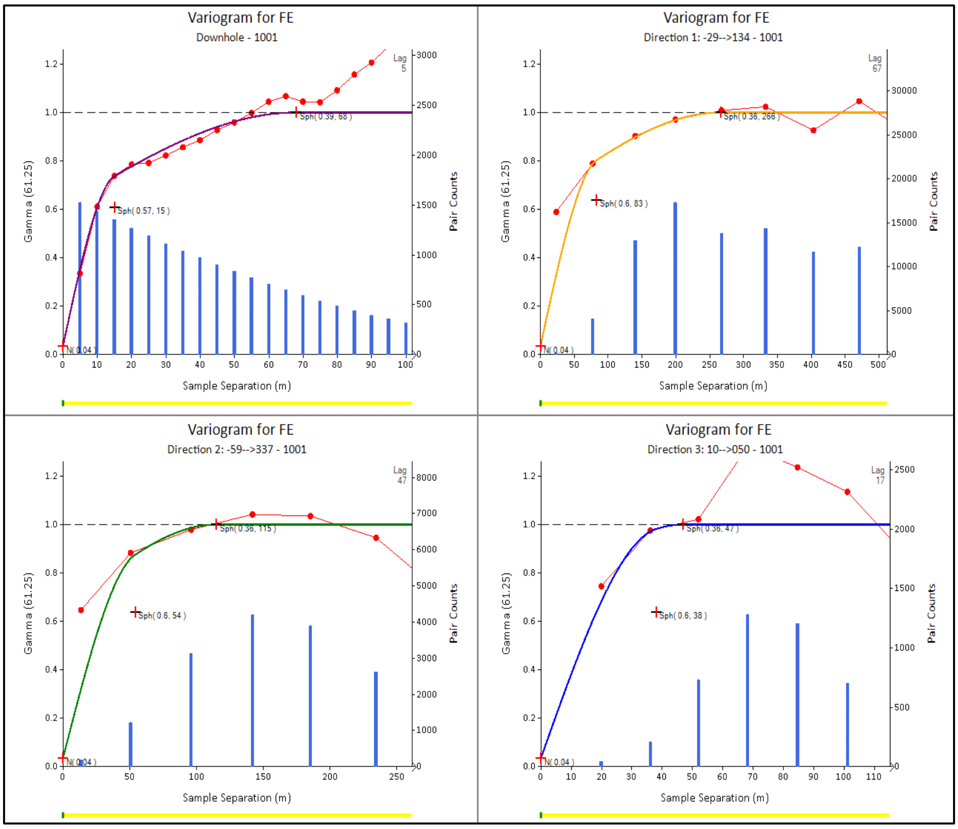


Figure 14-10: Variogram Models for Fe (Head), Domain MINZON 1001 (Moonshine)

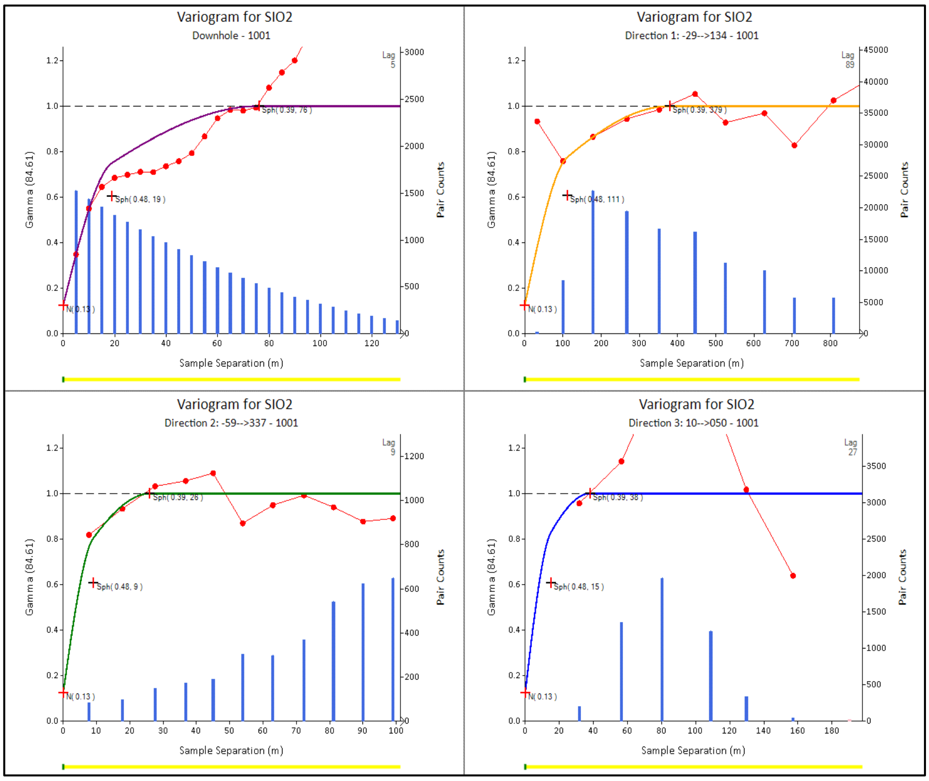


Figure 14-11: Variogram Models for SiO2 (Head), Domain MINZON 1001 (Moonshine)

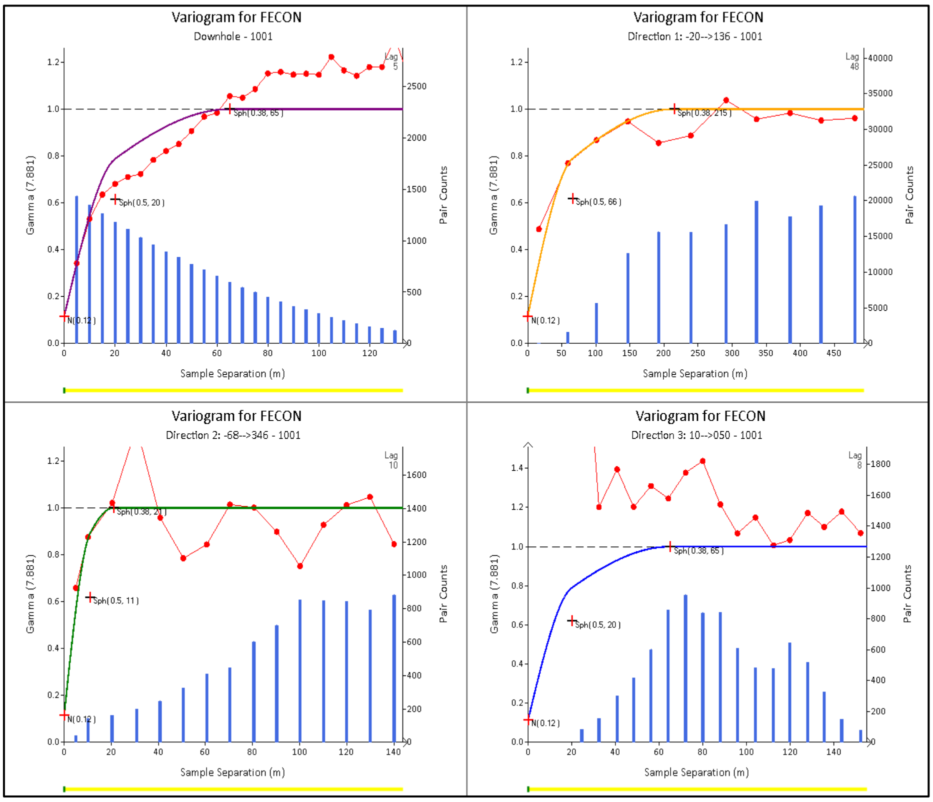


Figure 14-12: Variogram Models for Fe (Concentrate), Domain MINZON 1001 (Moonshine)

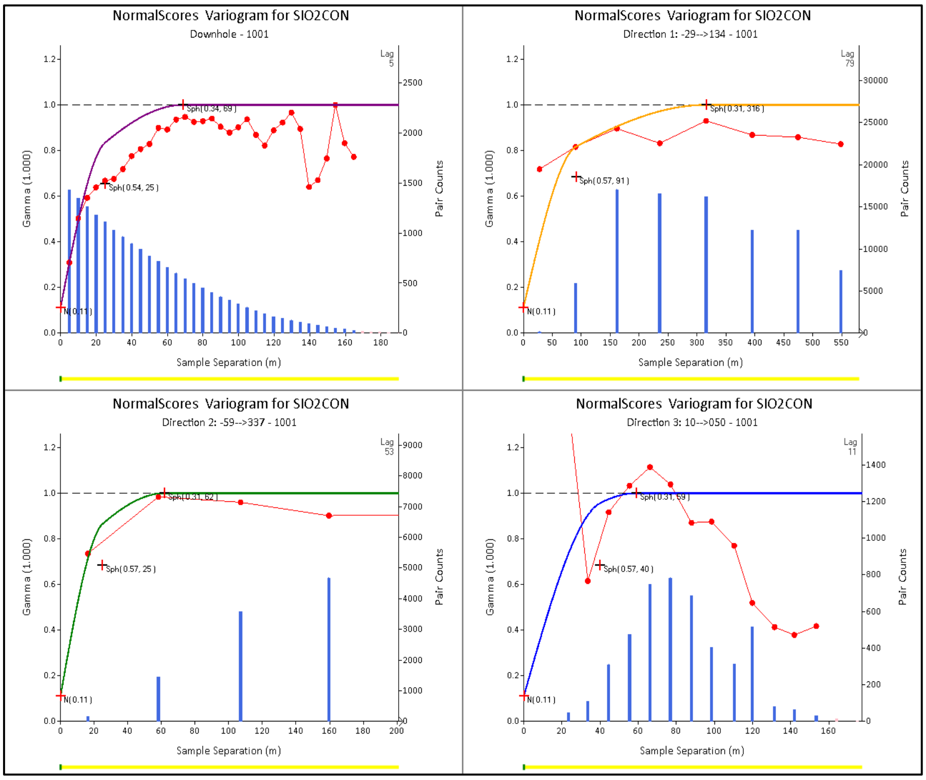


Figure 14-13: Variogram Models for SiO2 (Concentrate), Domain MINZON 1001 (Moonshine)

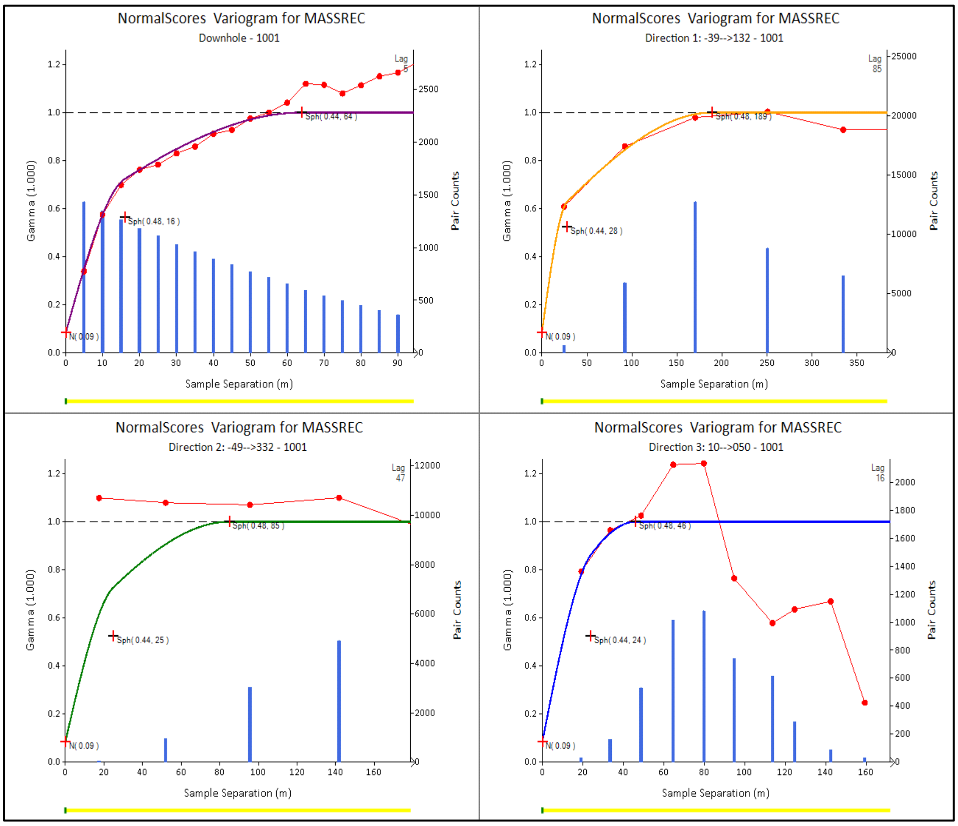


Figure 14-14: Variogram Models for Mass Recovery, Domain MINZON 1001 (Moonshine)

14.3.10. Density

A total of 624 diamond drill samples with bulk density measurements were captured within the mineralisation domains, and a further 400 samples taken from the BIF oxide zones, or from the footwall and hanging wall waste zones. Three mineralisation domains were sampled for Bulk Density data. Figure 14-15 shows a longitudinal section of two of the domains, from Moonshine and Moonshine North, with drillhole intervals containing bulk density data. The grid is 100 m x 100 m and the view to the east. The location of samples used to measure density was later used to guide the Mineral Resource classification (refer Section 0).

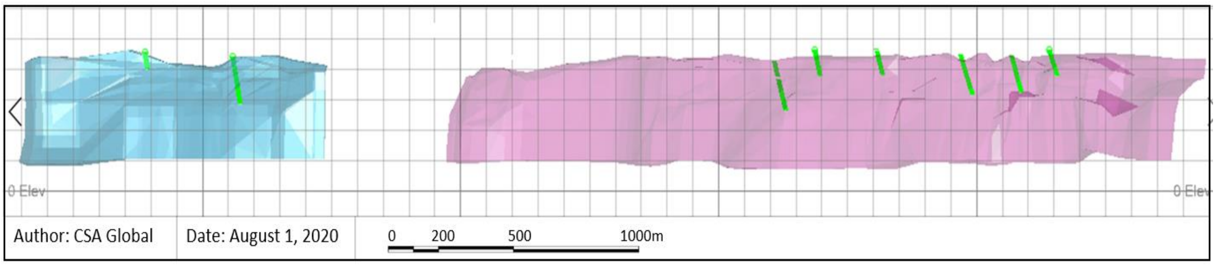


Figure 14-15: Longitudinal section, Moonshine North (blue) and Moonshine (pink) mineralisation domains with drillhole intervals containing bulk density samples (green)

Core samples were sealed prior to immersion in water. A conventional Archimedes wet and dry method was used to measure density, as discussed in Section 10.5.

The drill samples with bulk density data were flagged against the mineralisation and weathering domains, and the bulk density results statistically assessed to determine the mean and ranges, and to see if any excessively low or high bulk density values were present.

Algorithms were developed to calculate the density to apply to the Moonshine and Moonshine North block models based upon correlations between the head iron grade from assays, and the corresponding bulk density value of the sample. A correlation plot for the main mineralised domain at Moonshine is presented in Figure 14-16.

The density algorithms as applied to the Mineral Resources, are given here, where FE is the estimated block grade for Fe (%). The density algorithm for Moonshine was applied to the other Moonshine domains lacking Bulk Density data, and the Moonshine North algorithm was applied to the other Moonshine North domains.

- Moonshine: DENSITY = (0.0241*FE) + 2.624
- Moonshine North: DENSITY = (0.0295*FE) + 2.468
- Moonshine (East): DENSITY = (0.0293*FE) + 2.492
- Unmineralised BIF (oxide): DENSITY = (0.0152*FE) + 2.574
- Unmineralised BIF (fresh): DENSITY = (0.0278*FE) + 2.608; and
- Country rock (basalts, ultramafics): DENSITY = (0.0187*FE) + 2.683.

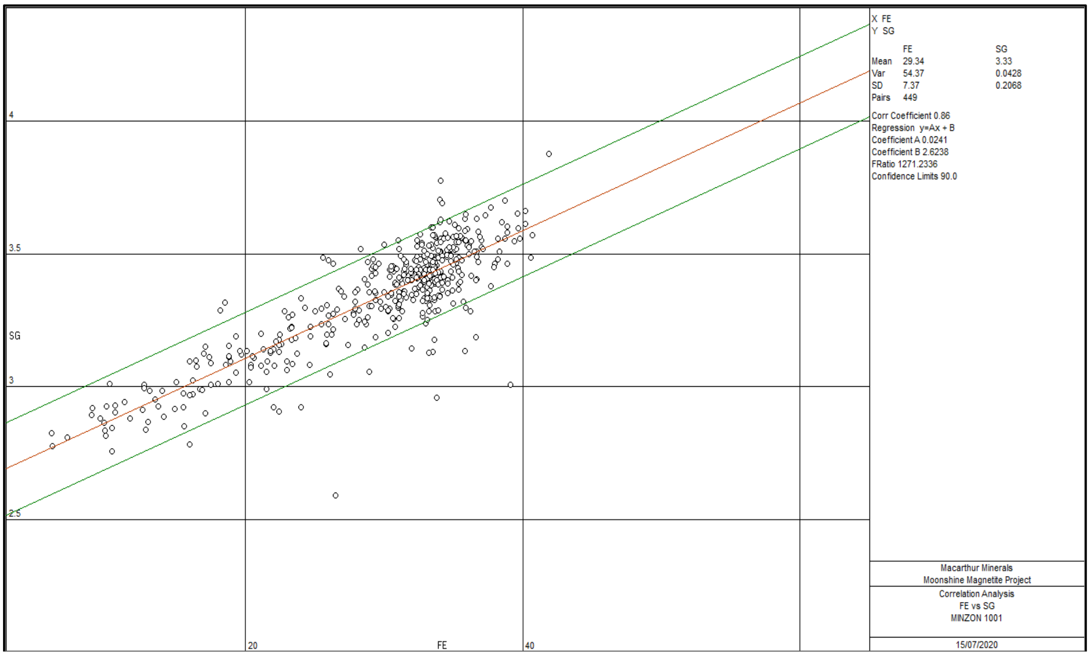


Figure 14-16: Correlation Plot, Fe (Head) vs Specific Gravity (Bulk Density), Mineralisation Domain MINZON 1001 (Moonshine)

14.3.11. Block Model

A block model was created to encompass the full extent of the Moonshine and Moonshine North deposits. Block model parameters are shown in Table 14-7 and block model attributes are shown in Table 14-1.

The block model used a parent cell size of 25 m(E) x 25 m(N) x 10 m(RL) with sub-celling to 2.5 m(E) x 2.5 (m)N x 2 m(RL) to maintain the resolution of the mineralised lenses. The northing parent cell size was selected based on approximately half of the average drill section spacing in better drilled areas of the deposit. The model cell dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip directions.

The volume block models were validated on screen to ensure blocks were coded correctly according to the input wireframes.

Table 14-7: Block Model Dimensions and Parameters

Block model parameters model: ms0720md			
	X	Y	Z
Origin	786,500	6,670,900	50
Extent	5,000	6,200	450
Block size (sub-block)	25 m (2.5 m)	25 m (2.5 m)	10 m (2.5 m)
Rotation	None		
Attributes:			
MINZON	Mineralisation Domain		
WEATH	Weathering Domain. 10 = Oxide, 30 = Fresh		
LITH	Lithological domain BIF = 1		
TOPO	Air = 0, In-situ = 50		
DEPOSIT	Moonshine (west) = 1, Moonshine (east) =2, Moonshine NW = 3, Moonshine NE = 4		
Head grades	Estimated grades (ordinary kriging): Fe, Al_2O_3, CaO, Cr, K_2O, LOI, MgO, MnO, P, S, SiO_2, TiO_2, V		
Concentrate grades	Estimated grade (ordinary kriging): FECON, AL2O3CON, CAOCON, CRCON, K2OCON, LOICON, MGOCON, MNOCON, PCON, SCON, SiO2CON, TiO2CON, VCON		
MASSREC	Estimated mass recovery (DTR) grade (ordinary kriging)		
RESCAT	1 = Measured, 2 = Indicated, 3 = Inferred, 4 = Unclassified		
DENSITY	Calculated or assigned bulk density		

14.3.12. Grade Interpolation

Kriging neighbourhood analysis (KNA) was used to guide the selection of sample search ellipse radii, and the number of samples to be used for each block estimate. The variogram models from the main Moonshine mineralisation domain (Section 14.3.9**Error! Reference source not found.**) were used in the KNA process.

Prior to grade interpolation, the mineralisation domain blocks were interpolated with the local wireframe dip and dip directions using Datamine's dynamic anisotropy. The interpolated values were used to control the orientation of the sample search ellipsoids for grade interpolation.

All head and concentrate grades from top cut and composited data, as detailed in Table 14-8 were interpolated into the parent cells by ordinary kriging. Blocks were estimated using a search ellipse of 240 m (major) x 120 m (semi-major) x 40 m (minor) dimensions, with a minimum of eight and a maximum of 18 samples from a maximum of four samples per drillholes. Search radii were increased, and the minimum number of samples reduced in subsequent sample searches if cells were not interpolated in the first two passes. Cell discretisation of 5 x 5 x 2 (X, Y, Z) was employed.

Hard boundary estimation was used when estimating within the mineralisation domains, such that samples from one mineralisation domain could not be used to interpolate blocks in an adjacent domain.

14.3.13. Block Model Validation

Model validation was carried out graphically and statistically to ensure that block model grades accurately represent the drillhole data. Drillhole cross-sections were examined to ensure that model grades honour the local composited drillhole grades. Representative cross sections through the Moonshine deposit (Figure 14-17) and Moonshine North (Figure 14-18) show the block and drill sample grades coloured by iron. In both examples, mineralisation is shown in the drillhole traces within the oxide weathering zone, above the block model blocks as shown, but these are not considered to be part of the Mineral Resource.

A number of statistical methods were employed to validate the block model, including:

- Comparison of block grade with nearest composites; and
- Comparison of kriged model and composite populations.

Results showed that the grade interpolation had performed as intended, with block grades reasonably reflecting the input sample grades. Validation methods and their results should be reviewed as a package and opinions should not be formed on the performance of the model on one set of data.

Swath plots for MINZON 1001 (Moonshine deposit) are presented in Figure 14-19 to Figure 14-22 from blocks and composited sample grades contained within the domain. Swath plots compare the trend of average grades of the model and input sample data, along a specified direction, from a specified domain. This demonstrates some smoothing of interpolated block grades compared to input sample data, but the sample data trends can be observed in the block grade distribution.

Mean Fe (%) grades from blocks and composited samples (clustered and de-clustered) were compared. The domains are selected where they contain blocks with a first search volume recorded, and only those blocks were used to calculate the mean block grade per domain. Results show a similarity in mean grade for the largest tonnage domains. Some domains show a significant difference between the model and sample mean grades. These domains usually have few samples, and the higher-grade samples are interpreted to have had a disproportional impact upon the volume of the domain, with a large volume of high-grade blocks supported by few samples.

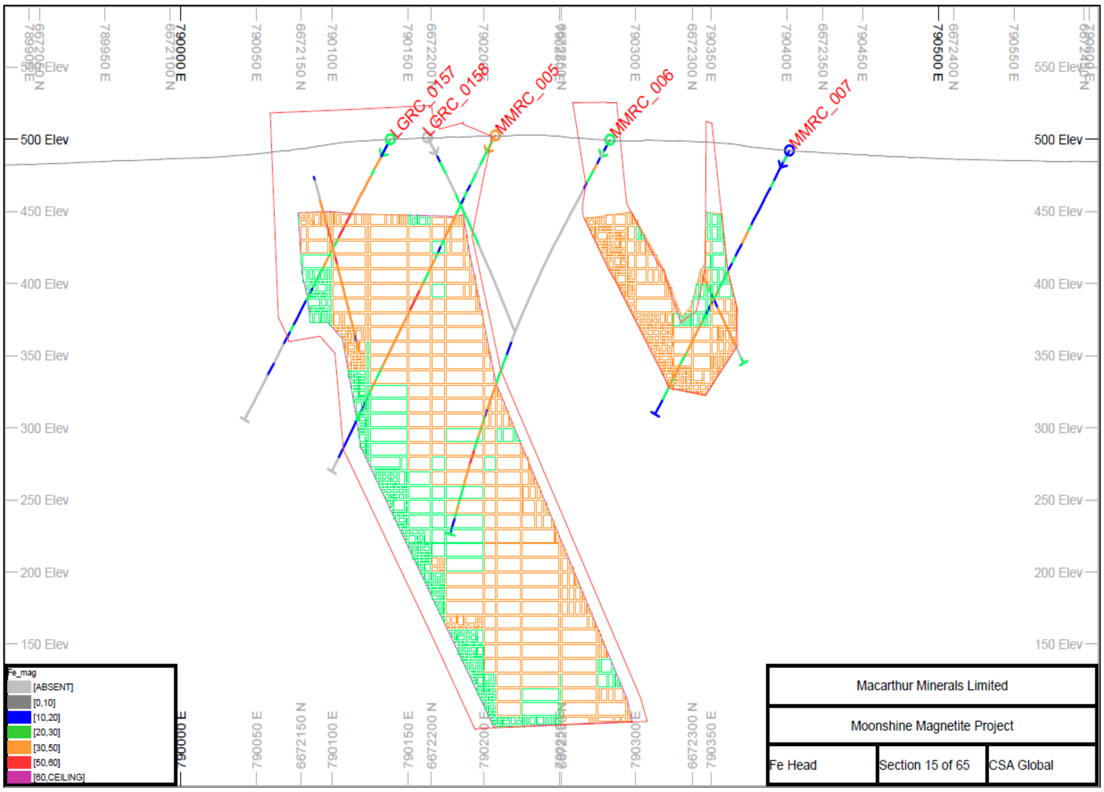


Figure 14-17: Representative cross section through Moonshine showing block model blocks and drillholes coloured by Fe %, with mineralisation domain, BIF domain, and topographic DTM wireframes shown

Note: Oxide domain blocks not shown. Date of image is July 2020.

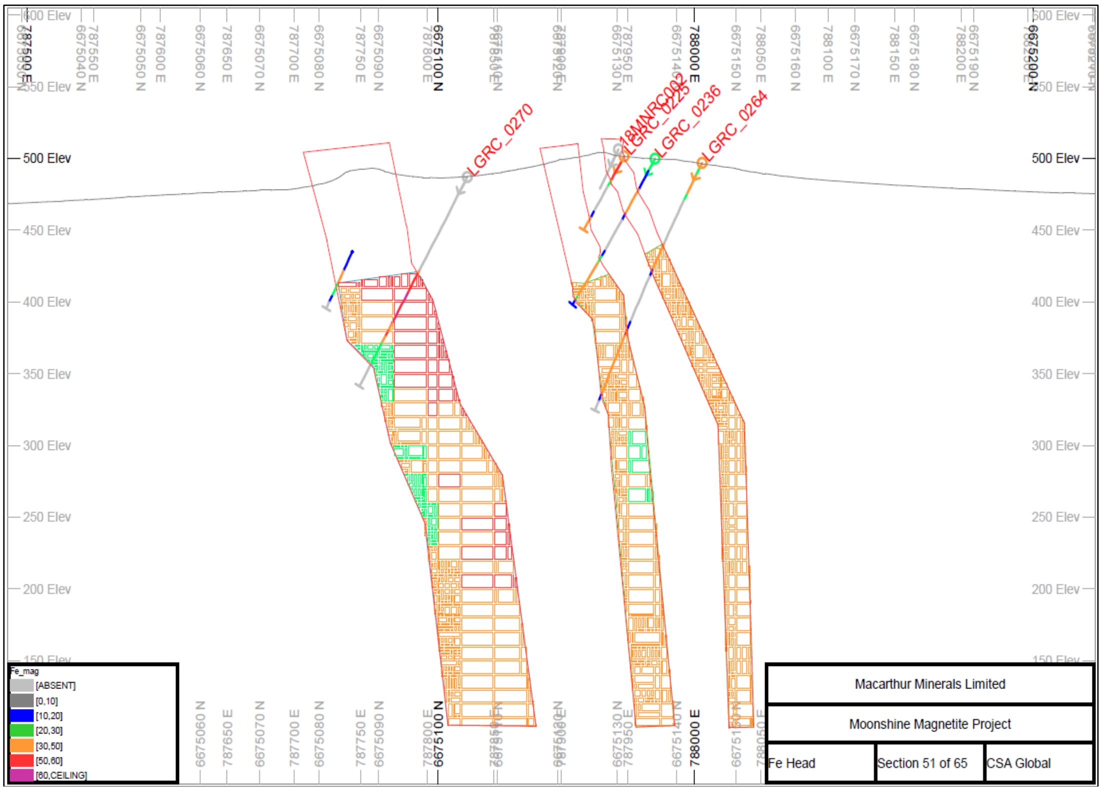


Figure 14-18: Representative cross section through Moonshine North showing block model blocks and drillholes coloured by Fe %, with mineralisation domain, BIF domain, and topographic DTM wireframes shown

Note: Oxide domain blocks not shown. Date of image is July 2020.

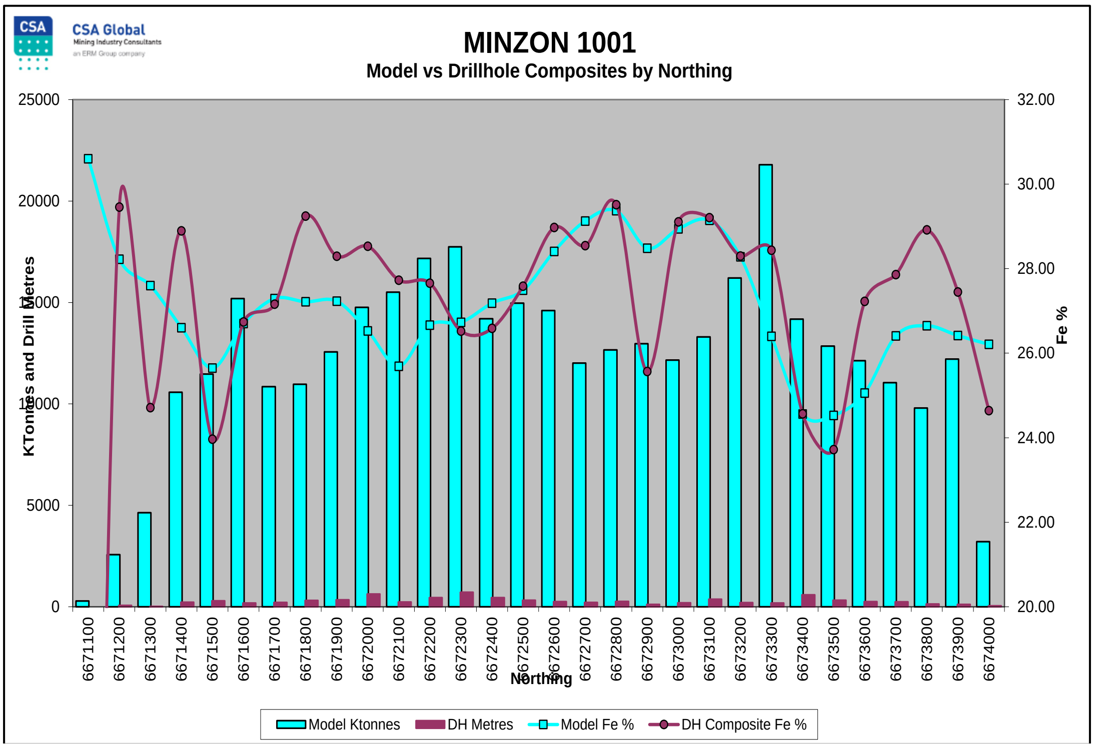


Figure 14-19: Swath Plot, Fe (Head) by Northing, MINZON 1001, Moonshine

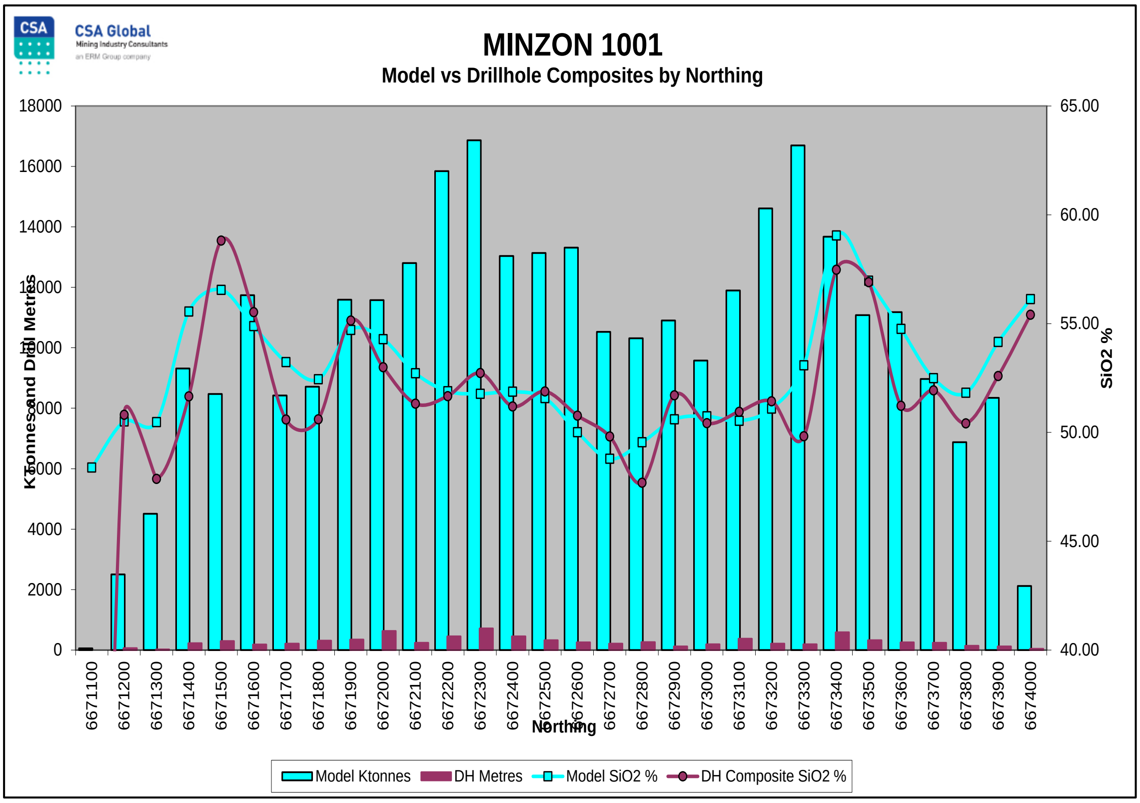


Figure 14-20: Swath Plot, SiO2 (Head) by Northing, MINZON 1001, Moonshine

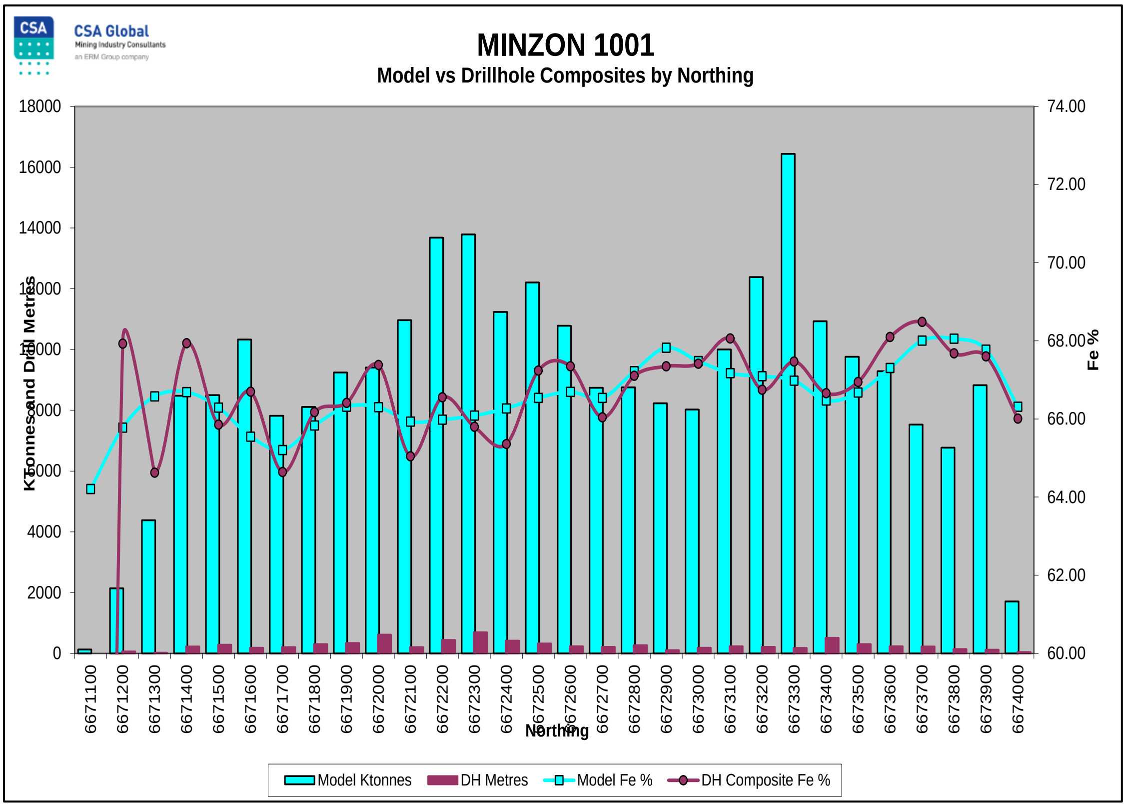


Figure 14-21: Swath Plot, Fe (Concentrate) by Northing, MINZON 1001, Moonshine

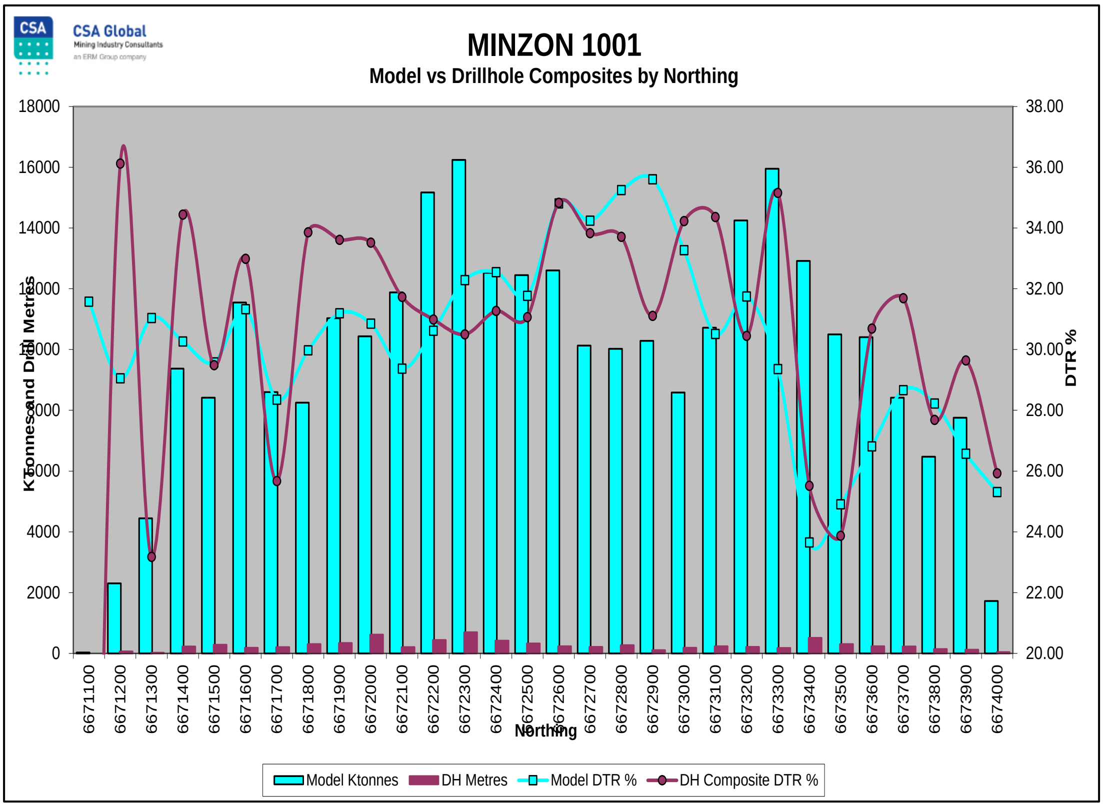


Figure 14-22: Swath Plot, Mass Recovery by Northing, MINZON 1001, Moonshine

14.3.14. Mineral Resource Classification

Classification of the Mineral Resource estimates was carried out considering the geological understanding of the deposit, QAQC of the samples, density data and drillhole spacing.

The Measured Mineral Resources were based upon a confirmed understanding of the geological and grade continuity. Drill spacing is typically 25 m along the northerly strike, with often two to three holes per section. The Measured volumes also contain samples subject to DTR testwork, with associated assays from the recovered concentrates. Bulk density measurements were also available.

The Indicated Mineral Resources were based upon an assumed understanding of the geological and grade continuity. Drill spacing is typically 25–50/100 m along the northerly strike, with at least one hole per section. The Indicated volumes also contain samples subject to DTR testwork, with associated assays from the recovered concentrates. Bulk density measurements may also be available.

The Inferred Mineral Resources were based upon an implied understanding of the geological and grade continuity. Some mineralisation domains are only cut by one drillhole, and the geological models are strongly guided by surface mapping of the BIF outcrops. Drill spacing is typically ≥100 m along the northerly strike. DTR and bulk density results are generally absent from within the Inferred volumes.

Figure 14-23 and Figure 14-24 demonstrate the application of the classification to the Mineral Resource estimate.

All available data was assessed and the QP's relative confidence in the data was used to assist in the classification of the Mineral Resource. The current classification appropriately reflects the QP's view of the deposit.

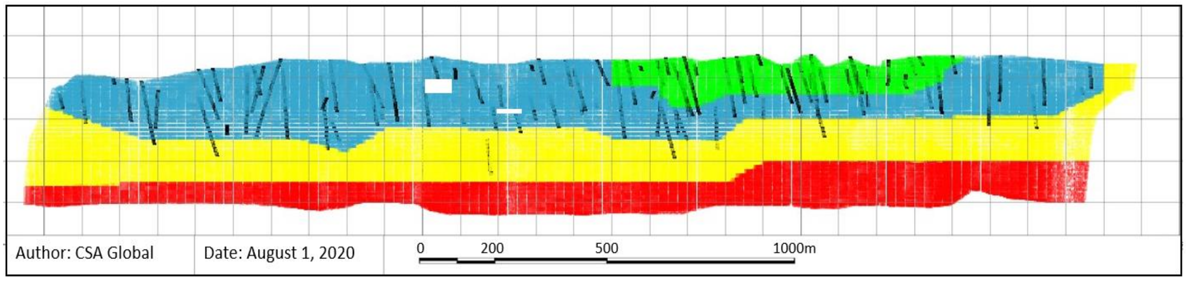


Figure 14-23: Longitudinal Section of Moonshine (west) Domain, Showing Mineral Resource Classification

Note: Green = Measured; cyan = Indicated; yellow = Inferred; red=unclassified; and drillhole intercepts (black traces). Grid square 100 m. View to east.

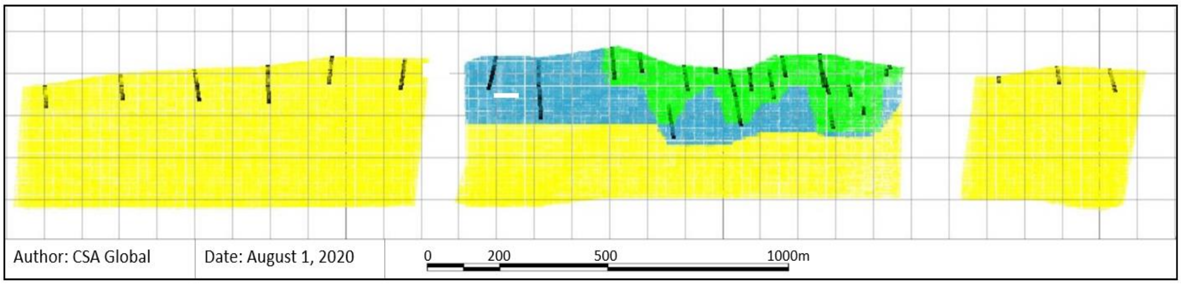


Figure 14-24: Longitudinal Section of Moonshine (west) Domain, Showing Mineral Resource Classification

Note: Green = Measured; cyan = Indicated; yellow = Inferred; and drillhole intercepts (black traces). Grid square 100 m. View to east.

14.3.15. Reasonable Prospects Hurdle

The QP believes there are reasonable prospects for eventual economic extraction of the Mineral Resource.

It is assumed that the Moonshine and Moonshine North Magnetite deposits could be mined by a conventional open cut mining method, followed by crushing and fine grinding and magnetic separation to achieve a magnetite product.

The Project is located 200 km to the northwest of Kalgoorlie-Boulder, which is a regional centre supporting a vibrant mining industry, with a population of approximately 30,000. A sealed road and an all-weather unsealed road allow year-round access to the Project.

Macarthur has been working on a route-to-market for the Project and has confirmed capacity should be available on the rail network owned by Arc Infrastructure. The rail network is located approximately 90 km south of the Project and runs direct for 500 km to the Port of Esperance. The rail network operates on an open access regime and currently services iron ore mines to the west of the Project as detailed in Section 18.

The Port of Esperance is owned by the Western Australian Government and has facilities for iron ore storage and handling with a ship-loader with proven capacity of 12 Mtpa. The Esperance Port is currently handling approximately 6 Mtpa and Macarthur is working towards securing capacity.

The market price for 65% Fe fines is currently over US$140 (A$192) per dry metric tonne at the Effective Date of this Mineral Resource (www.businessinsider.com) and has shown a steady climb in price over the past four years from a low of US$82 (A$ 109) in mid-2017.

The Yilgarn and Midwest regions of Western Australia host a number of similar BIF hosted magnetite deposits including one operational magnetite mine, the Karara magnetite project, operated by Karara

Mining Limited. The QP has undertaken a review of the Mineral Resource estimates and operating assumptions presented in scoping studies and publicly reported information to the ASX for the Karara magnetite project, Mount Ida magnetite project (Jupiter Mines Limited), Telecom Hill iron ore deposit, (Austsino Resources Group Limited) and Yerecoin magnetite deposit (Cliffs) in the Yilgarn Craton. These projects are considered analogous to the Lake Giles Iron Project in respect of deposit style, geographical location, Mineral Resource estimation and reporting criteria.

Mineral Resources of these projects were reported to a similar depth as the Inferred Mineral Resources of the Lake Giles Iron Project with estimated Free on Board costs in the range of A$57 to A$90 for open pit mining scenarios. Cut-off grades and DTR parameters presented below are in line with the those used for the Lake Giles Iron Project:

- Mount Ida: 10% magnetite Fe cut off; DTR P80 25 micron (SRK, 2018)
- Yerecoin: 15% DTR cut-off; DTR P85 75 micron (Cliffs, 2012)
- Telecom Hill: 15% DTR cut-off; DTR P80 38 micron (Austsino, 2017); and
- Karara: 20% DTR cut-off; DTR P80 35 micron (Gindalbie Metals, 2007).

The Karara magnetite Mineral Resource was reported in accordance with the JORC Code (2004), with reporting based on a DTR mass recovery above 20% and reported to a depth of 400 m below surface (Gindalbie Metals, 2007). The cut-off parameter of 20% DTR is marginally above the cut-off used for the Lake Giles magnetite deposits. Iron head grades and concentrate iron grades are considered in line with estimates reported for the Lake Giles Iron Project. Metallurgical testwork for the Karara project was based on a grind size of 80% passing 35 microns to achieve a product concentrate grade of 68.2% Fe. This grind size is slightly finer than the DTR testwork for the Lake Giles Iron Project at P80 45 microns that reported a concentrate grade ranging between 62.4% and 66.1%. These grades are considered within required ranges to achieve the specifications for the iron ore fines market.

The Karara project commenced mining in 2011 and is currently producing magnetite concentrate for export through Geraldton Port in Western Australia. The project logistics, geographical setting and deposit style are considered analogous to the Lake Giles Iron Project.

Macarthur is not aware of any significant environmental reasons why environmental approval is unlikely to be granted for the Project.

Tenure over the property is granted for at least another eight years with the option to extend, and annual expenditure payments have been diligently paid by Macarthur. The Australian system of government is very stable, with the major political parties supportive of the mining industry. Mining of iron mineralisation in Western Australia is a major contributor to the State's economy and the development of iron projects is supported at a government level, assuming all relevant approvals can be obtained.

The QPs are not aware of any potential issues regarding environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors, that could materially affect the Mineral Resource estimate.

14.3.16. Reporting of Mineral Resource Estimate

Mineral Resources are reported at an Effective Date of 29 September 2020.

Mineral Resources for Moonshine and Moonshine North are shown in Tables 14-8 to 14-10. Mineral Resources are reported above a DTR cut-off of 15%.

This cut-off is also the domain cut-off. The DTR cut-off is required to ensure a higher volume of magnetite-bearing mineralisation is selected, removing the rock volumes with low magnetite content, such as the siliceous bands within the magnetite-bearing rock (BIF).

Table 14-8: Mineral Resource Estimate, Moonshine and Moonshine North, where DTR >15%

Category	Tonnes (Mt)	Head grade (%)					Concentrate grade (%)					
		Fe	P	SiO_2	Al_2O_3	LOI	DTR	Fe	P	SiO_2	Al_2O_3	LOI
Measured	53.9	30.8	0.05	45.4	1.6	2.7	32.2	66.0	0.031	6.2	0.2	-0.7
Indicated	218.7	27.5	0.046	51.1	1.4	1.6	31.0	66.1	0.017	6.7	0.1	-0.1
Subtotal	**272.5**	**28.1**	**0.047**	**50.0**	**1.4**	**1.8**	**31.2**	**66.1**	**0.02**	**6.6**	**0.2**	**-0.2**
Inferred	449.1	27.1	0.047	52.6	1.0	1.4	29.2	65.0	0.026	8.4	0.1	0

Table 14-9: Mineral Resource Estimate, Moonshine, where DTR >15%

Category	Tonnes (Mt)	Head Grade (%)					Concentrate Grade (%)					
		Fe	P	SiO_2	Al_2O_3	LOI	DTR	Fe	P	SiO_2	Al_2O_3	LOI
Measured	34.4	28.2	0.045	51.5	1.2	1.7	30.6	65.8	0.013	6.9	0.2	-0.6
Indicated	193.0	27.1	0.045	52.1	1.4	1.4	30.5	66.5	0.014	6.3	0.1	0.0
Subtotal	**227.4**	**27.3**	**0.045**	**52.0**	**1.4**	**1.4**	**30.5**	**66.4**	**0.014**	**6.4**	**0.1**	**-0.1**
Inferred	167.5	27.0	0.047	52.4	1.3	1.4	30.4	66.0	0.016	7.2	0.1	0.0

Table 14-10: Mineral Resource Estimate, Moonshine North, where DTR >15%

Category	Tonnes (Mt)	Head Grades (%)					Concentrate Grade (%)					
		Fe	P	SiO_2	Al_2O_3	LOI	DTR	Fe	P	SiO_2	Al_2O_3	LOI
Measured	19.5	35.3	0.060	34.7	2.5	4.3	34.9	66.4	0.062	5.0	0.3	-0.9
Indicated	25.7	30.5	0.050	43.6	1.4	3.1	35.2	63.5	0.041	9.1	0.2	-0.5
Subtotal	**45.2**	**32.6**	**0.055**	**39.8**	**1.9**	**3.6**	**35.1**	**64.7**	**0.050**	**7.3**	**0.3**	**-0.7**
Inferred	281.7	27.1	0.048	52.7	0.8	1.4	28.5	64.5	0.033	9.1	0.1	0.0

Notes:

- The Mineral Resource estimate was prepared by David Williams, B.Sc., MAIG, a CSA Global employee, and the Qualified Person for the estimate
- Mineral Resources were estimated using Datamine Studio RM (Version 1.6.87).
- Assays were composited to regular 1 m or 5 m intervals, dependent upon the deposit.
- Composite assay grades were capped as required. Fe and DTR grades were not capped.
- Block-model grade interpolation was undertaken using ordinary kriging.
- Bulk density was calculated for each block in the Moonshine model using algorithms, based upon the estimated Head Fe block grade. Average bulk density of 3.3 t/m3 was applied to the other deposit models.
- Mineral Resources are reported from a model with parent block dimensions of 25 m x 25 m x 10 m.
- Tonnage and grade have been rounded to reflect the relative accuracy of the Mineral Resource estimate; therefore, columns may not total due to rounding.
- Resource classification is as defined by the Canadian Institute of Mining, Metallurgy and Petroleum in their document “CIM Definition Standards for Mineral Resources and Mineral Reserves” of 10 May 2014.
- Mineral Resources that are not Mineral reserves do not have demonstrated economic viability.
- The QP and Macarthur are not aware of any current environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might materially affect these Mineral Resource estimates.

14.4. Sandalwood, Clark Hill North, Clark Hill South and Snark

14.4.1. Drillhole Database

The drillhole data was provided by Macarthur in a Microsoft Access format comprising collar, survey, assay, lithological and weathering data. Drillhole statistics are presented in Table 14-11. The database was provided to CSA on 20 October 2009, with no additional data related to the Lake Giles Iron Project, excluding the Moonshine and Moonshine North deposits, provided thereafter.

Table 14-11: Drilling Data as of 20 October 2009

Deposit	No. of holes	No. of samples	Total metres
Clark Hill North	53	1,511	8,589
Clark Hill South	5	215	1,270
Sandalwood	27	1,029	6,050
Snark	16	487	2,969
Total	**101**	**3,242**	**18,878**

Assay data presenting as negative values were treated as missing samples, and the assay grades set to absent. All assay fields were set to their appropriate oxidation state during importation of data from the laboratory certificates into the database.

Drillhole database tables were imported into Datamine and checks carried out for erroneous drillhole collars, sample overlaps, and any missing data. A drillhole file was created in Datamine capturing collar, survey, assay, geology, DTR and density data.

Drillhole collar plot for the deposits are presented in Section 10.1.

14.4.2. Topography

A DTM of the topography, imported from contour maps of the Project area, was imported into Datamine and saved as a wireframe surface.

The surface was validated against several drill collars, representing different geographical locations of the resource, to ensure matching elevation levels between drillhole survey and topographic survey. The topographic survey is considered adequate to support the Mineral Resource estimates.

14.4.3. Geological Interpretation

Mineralisation domains were interpreted and modelled in cross section, using drillhole logging and sample analyses to guide the interpretation. The interpretation and wireframes were generated based on a 100 m x 50 m and 200 m x 100 m exploration drilling patterns.

Wireframe solids were generated based on the sectional interpretations provided by Macarthur to delineate the mineralisation domains. A lower cut-off of 15% Fe combined with the geological logging was used to define the mineralised envelopes.

A base of oxidation surface was modelled using the geological logging, magnetic susceptibility of drill samples, and the mass recovery results from the sample analyses. Mineral Resources are only reported below the base of oxidation.

14.4.4. Sample Coding

Drillhole samples within the Datamine drillhole files were flagged with unique codes according to the mineralisation and weathering domain within which they were located.

14.4.5. Sample Compositing

Analysis of the exploration data intervals showed the majority of the raw sample intervals are between 1 m to 5 m in length, but there are a number of non-regular sample data. The raw samples range in length from 0.18 m to 12.0 m, with about 45% being 5 m. The 5 m length was considered appropriate for compositing to retain the original data variability. Use of this composite size minimised splitting of raw samples to smaller intervals.

Compositing was completed to honour the geological boundaries of the mineralised lodes by breaking the composites at the lode boundaries. This process resulted in sample lengths of less than 5 m at lode contacts. Approximately 1% of the composites have a length less than or equal to 1.1 m.

14.4.6. Statistical Analyses

Statistical summaries for key grades from composited sample data within the mineralisation domains are presented in Table 14-12 to Table 14-19. The data show the result of separating the magnetite from the whole sample using the Davis Tube method, with Fe grades significantly higher in the product, and a corresponding material decrease in silica and phosphorous grades. The higher silica and phosphorous grades are associated with silica and non-magnetic minerals caught in the gangue material, such as the siliceous bands in the host BIF rock.

Table 14-12: Summary statistics, head grades – Sandalwood (values in %); P2O5 values were stoichiometrically adjusted to P at this stage of work

Statistic	Al_2O_3	Fe	LOI	P_2O_5	S	SiO_2
Number	281	281	281	281	281	281
Minimum	0.02	12.1	-1.8	0.07	0.002	25.1
Maximum	11	47.3	4.1	0.3	2.6	60.2
Mean	1.58	30.85	-0.63	0.16	0.18	48.4
Standard deviation	2.33	5.66	0.92	0.027	0.3	4.02
Variance	5.41	32.05	0.85	0.001	0.09	16.19
Coefficient of variation	1.47	0.18	-1.47	0.18	1.62	0.08

Table 14-13: Summary Statistics, Concentrate Grades – Sandalwood (values in %)

Statistic	Al_2O_3	Fe	LOI	P	S	SiO_2	DTR
Number	275	275	275	275	275	275	281
Minimum	0.005	51.7	-3.7	0.005	0.002	1.5	0.59
Maximum	0.7	70.9	0.2	0.079	6.8	25.9	57.2
Mean	0.069	64.7	-2.77	0.031	0.27	9.47	33.0
Standard deviation	0.095	3.45	0.5	0.014	0.62	4.62	11.35
Variance	0.009	11.9	0.25	0.001	0.39	21.32	128.9
Coefficient of variation	1.37	0.05	-0.18	0.442	2.3	0.49	0.34

Table 14-14: Summary Statistics, Head Grades – Clark Hill North (values in %)

Statistic	Al_2O_3	Fe	LOI	P	S	SiO_2
Number	443	443	421	243	440	443
Minimum	-0.01	3.3	-3.58	0.021	-0.001	25.6
Maximum	14.6	41.5	17.1	0.137	4.9	65.6
Mean	1.97	28.3	0.27	0.063	0.28	47.1
Standard deviation	2.88	9.36	2.6	0.022	0.54	4.39
Variance	8.29	87.6	6.74	0.001	0.29	19.26
Coefficient of variation	1.46	0.33	9.69	0.36	1.91	0.09

Table 14-15: Summary Statistics, Concentrate Grades – Clark Hill North (values in %)

Statistic	Al_2O_3	Fe	LOI	P	S	SiO_2
Number	262	262	-	189	261	262
Minimum	0.005	40.3	-	0.003	0.001	2.1
Maximum	4.96	70.7	-	0.14	2.76	26.9
Mean	0.17	63.2	-	0.042	0.26	10.8
Standard deviation	0.37	4.56	-	0.024	0.48	5.4
Variance	0.14	20.75	-	0.001	0.23	29.4
Coefficient of variation	2.13	0.07	-	0.57	1.9	0.5

Table 14-16: Summary Statistics, Head Grades – Clark Hill South (values in %)

Statistic	Al_2O_3	Fe	LOI	P	S	SiO_2
Number	11	11	11	11	11	11
Minimum	0.09	20.13	0.00	0.05	0.00	43.69
Maximum	4.75	35.88	0.16	0.07	0.27	49.62
Mean	0.64	32.63	0.02	0.06	0.08	47.06
Standard deviation	1.37	4.51	0.05	0.01	0.11	2.06
Variance	1.89	20.30	0.00	0.00	0.01	4.22
Coefficient of variation	2.17	0.14	3.10	0.11	1.32	0.04

Table 14-17: Summary Statistics, Concentrate Grades – Clark Hill South (values in %)

Statistic	Al_2O_3	Fe	LOI	P	S	SiO_2
Number	11	11	-	11	11	11
Minimum	0.02	59.40	-	0.01	0.00	5.50
Maximum	0.27	67.00	-	0.03	0.16	15.80
Mean	0.08	62.35	-	0.02	0.05	12.26
Standard deviation	0.07	2.47	-	0.01	0.06	3.24
Variance	0.01	6.10	-	0.00	0.00	10.52
Coefficient of variation	0.90	0.04	-	0.29	1.23	0.27

Table 14-18: Summary Statistics, Head Grades – Snark (values in %)

Statistic	Al_2O_3	Fe	LOI	P	S	SiO_2
Number	119	119	81	109	119	119
Minimum	0.1	9.9	-0.9	0.035	0.008	31.3
Maximum	12.6	40.2	8.7	0.125	1.1	69.6
Mean	1.92	28.8	2.03	0.067	0.161	47.9
Standard deviation	2.65	6.0	1.82	0.014	0.18	5.3
Variance	7.0	36.3	3.31	0.000	0.03	28.4
Coefficient of variation	1.38	0.21	0.9	0.20	1.12	0.11

Table 14-19: Summary Statistics, Concentrate Grades – Snark (values in %)

Statistic	Al_2O_3	Fe	LOI	P	S	SiO_2	DTR
Number	69	69	66	69	69	69	67
Minimum	0.008	55.5	-3.4	0.016	0.001	2.44	0.74
Maximum	0.81	70.9	-2.1	0.066	2.1	21.4	37.0
Mean	0.15	66.3	-2.83	0.028	0.33	7.23	24.3
Standard deviation	0.15	3.36	0.3	0.009	0.38	4.16	8.43
Variance	0.02	11.3	0.09	0.00	0.14	17.3	71.0
Coefficient of variation	0.96	0.05	-0.11	0.32	1.13	0.57	0.35

14.4.7. Top Cutting of Grades

Top cuts were applied to the Sandalwood, Clark Hill North, Clark Hill South, or Snark composited sample assays, where appropriate. Top cuts were selected and applied if there was an extended high-grade tail on the histogram of results within the mineralisation domains.

The samples with grades greater than the nominated top cut grades values were reviewed in Datamine to determine if they were clustered with other data or located in isolation.

No bottom cutting of grades was used.

14.4.8. Variography

A discussion on the use of variography is provided in Section 14.3.9

Variograms for Sandalwood top cut and composited sample data were modelled after combining all mineralisation domains into a single population due to the low count of sample numbers. Traditional semi-variograms were modelled.

Variograms were modelled from head and concentrate assays for Fe, SiO2, Al2O3, LOI and S, with a variogram also modelled for the P (concentrate) and DTR sample data. The variograms were poorly structured due to very sparse data. Results are presented in Table 14-20. Due to low numbers of samples for the other deposits, it was not possible to model variograms. The Sandalwood variogram parameters were applied to these models during grade interpolation.

Table 14-20: Variogram Sills and Ranges, Sandalwood

Grade variable	Axes	Direction	Nugget	Sill 1	Range 1 (m)	Sill 2	Range 2 (m)
Fe (Head)	1	80→040	0.55	0.2	49	0.25	155
	2	10→220			16		83
	3	0→130			44		144
SiO_2 (Head)	1	80→040	0.55	0.21	260	0.24	420
	2	10→220			77		175
	3	0→130			85		200
Al_2O_3 (Head)	1	50→040	0.22	0.4	50	0.38	128
	2	40→220			50		55
	3	0→130			400		620
LOI (Head)	1	80→060	0.28	0.23	36	0.49	131
	2	10→240			128		410
	3	0→150			450		810
S (Head)	1	50→040	0.28	0.29	24	0.43	116
	2	40→220			40		100
	3	0→130			49		150
Fe (Concentrate)	1	80→040	0.16	0.54	28	0.3	58
	2	10→220			120		240
	3	0→130			265		333
P (Concentrate)	1	50→040	0.18	0.51	54	0.31	160
	2	40→220			34		95
	3	0→130			42		164
SiO_2 (Concentrate)	1	80→050	0.22	0.4	30	0.38	104
	2	10→230			50		89
	3	0→130			105		230
Al_2O_3 (Concentrate)	1	80→040	0.35	0.11	26	0.54	86
	2	10→220			118		340
	3	0→130			72		370
S (Concentrate)	1	70→080	0.55	0.19	80	0.26	133
	2	20→260			180		365
	3	0→170			630		630
LOI (Concentrate)	1	80→040	0.1	0.35	24	0.55	77
	2	10→220			63		255
	3	0→130			63		255
Mass recovery	1	80→040	0.45	0.29	40	0.26	88
	2	10→220			18		93
	3	0→130			18		93

14.4.9. Density

Density measurements were taken from drill sample data located at Clark Hill. A total of 122 diamond core billets were taken from four diamond holes, with 63 of the samples located within the BIF host rock.

Density measurements were taken using a conventional Archimedes technique. Discussion is provided in Section 10.5.

A review of results by Allen (2009) showed some very low values and some very high results, which were excluded from the dataset. A statistical analysis determined an average density value of 3.3 t/m3 for all samples, and 3.4 t/m3 for BIF samples. The global mean results of 3.3 t/m3 was applied to the Mineral Resource block model and is a typical density value for magnetite mineralisation hosted by BIF. The QP considered there to be insufficient number of samples to model an algorithm for density, as was used for the Moonshine Mineral Resource (Section 14.3.10).

14.4.10. Block Model

Separate block models were prepared for Sandalwood, Clark Hill North, Clark Hill South, and Snark.

Block model parameters are shown in Table 14-21.

The northing parent cell sizes were selected based on approximately half of the average drill section spacing. The model cell dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip directions.

The volume block models were validated on screen to ensure blocks were coded correctly according to the input wireframes.

Table 14-21: Block Model Dimensions and Parameters

	X	Y	Z
Sandalwood krgmod2d			
Origin	788,400	686,800	180
Extent	4,100	9,700	340
Block size (sub-block)	50 m (5 m)	50 m (5 m)	10 m (2 m)
Clark Hill North krgmod2d			
Origin	783,800	694,400	180
Extent	4,200	2,100	340
Block size (sub-block)	50 m (5 m)	50 m (5 m)	10 m (2 m)
Clark Hill South chs_v2			
Origin	786,000	691,500	150
Extent	2,400	2,700	400
Block size (sub-block)	10 (1)	100 (2)	50 (2)
Snark krgmod2d			
Origin	780,900	695,500	180
Extent	5,500	4,700	340
Block size (sub-block)	50 m (5 m)	50 m (5 m)	10 m (2 m)
Attributes:			
MINZON	Mineralisation Domain		
OXID	Weathering Domain. 0 = Oxide, 3 = Fresh		
PARTORE [1]	Used to apply tonnage reduction during reporting (Sandalwood only)		
Head grades	Estimated grades (ordinary kriging): Fe, Al_2O_3, LOI, P, S, SiO_2		

Concentrate grades	Estimated grade (ordinary kriging): FE_C, AL2O3_C, LOI_C, P_C, S_C, SiO2_C
DTR	Estimated mass recovery (DTR) grade (ordinary kriging)
CLASS	1 = Measured, 2 = Indicated, 3 = Inferred, 4 = Unclassified
DENSITY	Calculated or assigned bulk density

Note: PARTORE was used to limit the volumes of modelled mineralisation that could be reported as Mineral Resources. Only those volumes within 200 m of a drillhole were reported.

14.4.11. Grade Interpolation

All head and concentrate grades from top cut and composited data, as detailed in Table 14-7, were interpolated into the parent cells by ordinary kriging. Blocks were estimated using a search ellipse of 300 m (major) x 100 m (semi-major) x 100 m (minor) dimensions, with a minimum of 12 and maximum of 30 samples from a maximum of six samples per drillholes. Search radii were increased, and the minimum number of samples reduced in subsequent sample searches if cells were not interpolated in the first two passes. Cell discretisation of 3 x 3 x 3 (X, Y, Z) was employed.

Hard boundary estimation was used when estimating within the mineralisation domains, such that samples from one mineralisation domain could not be used to interpolate blocks in an adjacent domain.

14.4.12. Block Model Validation

Model validation was carried out graphically and statistically to ensure that block model grades accurately represent the drillhole data. Drillhole cross-sections were examined to ensure that model grades honour the local composited drillhole grades.

A number of statistical methods were employed to validate the block model, including:

- Comparison of block grade with nearest composites
- Comparison of kriged model and composite populations.

Results showed that the grade interpolation had performed as intended, with block grades reasonably reflecting the input sample grades. Validation methods and their results should be reviewed as a package and opinions should not be formed on the performance of the model on one set of data.

A swath plots for the estimated DTR block grades and input composited sample data for Sandalwood is presented in Figure 14-25. This demonstrates some smoothing of interpolated block grades compared to input sample data, but the sample data trends can be observed in the block grade distribution.

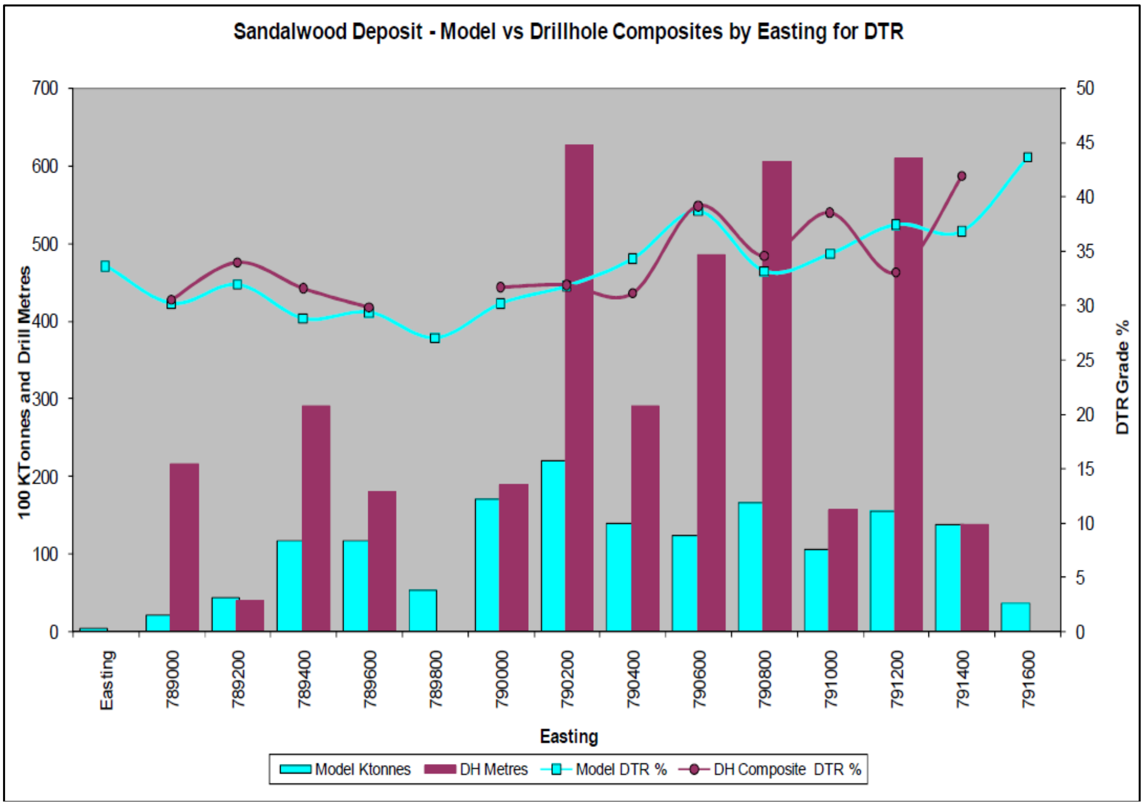


Figure 14-25: Swath Plot, DTR, by Easting – Sandalwood

14.4.13. Mineral Resource Classification

Classification of the Mineral Resource estimates was carried out taking into account the geological understanding of the deposit, QAQC of the samples, density data and drillhole spacing. Sandalwood, Clark Hill North, Clark Hill South, and Snark are classified as Inferred.

The Inferred Mineral Resource classification is based upon an implied understanding of the geological and grade continuity. Some mineralisation domains are only cut by one drillhole, and the geological models are strongly guided by surface mapping of the BIF outcrops.

All available data was assessed and the QP's relative confidence in the data was used to assist in the classification of the Mineral Resource.

The current classification assignment appropriately reflects the QP's view of the deposits.

14.4.14. Reasonable Prospects Hurdle

The QP believes there are reasonable prospects for eventual economic extraction of the Mineral Resource.

It is assumed the deposits could be mined by a conventional open cut mining method, followed by crushing and fine grinding and magnetic separation to achieve a magnetite product.

The Project is located 200 km to the northwest of Kalgoorlie-Boulder, which is a regional centre supporting a vibrant mining industry, with a population of approximately 30,000. A sealed road and an all-weather unsealed road allow year-round access to the Project.

Other relevant discussion is provided in Section 0.

The QP is not aware of any potential issues regarding environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors, that could materially affect the Mineral Resource estimate.

14.4.15. Reporting of Mineral Resource Estimate

Mineral Resources are reported at an Effective Date of 29 September 2020.

Mineral Resources for Sandalwood, Clark Hill North, Clark Hill South, and Snark are shown in Table 14-22. Mineral Resources are reported above a DTR cut-off of 15%. The DTR cut-off is required to ensure a higher volume of magnetite-bearing mineralisation is selected, removing the rock volumes with low magnetite content, such as the siliceous bands within the magnetite-bearing rock (BIF).

Table 14-22: Mineral Resource Estimate, Sandalwood, Clark Hill North, Clark Hill South, and Snark, Where DTR >15%

Deposit	Category	Tonnes (Mt)	Head grade (%)				Concentrate grade (%)					
			Fe	SiO_2	Al_2O_3	LOI	DTR	Fe	P	SiO_2	Al_2O_3	LOI
Sandalwood	Inferred	334	31.1	48.4	1.5	-0.6	33.1	64.7	0.03	9.5	0.06	-2.7
Snark	Inferred	69	27.8	49.8	1.6	2.4	23.4	66.2	0.03	7.5	0.13	-2.8
Clark Hill North	Inferred	130	25.8	42.6	1.7	0.14	33.2	62.4	0.04	12.1	0.16	-2.6
Clark Hill South	Inferred	15	32.3	47.0	0.6	0.02	31	63.8	0.02	9.8	0.14	0.0

Notes:

- *The Mineral Resource estimate was prepared by David Williams, B.Sc., MAIG, a CSA Global employee, and the Qualified Person for the estimate.*
- *Mineral Resources were estimated using Datamine Studio RM (Version 1.6.87).*
- *Assays were composited to regular 1 m or 5 m intervals, dependent upon the deposit.*
- *Composite assay grades were capped as required. Fe and DTR grades were not capped.*
- *Block-model grade interpolation was undertaken using ordinary kriging.*
- *Bulk density was calculated for each block in the Moonshine model using algorithms, based upon the estimated Head Fe block grade. Average bulk density of 3.3 t/m3 was applied to the other deposit models.*
- *Mineral Resources are reported from a model with parent block dimensions of 25 m x 25 m x 10 m.*
- *Tonnage and grade have been rounded to reflect the relative accuracy of the Mineral Resource estimate; therefore, columns may not total due to rounding.*
- *Resource classification is as defined by the Canadian Institute of Mining, Metallurgy and Petroleum in their document "CIM Definition Standards for Mineral Resources and Mineral Reserves" of 10 May 2014.*
- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
- *The QP and Macarthur are not aware of any current environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might materially affect these Mineral Resource estimates.*

14.4.16. Previous Mineral Resource Estimates

The Mineral Resources were previously reported in 2009 as per Table 14-23. The tenure covering the Sandalwood and Snark deposits has marginally reduced in area since that time, which required a re-reporting of the Mineral Resources within the adjusted tenure boundaries.

The Issuer is not treating the previous mineral resource estimates as current mineral resources. These previous mineral resource estimates are presented for historical information and context only. Current Mineral Resource estimates are presented in Section 14.4.15 of this report.

The QP reviewed the 2009 Clark Hill South Mineral Resource estimate, and a decision was made to update the geological models to ensure only the mineralisation domains cut by drilling were used to report the Mineral Resource. This has resulted in a decrease of 51 Mt from the 2009 Mineral Resource.

The QP is confident that additional drilling at the deposit will allow an increase to the Mineral Resource tonnages as currently reported at Clark Hill South, if Macarthur choose to pursue the development of the Clark Hill South deposit.

Table 14-23: Previous Mineral Resource estimate – not current, Sandalwood, Clark Hill North, Clark Hil, South and Snark, where DTR >15%, as reported in 2009 (Allen, 2009) and 2010 (Macarthur, 2010)

Deposit	Category	Tonnes (Mt)	Head grade (%)				Concentrate grade (%)					
			Fe	SiO_2	Al_2O_3	LOI	DTR	Fe	P	SiO_2	Al_2O_3	LOI
Sandalwood	Inferred	335	31.1	-	-	-	33.1	64.0	0.031	9.64	0.07	-2.77
Snark	Inferred	75	27.7	-	-	-	-	-	-	-	-	-
Clark Hill North	Inferred	130	25.8	42.6	1.7	0.14	33.2	62.4	0.04	12.1	0.16	-2.6
Clark Hill South	Inferred	66	30.3	-	-	-	-	-	-	-	-	-

15. Mineral Reserve Estimates

15.1. Summary

The Mineral Reserve for the Lake Giles Iron Project is estimated at 236.6 Mt at an average grade of 28.2% Fe and DTR of 31.3%, as presented in Table 15-1. The Mineral Reserve estimate was prepared by Orelogy Consulting Pty Ltd (Orelogy) after diluting the resource block model for the Moonshine and Moonshine North deposits completed by CSA Global Pty Ltd (CSA Global) previously reported to the market on 11 August 2020.

Table 15-1: Mineral Reserve Estimate – Lake Giles Iron Project, Moonshine and Moonshine North, DTR >15%

Category	Tonnes (Mt)	Head Grades (%)					Concentrate Grades (%)					
		Fe	SiO_2	Al_2O_3	P	LOI	DTR	Fe	SiO_2	Al_2O_3	P	LOI
Moonshine												
Proven	34.2	28.1	51.6	1.2	0.04	1.7	30.5	65.9	6.8	0.15	0.02	-0.6
Probable	166.4	27.2	51.9	1.4	0.05	1.4	30.7	66.6	6.2	0.11	0.02	0.0
Sub-total	**200.6**	**27.4**	**51.9**	**1.4**	**0.05**	**1.4**	**30.6**	**66.5**	**6.3**	**0.12**	**0.02**	**-0.1**
Moonshine Nth												
Proven	17.8	35.4	35.4	2.2	0.06	4.2	34.3	66.5	5.0	0.32	0.03	-0.9
Probable	18.2	30.4	44.7	1.3	0.05	2.9	35.9	63.2	9.4	0.24	0.04	-0.3
Sub-total	**36.0**	**32.9**	**40.1**	**1.7**	**0.05**	**3.5**	**35.1**	**64.8**	**7.3**	**0.28**	**0.05**	**-0.6**
Combined												
Proven	51.9	30.6	46.0	1.5	0.05	2.6	31.8	66.1	6.1	0.22	0.03	-0.7
Probable	184.7	27.6	51.2	1.4	0.05	1.5	31.2	66.2	6.6	0.13	0.02	-0.1
TOTAL	**236.6**	**28.2**	**50.1**	**1.4**	**0.05**	**1.8**	**31.3**	**66.2**	**6.5**	**0.15**	**0.02**	**-0.2**

Notes

- Canadian Institute of Mining, Metallurgy and Petroleum “CIM Definition Standards for Mineral Resources and 2. Mineral Reserves” (CIM, 2014) definitions were followed for Mineral Reserves.
- Mineral Reserves are reported using a Davis Tube Recovery (DTR) cut-off grade of 15% after applying dilution to the resource model.
- Mineral Reserves were estimated using a 62% Fe benchmark price of USD100/dmt with a 20% premium for 65% Fe and concomitant Fe concentrate grade bonus.
- Mineral Reserves account for mining dilution of 2.5% at a grade of 14% DTR and mining ore loss of 2.0% at a grade of 30% DTR.
- Mineral Reserves are reported on a Dry Tonnage Basis with an average bulk density of 3.2 t/m^3.
- The average strip ratio is 2.6:1.
- Mineral Reserves are a part of Mineral Resources.
- Proven Mineral Reserves are based on Measured Mineral Resources only and Probable Mineral Reserves are based on Indicated Mineral Resources only; and
- The sum of individual amounts may not equal due to rounding.

15.2. Resource Block Model

The Mineral Resource block model, named “ms0720md.dm”, contained attributes for interpolated head grades for Fe, SiO_2, Al_2O_3, P, S and LOI, and interpolated concentrate grades based on DTR analysis for the same suite of elements. The model framework is presented in Table 15-2.

Table 15-2: Resource block model framework

Parameter	Unit	X – Easting	Y – Northing
Origin (Min. Extent)	m	786,500	6,670,900
Maximum Extent	m	791,500	6,677,100
Range	m	5,000	6,200
Largest (parent) cell	m	25	25
Smallest sub cell	m	2.5	2.5
No. of Parent Cells	m	200	248

15.3. Pit Optimisation

15.3.1. Pit Slope Geotechnical Assessment

The geotechnical assessment for the Moonshine and Moonshine North pits was undertaken by Pells Sullivan Meynink Pty Ltd (PSM). Geotechnical drilling and associated activities targeted the approximate location of walls for the first 10 years based on preliminary designs to provide a BFS level assessment for the initial stages covering the first 10 years of operation.

The slope recommendations are reported in terms of structural domains presented in Figure 15-1. The recommend slope design parameters are presented in Table 15-3.

Table 15-3: Pit Slope Design Recommendations

	Domain	Rock Mass Unit	IRA[1] (º)	BFA[2] (º)	Bench Height (m)	Berm Width (m)
All	All	Saprolite	35	50	10	6
Footwall (northeast facing)	Domain 1. 3, 4	All Fresh	56	60	20	8
	Domain 6A		42	50	20	8
Hangingwall (southwest facing)	Domain 2, 5, 6A & 6B	All Fresh, excluding Ultramafics	56	75	20	8
		Ultramafics	46	60	20	8

1. Inter-ramp angle (IRA), measured to-toe over a stack of benches unbroken by a ramp.
2. Bench face angle (BFA).

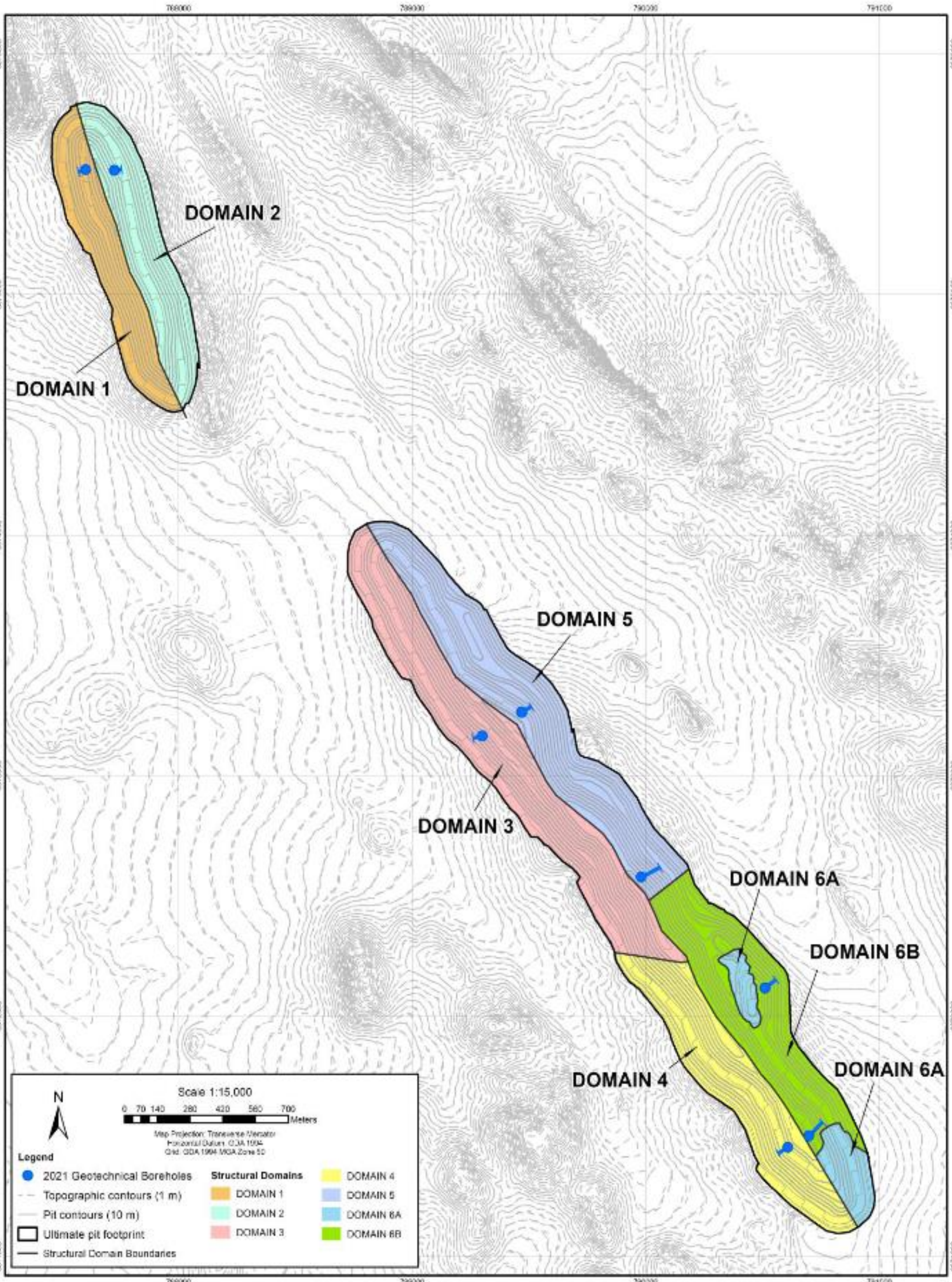


Figure 15-1: Structural domains (designs shown are preliminary)

15.4. Mining Dilution and Ore Loss

The Lake Giles orebodies are sub-vertical in nature and vary from 10 m in width to more 100 m in width and usually separated by wide waste zones. Based on the geometry of the ore body and the proposed open cut mining method a dilution assessment was made using edge dilution as the most appropriate technique. In general, edge dilution occurs along the ore/waste boundary only. If dilution was expected to occur within the orebody as well as the boundaries, another approach to cater for the internal dilution would have been considered.

The resource model was re-blocked to 6.25 m (E) x 6.25 m (N) x 5 m (RL) block size based on ¼ of the parent cell size laterally and a 5 m bench height. The ore tonnes and metal contents for the blocks were preserved as ore parcels within the regular blocks and an ore percentage assigned.

The re-blocked model was interrogated across the strike of the orebody from both directions to identify edge blocks with an ore percent of less than 100% and an adjacent block that is 100% waste, and isolated blocks that have 100% waste blocks on both sides.

Diluent material with default grades was added to the ore percent once to an edge block and twice for isolated blocks, up to a maximum ore of 100% with a consequent reduction in waste percent. The quantity of material transferred in this process was defined using a skin width of 2.0 m (32%) to represent the selectivity that should be achievable by a 300-400t excavator.

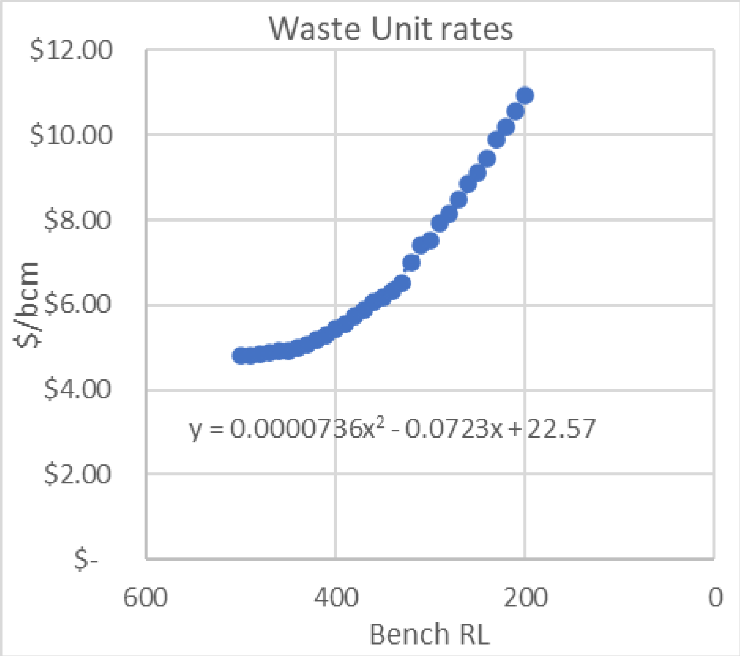
The overall dilution modelling resulted in 2.0% dilution at a grade of 14% DTR and 2.5% ore loss at grade of 30% DTR.

15.5. Pit Optimisation Parameters

The first stage of the conversion of a mineral resource into a mineable open pit reserve is the open pit optimisation process. It is at this stage that all the latest physical, technical, and economic parameters are applied to the orebody to determine the "ideal" open pit excavation geometry.

The open pit optimisation process undertaken for this study has the following key assumptions on the constraints and parameters utilised:

- Only material classified as Measured and Indicated in the Mineral Resource model were considered as potential ore during the optimisation process.
- Mining dilution and mining recovery parameters were modelled in the block model.
- Product prices and royalties were provided by Macarthur.
- Front-end processing costs supplied by Engenium were applied to ore tonnes.
- Grade control and ore re-handling were sourced from Contractors.
- Annual fixed mining overheads for the Owners team were applied as a unit rate to the ore tonnes processed.
- Logistics costs were supplied by Macarthur and were applied as selling costs per tonne of concentrate.
- Overall pit slope angles were based on geotechnical recommendations for specific materials by domain.
- Waste Mining costs were assigned by bench in the block model using regression for the unit rates provided by the selected Mining Contractor. The extra cost of mining ore, referred to as the Ore Mining Premium (OMP), were also assigned in the model as a grade item and added to the processing costs during optimisation. The regressions are shown graphically in Figure 15-2. The optimisation input parameters are detailed in Table 15-4.



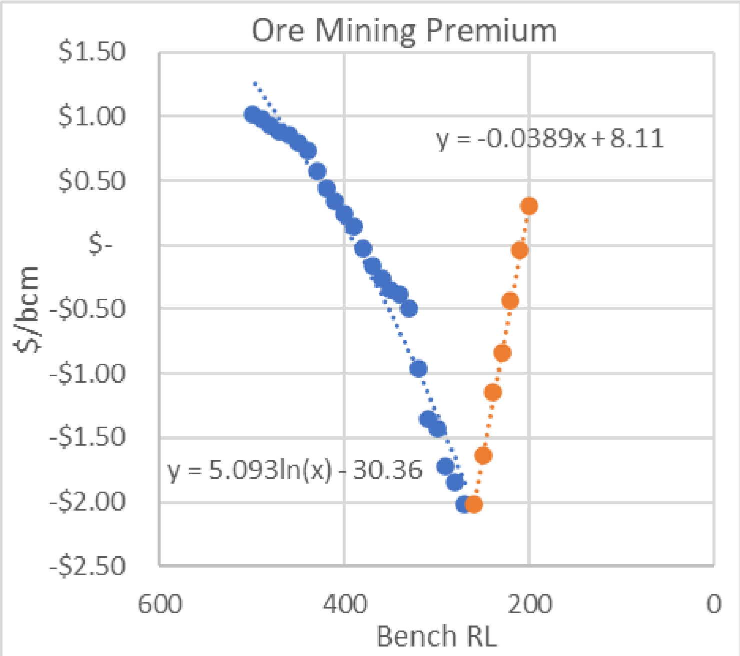


Figure 15-2: Regressions for mining unit rates per bcm mined including allowance for fuel

Table 15-4: Pit Optimisation Input Parameters

Optimisation Parameter	Unit	Value
Financial Parameters		
Iron Ore Price for 66% Product	USD/t concentrate	125
Shipping and Insurance	USD/t concentrate	13.20
Price FOB	USD/t	111.80
Exchange rate	USD: AUD	0.73
Government Royalty	%	5.0
Net Price	$/t (AUD)	145.49
Discount Rate	%	8.0
Selling Parameters		
Concentrate Production	Mt/a (wet)	3.3
Road transport	$/wt concentrate	9.09
Rail transport	$/wt concentrate	15.64
Port Charges	$/wt concentrate	7.58
Moisture content	%	9.0
Total selling cost	$/dt concentrate	29.64
Processing Parameters		
Design throughput capacity	Mt/a (dry)	9.68
Owner Mining Overhead	$d/t ore	1.26
Grade control	$/dt ore	0.13
Ore mining premium > 265 mRL: < 265 mRL:	$/dt ore $/dt ore	OMP = (5.093 x LN(Bench RL) – 30.32)/SG OMP = (-0.039 x (Bench RL) + 8.11)/SG
Ore Blasting premium	$/dt ore	0.33
Ore Feed Rehandle (55%)	$/dt ore	0.80
Reclaim from Stockpile (20% of ore mined)	$/dt ore	0.49
Dry reject rehandle (149 t/h)	$d/t ore	0.31
Crushing	$d/t ore	0.84
Processing	$/dt ore	10.21
Tailings & Filtration	$d/t ore	0.97
Site general and administration	$/dt ore	1.13
Sustaining Capital	$/dt ore	0.30
TOTAL Processing Cost (excl. OMP)	$/dt ore	16.44
Mining parameters		
Mining rate	Mt/a	45

Slope Parameters (OSA[1]) Oxide Moonshine Nth HW[2] Oxide others Fresh FW[2] (D1, D3, D4) Fresh HW[2] (D6A) Fresh HW[2] (D2, D5, D6B)	 Degrees Degrees Degrees Degrees Degrees	 27 33 41 37 41
Drill and Blast Oxide waste Fresh Waste	 $/dt $/dt	 0.50 0.81
Load and Haul waste	$/dt	MCAF = (0.0000736 x (Bench RL) ^{2} – 0.0723 x (Bench RL) + 22.58)/SG
Load and Haul waste	$/dt	MCAF = (0.0000736 x (Bench RL) ^{2} – 0.0723 x (Bench RL) + 22.58)/SG

Notes:
1. Overall Slope Angle (OSA), measured toe-crest over a stack of benches including ramp allowance; and
2. Hangingwall (HW), Footwall (FW).

15.6. Open Pit Optimisation Results

The optimisation produces a sequence of nested 3D pit shells based on the Revenue Factor (RF) applied in Whittle™. The RF is a factor applied to the commodity price within Whittle™ to reflect a range of higher and lower prices compared to the base commodity price supplied with a RF=1.0. Lower factored prices will produce smaller shells, the higher price factor, the larger the shell.

The Base Case optimisation has been undertaken using Measured and Indicated mineral resources after the application of dilution and ore loss.

Table 15-5: Whittle Pit Optimisation Results for M&I material

Shell	RF	Total Ore						Conc	Waste	Total
		Mt	Fe (%)	DTR (%)	Fe (%) Conc	S (%) Conc	SiO_2 (%) Conc	Mt	Mt	Mt
1	0.69	6.0	30.1	37.3	62.0	1.2	10.8	2.2	18.5	24.5
2	0.7	6.1	30.1	37.3	61.9	1.2	10.8	2.3	18.8	24.9
3	0.71	17.0	29.6	34.8	64.9	1.1	7.7	5.9	44.2	61.2
4	0.72	28.1	29.1	33.3	65.6	1.2	7.0	9.4	63.2	91.2
5	0.73	96.0	28.4	32.5	66.1	1.3	6.6	31.2	215.3	311.3
6	0.74	126.3	29.2	32.6	66.1	1.3	6.5	41.2	301.1	427.4
7	0.75	130.8	29.2	32.5	66.1	1.3	6.5	42.6	310.3	441.1
8	0.76	146.4	28.9	32.3	66.1	1.3	6.5	47.2	340.9	487.3
9	0.77	154.1	28.8	32.2	66.1	1.3	6.5	49.6	357.2	511.3
10	0.78	162.0	28.7	32.1	66.1	1.3	6.6	51.9	374.6	536.6
11	0.79	176.3	28.6	32.0	66.1	1.3	6.7	56.4	414.9	591.2
12	0.8	185.4	28.5	31.9	66.1	1.3	6.7	59.1	439.0	624.3
13	0.81	210.1	28.3	31.5	66.1	1.4	6.5	66.3	504.3	714.4

14	0.82	216.3	28.3	31.5	66.1	1.4	6.5	68.2	524.8	741.2
15	0.83	221.9	28.3	31.5	66.1	1.4	6.5	69.9	541.3	763.3
16	0.84	222.6	28.3	31.5	66.1	1.4	6.5	70.1	543.8	766.3
17	0.85	225.2	28.3	31.5	66.1	1.4	6.5	70.8	551.0	776.2
18	0.86	228.9	28.26	31.4	66.1	1.4	6.5	72.0	564.7	793.7
19	0.87	232.6	28.3	31.4	66.1	1.4	6.5	73.1	580.2	812.8
20	0.88	234.3	28.3	31.4	66.1	1.4	6.5	73.6	586.0	820.3
21	0.89	235.4	28.3	31.4	66.1	1.4	6.5	73.9	590.4	825.8
22	0.9	237.6	28.3	31.4	66.1	1.3	6.5	74.6	598.9	836.5
23	0.91	239.1	28.2	31.4	66.1	1.3	6.5	75.0	604.8	843.8
24	0.92	241.7	28.2	31.4	66.1	1.3	6.5	75.8	615.8	857.4
25	0.93	243.5	28.2	31.3	66.1	1.3	6.5	76.3	622.7	866.2
26	0.94	244.8	28.2	31.3	66.1	1.3	6.5	76.7	630.0	874.9
27	0.95	245.8	28.2	31.3	66.1	1.3	6.5	77.0	634.0	879.8
28	0.96	247.1	28.2	31.3	66.1	1.3	6.5	77.4	639.3	886.4
29	0.97	248.4	28.2	31.3	66.1	1.3	6.5	77.7	645.3	893.7
30	0.98	250.6	28.2	31.3	66.1	1.3	6.5	78.4	656.6	907.2
31	0.99	252.0	28.2	31.3	66.1	1.3	6.5	78.8	663.9	916.0
32	1	252.5	28.2	31.3	66.1	1.3	6.5	78.9	666.5	919.0
33	1.01	253.2	28.2	31.3	66.1	1.3	6.5	79.1	670.0	923.2
34	1.02	253.5	28.2	31.3	66.1	1.3	6.5	79.2	672.1	925.5
35	1.03	253.8	28.2	31.2	66.1	1.3	6.5	79.3	673.6	927.4
36	1.04	254.7	28.2	31.2	66.1	1.3	6.5	79.6	677.6	932.4
37	1.05	255.0	28.2	31.2	66.1	1.3	6.5	79.6	679.0	934.0
38	1.06	255.8	28.1	31.2	66.1	1.3	6.5	79.9	683.2	939.0

Whittle applies time-based costs to the nested shells using a discount factor and generates two specific cases:

- The worst-case scenario which assumes that, for any given shell, extraction is undertaken bench by bench sequentially from the top to bottom of the pit. This results in a significant amount of any overlying overburden being removed prior to presentation of ore; and
- The best-case scenario assumes that, for any given shell, extraction is undertaken sequentially from the smallest shell through all the intervening shells out to the shell selected. This approach generates the least amount of waste stripping but has an impractically high number of small incremental pushbacks.

These two DCF cases provide the extremities of the theoretical value that can be generated from a deposit. The shell that represents what will happen in reality will in fact lie somewhere between these two endpoints.

A technique for final shell selection as described by in the paper “Skin analysis in the selection of Final Pit limits” by Hansen et. al 2001 was used to select optimal shell with the worst-case shell 13 used as the penultimate shell. This identified shell 20 with a revenue factor of 0.88 as the preferred shell for design. This shell provided sufficient ore for a 25-year mine life and captured 99.5% of Best-case value with 93% of the ore and 89% of the waste.

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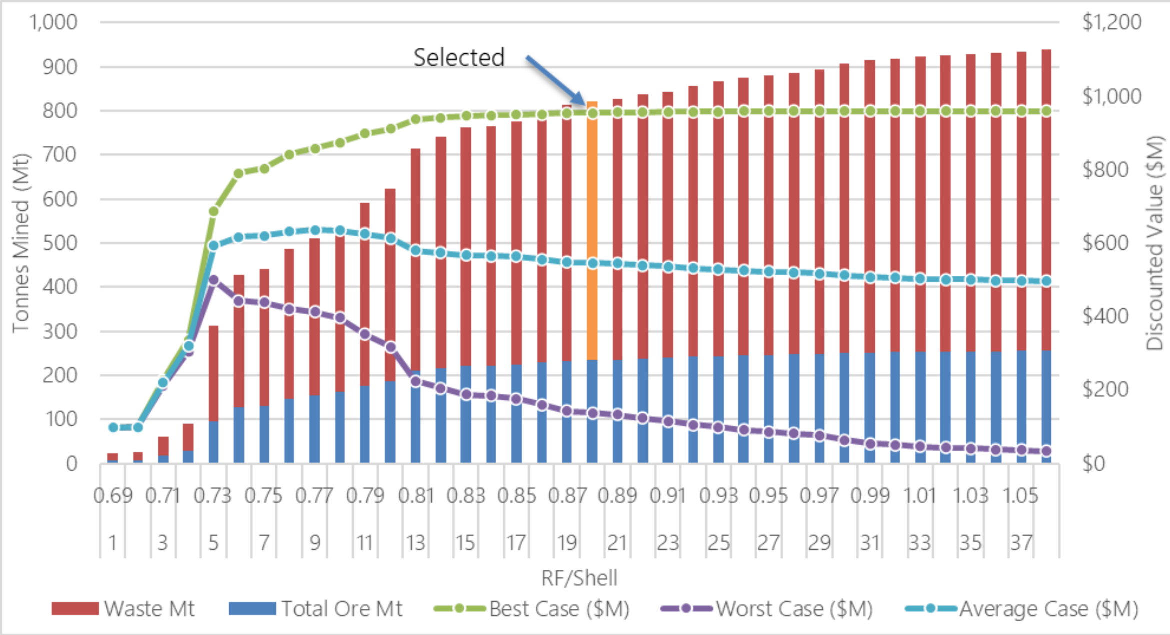


Figure 15-3: Pit by Pit chart for the pit optimisation results showing the selected case for design

15.7. Mine Design

15.7.1. Slope Design Criteria

The final pits and interim stages were designed in single 10 m bench heights in oxide and double 10 m benches in fresh rock with batter and berm configurations as recommended by PSM and presented Figure 15-3.

15.7.2. Ramp Design Criteria

The ramps and haul roads were designed to accommodate a 180-tonne class haul truck (Cat 789D) with an overall width of 7.65 m. Dual-lane ramps were designed with a pavement width of 3.0 time the overall truck width plus an allowance for a windrow and drain, totalling 29 m. Single lanes were designed for a pavement width of 1.5 time the overall truck width plus allowance for safety bund and drain, totalling 18 m. All ramp gradients have been set to 10% (one vertical metre for every 10 horizontal metres), which is the industry accepted ramp gradient for uphill loaded hauls.

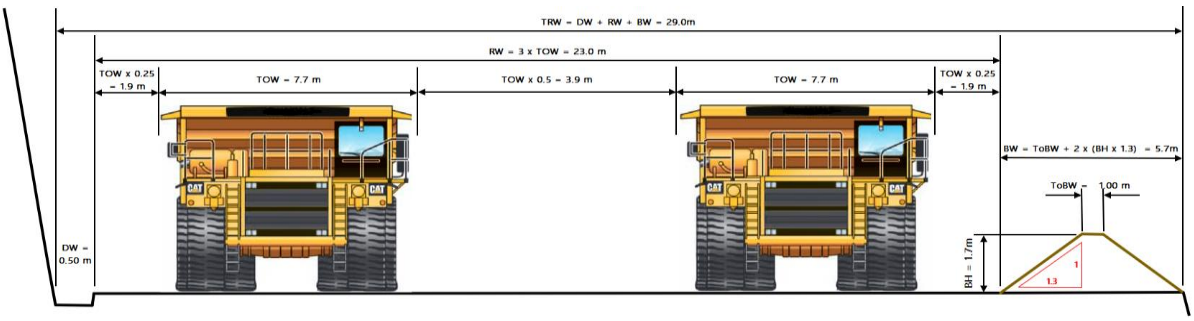


Figure 15-4: Dual lane with single windrow and drain

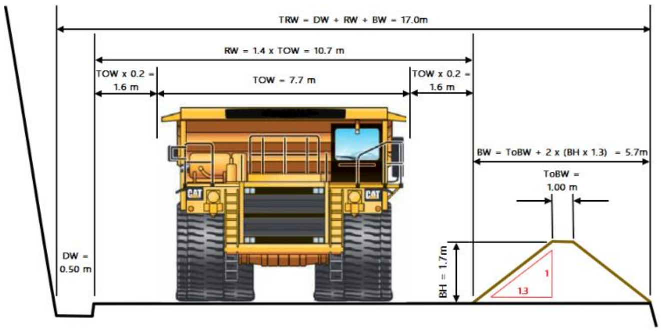


Figure 15-5: Single lane with single windrow and drain

15.7.3. Open Pit Mine Design Results

The final pit design comprises two separate pits with a total of seven internal stages. An overview of the final pit showing internal stages is presented in Figure 15-6.

Moonshine North pit is approximately 1,450 m long, 500 m wide and 225 m deep and Moonshine is approximately 3.7 km long, 700 m wide and 250 m deep. Each stage has a separate ramp system that exits on the west side to provide short hauls to waste dumps and ROM pad.

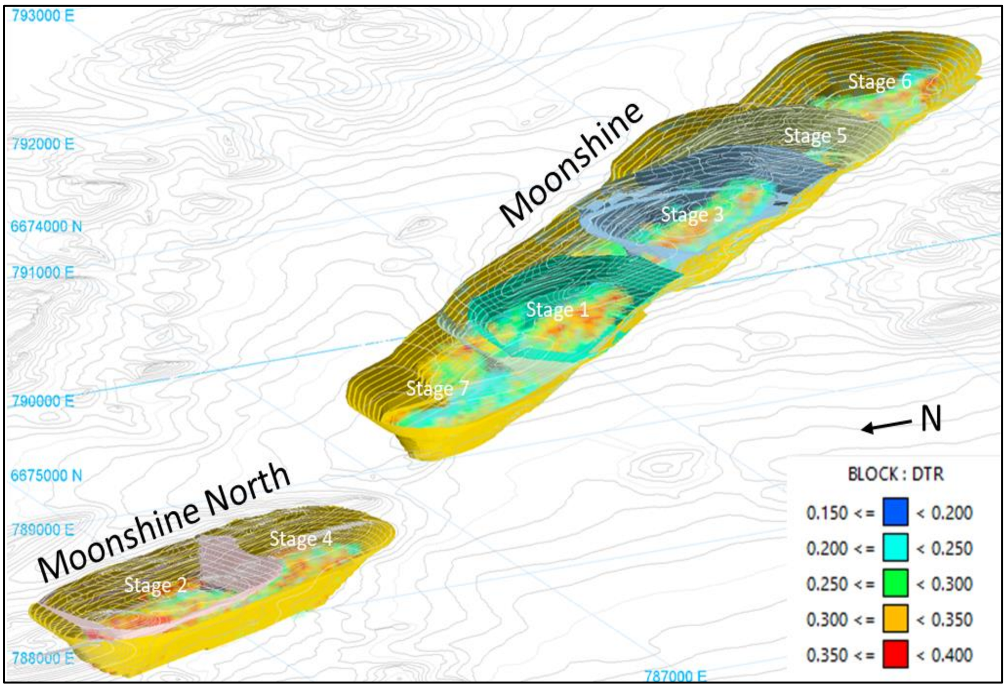


Figure 15-6: Moonshine and Moonshine North pits showing stages and mineralisation coloured by DTR

The design process provides a practical solution to the Whittle shells by adding an arrangement of benches, berms, roads and ramp systems. This process has added 6.6% additional waste and captured 0.9 % additional ore than defined by the Whittle shell. Straightening of walls and the addition of ramps has contributed to the additional waste captured in the design process.

A comparison between the Whittle shell and the final designs is shown in Figure 15-7. Cross sections through the Moonshine North and Moonshine are shown in Figure 15-8 and Figure 15-9 respectively.

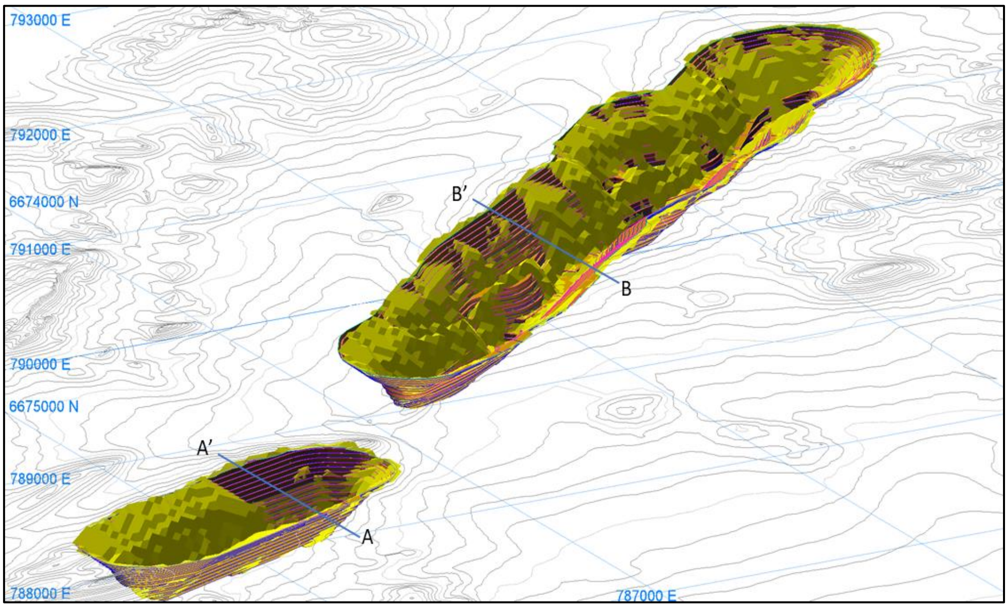


Figure 15-7: Comparison of pit design with Whittle shell

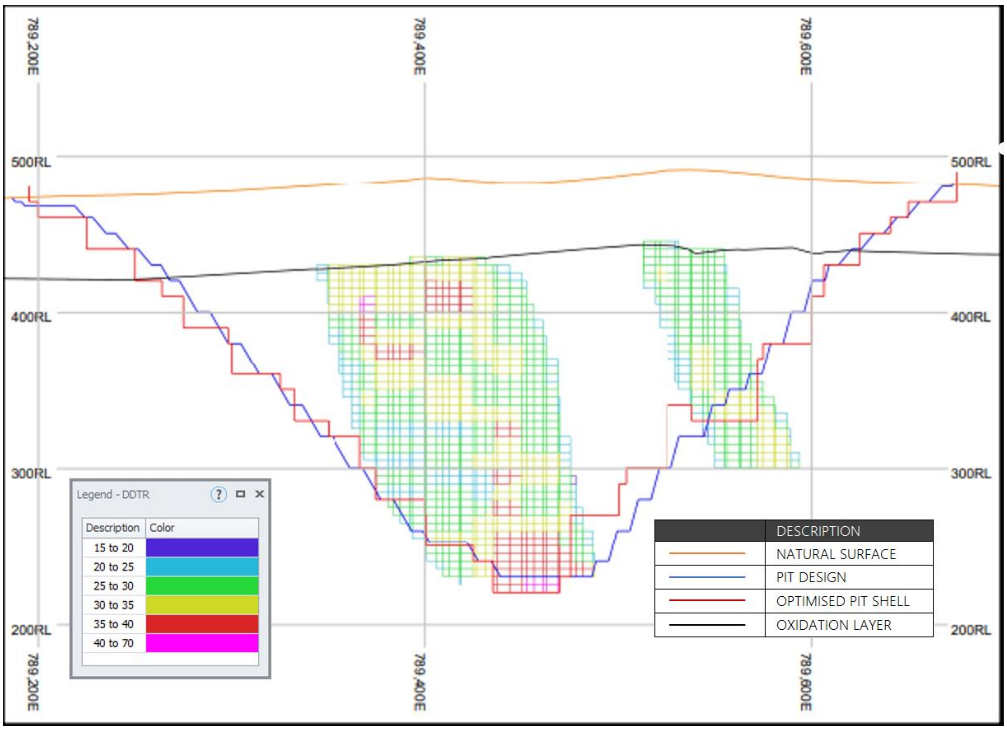


Figure 15-8: Moonshine North Pit designs in versus RF 0.88 Pit Shell (Section AA')

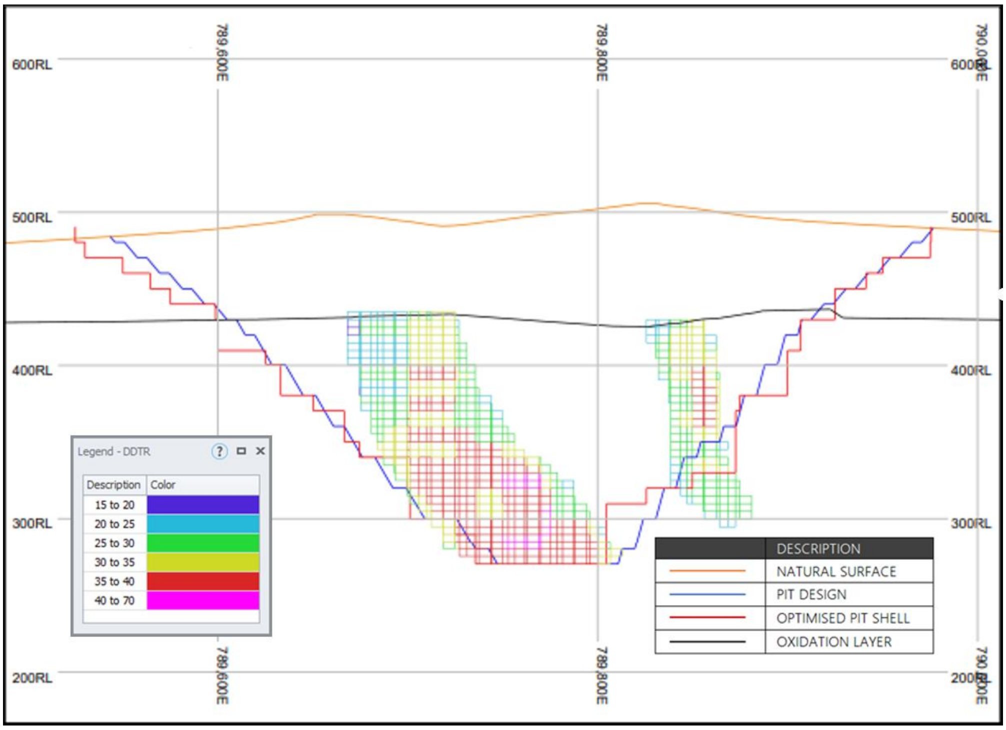


Figure 15-9: Moonshine Pit designs in versus RF 0.88 Pit Shell (Section BB')

15.8. Mineral Reserve Statement

Orelogy have developed a Mineral Reserve Estimate for the Lake Giles Iron Project (LGIP) open pits as of 21 February 2022 in accordance with Australian JORC 2012 and Canadian NI43-101 Technical Reporting standards.

Mineral Resources were converted to Mineral Reserves in line with the material classifications which reflect the level of confidence within the resource estimate. The Mineral Reserve reflects that portion of the Mineral Resource which can be economically extracted by open pit mining methods. The Mineral Reserve considers the modifying factors and other parameters described above, including but not limited to the mining, metallurgical, social, environmental, statutory, and financial aspects of the project.

The Proven and Probable Mineral Reserves total 236.6 Mt at an average grade of 28.2% Fe using a cut-off grade of 15% DTR after applying dilution and ore losses. The total tonnage to be mined is estimated at 853.4 Mt at a strip ratio of 2.6:1. The Mineral Reserves are summarised in Table 15-6.

Table 15-6: Mineral Reserve Estimate – Lake Giles Iron Project, Moonshine and Moonshine North, DTR >15%

Category	Tonnes (Mt)	Head Grades (%)					Concentrate Grades (%)					
		Fe	SiO_2	Al_2O_3	P	LOI	DTR	Fe	SiO_2	Al_2O_3	P	LOI
Moonshine												
Proven	34.2	28.1	51.6	1.2	0.04	1.7	30.5	65.9	6.8	0.15	0.02	-0.6
Probable	166.4	27.2	51.9	1.4	0.05	1.4	30.7	66.6	6.2	0.11	0.02	0.0
Sub-total	**200.6**	**27.4**	**51.9**	**1.4**	**0.04**	**1.4**	**30.6**	**66.5**	**6.3**	**0.12**	**0.02**	**-0.1**
Moonshine Nth												
Proven	17.8	35.4	35.4	2.2	0.06	4.2	34.3	66.5	5.0	0.32	0.03	-0.9
Probable	18.2	30.4	44.7	1.3	0.05	2.9	35.9	63.2	9.4	0.24	0.04	-0.3
Sub-total	**36.0**	**32.9**	**40.1**	**1.7**	**0.05**	**3.5**	**35.1**	**64.8**	**7.3**	**0.28**	**0.05**	**-0.6**
Combined												
Proven	51.9	30.6	46.0	1.5	0.05	2.6	31.8	66.1	6.1	0.22	0.03	-0.7
Probable	184.7	27.6	51.2	1.4	0.05	1.5	31.2	66.2	6.6	0.13	0.02	-0.1
TOTAL	**236.6**	**28.2**	**50.1**	**1.4**	**0.05**	**1.8**	**31.3**	**66.2**	**6.5**	**0.15**	**0.02**	**-0.2**

Notes

- Canadian Institute of Mining, Metallurgy and Petroleum "CIM Definition Standards for Mineral Resources and 2. Mineral Reserves" (CIM, 2014) definitions were followed for Mineral Reserves.
- Mineral Reserves are reported using a Davis Tube Mass Recovery (DTR MR) cut-off grade of 15% after applying dilution to the resource model.
- Mineral Reserves were estimated using a 62% Fe benchmark price of USD100/dmt with a 20% premium for 65% Fe and concomitant Fe concentrate grade bonus.
- Mineral Reserves account for mining dilution of 2.5% at 14% DTR and mining ore loss of 2.0% at 30% DTR.
- Mineral Reserves are reported on a Dry Tonnage Basis with an average bulk density of 3.2 t/m3.
- The average strip ratio is 2.6:1.
- Mineral Reserves are a part of Mineral Resources.
- Proven Mineral Reserves are based on Measured Mineral Resources only and Probable Mineral Reserves are based on Indicated Mineral Resources only; and
- The sum of individual amounts may not equal due to rounding.

16. Mining Methods

16.1. Introduction

Orelogy Consulting Pty Ltd. (Orelogy) was engaged to complete a mining study with the objectives of developing an integrated Life of Mine (LOM) schedule for the mining and processing operation for the Project and defining a Mineral Reserve estimate.

The study is based on mining the magnetite ore and processing on site at a rate of approximately 10.0 Mtpa throughput to produce 3.0 Mtpa (dry) beneficiated magnetite concentrate. The concentrate product is planned to be hauled approximately 90 km south to a railhead from where it will be railed to the Esperance port and loaded onto ships for export.

The mining study relates only to the Moonshine and Moonshine North magnetite deposits of the Lake Giles Iron Project

16.2. Geotechnical Assessment

16.2.1. Site Investigation

The geotechnical assessment for the Moonshine and Moonshine North pits was undertaken by Pells Sullivan Meynink Pty Ltd (PSM) and comprised an assessment of diamond drill holes from both mine pits

The 2021 diamond drilling program comprised nine diamond cored boreholes using either HQ3 or PQ3 for a total of 1,598.4 m drilled near or through the proposed final walls at an inclination between 65º and 90º as summarised in Table 16-1. The location of the boreholes is shown in Figure 16-1. Designs shown are preliminary. Logging of geotechnical characteristics comprised:

- Rock Quality Designation (RQD) and field estimated strength
- Weathering and lithology; and
- Defects – type, inclination and orientation, shape and roughness, infill type and thickness.
-

Table 16-1: Summary of Geotechnical Boreholes

Borehole ID	Easting (mE)	Northing (mN)	Elevation (mRL)	Inclination (º)	Azimuth (º)	Depth (m)
LGDD_076	789469	6673262	485.5	-77	060	230.1
LGDD_077	790512	6672115	497.7	-70	052	180.1
LGDD_078	790610	6671453	505.0	-74	240	186.1
LGDD_079	787726	6675515	486.9	-79	081	175.7
LGDD_080	790700	6671500	505.0	-65	045	200.0
LGDD_081	789302	6673162	477.3	-77	260	195.1
LGDD_082	789983	6672576	498.7	-65	062	219.8
LGDD_083	787602	6675518	481.8	-79	255	174.1
LGDD_084	789472	6673261	485.6	-90	0	37.4

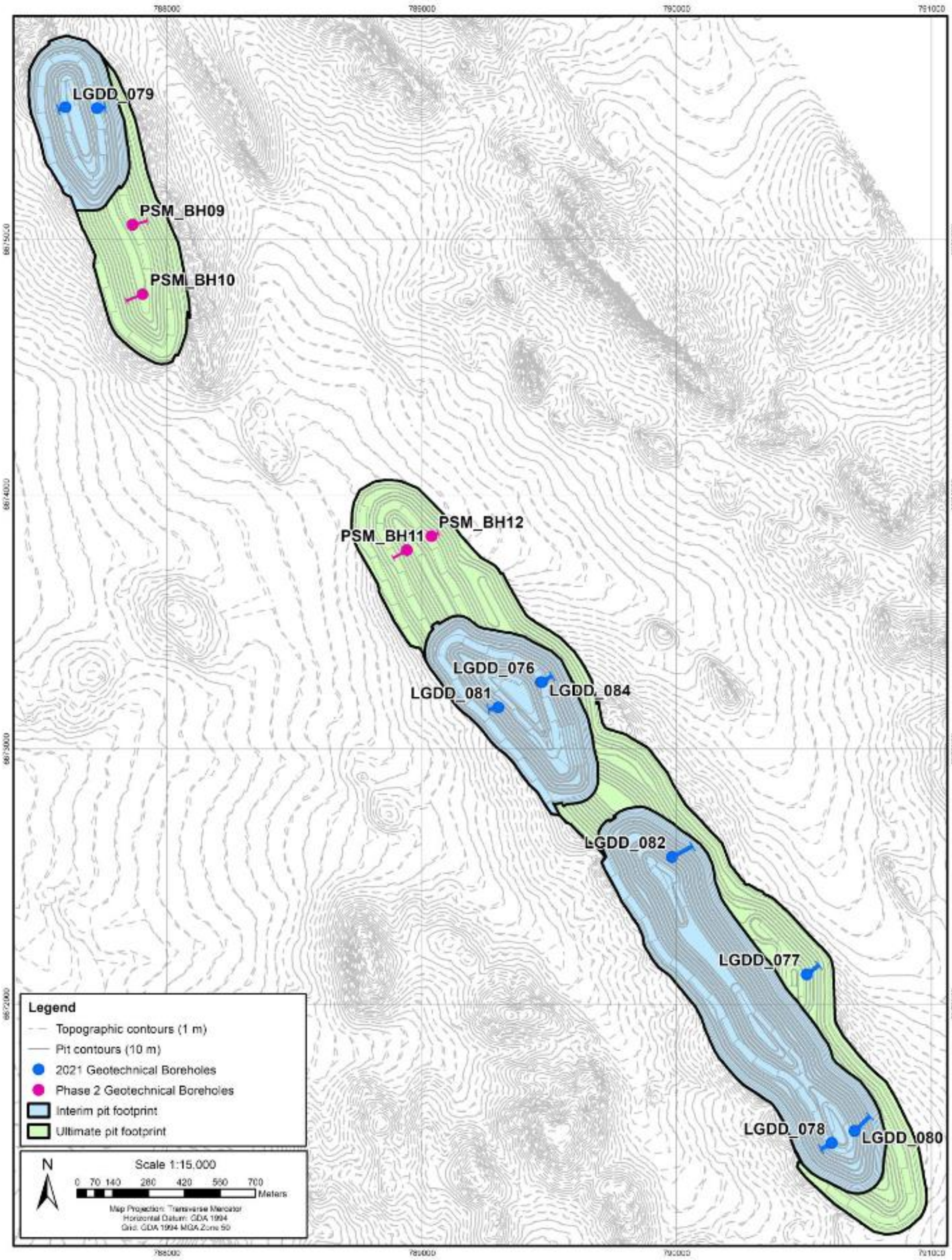


Figure 16-1: Geotechnical diamond borehole locations (PSM, 2022)

Diamond core was oriented to enable logging of geological structures using the Boart Longyear "TruCore" system and used to produce Stereo plots. Stereo plots of oriented core measurements from Macarthur historical diamond holes were also produced.

A total of 203 samples were collected for testing purposes and transported to Trilab laboratories in Perth. Testing comprised:

- 9 x Unconfined compressive strength test (UCS)

- 5 x triaxial test (TXL) – consolidated undrained testing of soil of low rock strength with pore pressure measurements; and
- 6 x Direct shear tests of defect (DS).

Optical Televiewer (OTC) and Acoustic Televiewer (ATV) borehole imaging was carried out on six of the nine boreholes. The ATV data was used to collect structure orientation and defect aperture. Cavities were observed in the geotechnical logging and confirmed in the OTV imaging. These varied in size up to 1.1 m.

16.2.2. Analysis and Modelling

PSM developed a geotechnical rock mass model comprised of eleven rock mass units (RMUs) of similar rock mass quality and grouping areas including weathering, strength, alteration, degree of fracturing and defect character. The seven primary RMUs effecting the wall stability were assessed and classified as summarised in Table 16-2.

Table 16-2: Summary of Rock Mass properties

Domain	Cohesion (kPa)	Friction Angle (°)	Hoek-Brown Strength Parameters				Tensile Strength (Mpa)	Elastic Modulus (MPa)
			UCS (Mpa)[3]	GSI[1]	m_i[2]	D		
Saprolite	25	35	Not applicable				0.0149	100
BIF Weathered	220	50	40	45	19	0.7	0.0149	900
BIF Fresh	-	-	100	70	19	0.7	0.3539	7400
Basalt Weathered	80	25	1.5	45	25	0.7	0.0004	900
Basalt Fresh	-	-	100	70	25	0.7	0.2690	7400
Ultramafic Weathered	105	35	5	40	25	0.7	0.0009	500
Ultramafic Fresh	-	-	80	65	25	0.7	0.1372	4900

1. Geological Strength Index (Hoek and Brown, 1997).
2. Empirical constant related to curvature of triaxial testing results.
3. Mohr-Coulomb parameters based on maximum normal stress range of 500kPa.

A structural model was developed containing six geotechnical slope domains. These were used in the slope stability analysis comprised of failure mechanisms, kinematic analyses, limit equilibrium stability analysis (LEA) and finite element analyses (FEM). The critical failure mechanisms are illustrated in Figure 16-2.

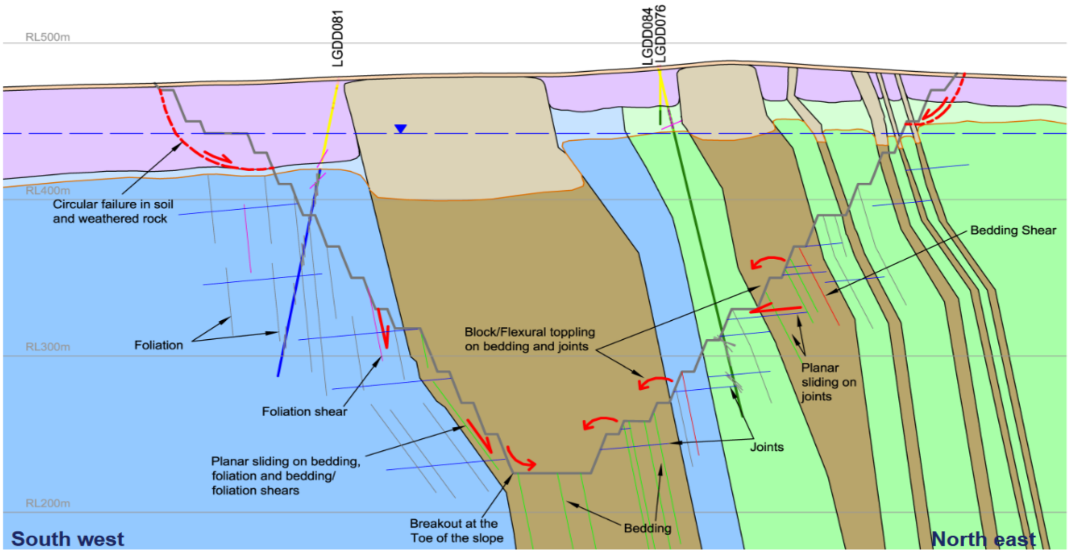


Figure 16-2: Schematic cross section showing critical failure mechanisms

Numerical modelling was undertaken in FEM software RS2 to assess the overall stability and deformation mechanisms of the footwall and hanging wall. In summary this found:

- The footwall converged to a strength reduction factor (SRF) of 1.2, indicating the rock mass shear strength is adequate to support the modelled slope angles which was consistent with the LEA.
- The hanging wall model indicated susceptibility to toppling mechanisms attributable to sensitivity to defect length and spacing assumptions in the Basalt for defect spacing of 20 m or less. When modelled with no defect spacings the model converged to SRF of 1.3 which was consistent with the LEA. PSM noted that similar rock masses are unlikely to form regular spaced defect with persistence of 50 m and instead are more likely to form isolated foliated zones with less persistence. The toppling failure of the Basalt is considered a design risk and recommend that Basalt rock is mapped during mining to assess spacing and persistence.

16.2.3. Geotechnical slope design recommendations

The slope recommendations are reported in terms of structural domains. The recommend slope design parameters are presented in Table 16-3.

Table 16-3: Pit Slope Design Recommendations

	Domain	Rock Mass Unit	IRA1 (°)	BFA2 (°)	Bench Height (m)	Berm Width (m)
All	All	Saprolite	35	50	10	6
Footwall (northeast facing)	Domain 1. 3, 4	All Fresh	56	60	20	8
	Domain 6A		42	50	20	8
Hanging wall (southwest facing)	Domain 2, 5, 6A & 6B	All Fresh, excluding Ultramafics	56	75	20	8
		Ultramafics	46	60	20	8

1. Inter-ramp angle (IRA), measured to-toe over a stack of benches unbroken by a ramp.
2. Bench face angle (BFA).

16.3. Mine Designs

16.3.1. Open Pit Phases

The staging logic for the pit design was guided by selected Whittle™ pit shells constrained by operability, safety, minimum mining width for cutbacks, staged ramp logic, bench turnover limits, equipment productivities and ore continuity logic.

The Moonshine pit is the larger of the two pits and contains 200.6 Mt of magnetite ore at an overall strip ratio of 2.4:1. Due to the size of the pit a total of five internal stages were developed. Stage 1 is the highest value area indicated by shell 3. Stage 4 is based on the central area of shell 6. Stage 5 is based on shell 6 and encompasses both stage 1 and 4 extending out to the final shell to the north-east side of the pit for mining width. Stage 6 is the final cutback extension to the south-east. Stage 7 is the lowest value cutback and extends the final pit to the north-west.

The ore in Moonshine North pit is slightly higher in Fe grade and DTR than Moonshine pit, has a higher strip ratio and pockets of material with elevated silica reporting through to the concentrate. This pit was developed in two stages to reduce the pre-strip required to expose orebody compared to mining the pit as a single stage. Both stages are mined in the first 10 years and blended with material from the Moonshine pit to manage silica levels in the concentrate.

The final pits contain a total of 236.6 Mt at an average grade of 28.2% Fe and 31.3% DTR reported above a cut-off grade of 15% DTR. The total tonnage to be mined is estimated at 861.5 Mt at a strip ratio of 2.6:1. The Moonshine pits contains 85% of the magnetite ore with a lower strip ratio at 2.4:1 compared to the smaller Moonshine North pit which has a strip ratio of 3.8:1. Table 16-4 outlines the inventory by stage on a dry tonnes basis.

Table 16-4: Moonshine and Moonshine North Pit Inventories reported by Stage

Stage	Ore	Grades %							Waste	Total	S/R
	Mt	Fe	SiO_2	Al_2O_3	P	S	LOI	DTR	Mt	Mt	W:O
Moonshine											
1	22.4	28.3	50.5	1.5	0.05	1.2	1.2	31.2	53.8	76.2	2.4
4	22.2	27.8	51.6	1.2	0.05	0.9	1.4	31.2	65.3	87.5	2.9
5	69.9	27.3	51.9	1.3	0.05	1.0	1.4	30.7	154.0	223.8	2.2
6	55.9	27.4	52.1	1.3	0.05	0.9	1.4	31.1	135.3	191.3	2.4
7	30.2	26.7	52.7	1.8	0.04	1.1	1.7	28.5	79.1	109.2	2.6
Total	**200.6**	**27.4**	**51.9**	**1.4**	**0.05**	**1.0**	**1.4**	**30.6**	**487.5**	**688.0**	2.4
Moonshine North											
2	6.4	31.8	43.8	1.3	0.05	1.3	3.0	35.0	52.1	58.5	8.2
3	29.6	33.1	39.3	1.8	0.06	1.5	3.7	35.1	85.4	115.0	2.9
Total	**36.0**	**32.9**	**40.1**	**1.7**	**0.05**	**1.4**	**3.5**	**35.1**	**137.5**	**173.5**	3.8
TOTAL	**236.6**	**28.2**	**50.1**	**1.4**	**0.05**	**1.1**	**1.8**	**31.3**	**624.9**	**861.5**	2.6

Figure 16-3: Internal Stage Designs for Moonshine Pit

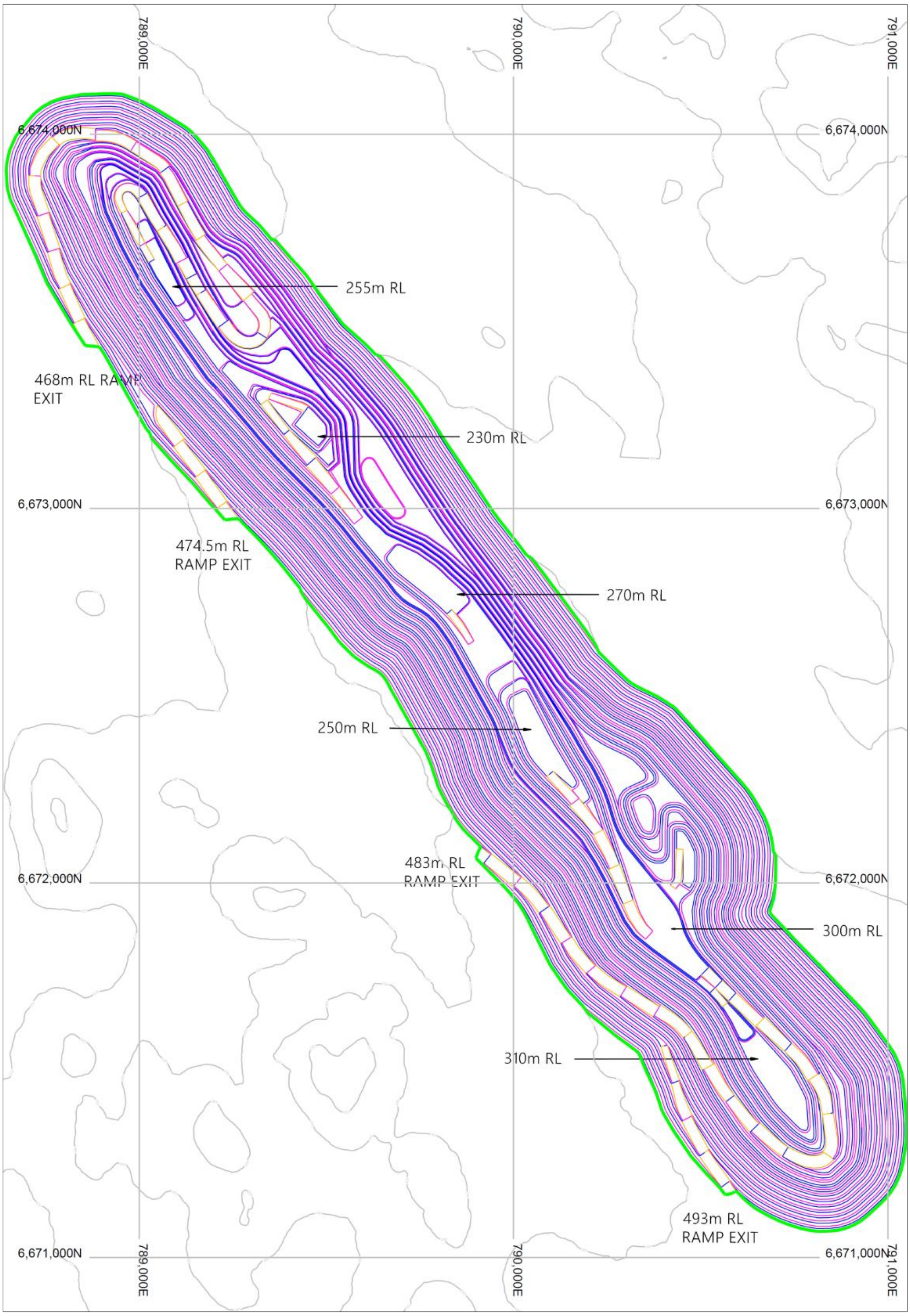


Figure 16-4: Final Design for Moonshine pit

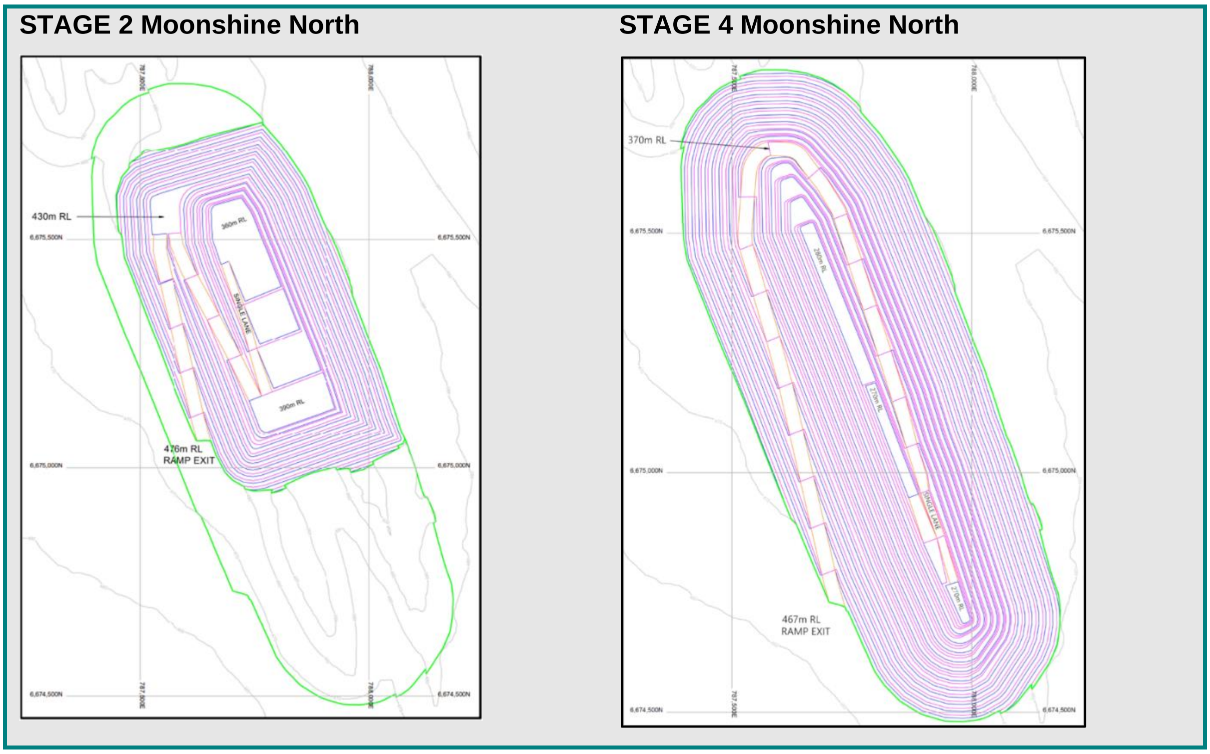


Figure 16-5: Internal and Final Designs for Moonshine North pit

16.3.2. Overburden and Waste Rock Storage

Further drilling has the potential to expand the pits both at depth and along strike, therefore all waste rock not used for infrastructure will be stored in three external waste rock dumps (WRDs) as shown in Figure 16-7. A total storage capacity of approximately 652 Mt of waste material is required. This is comprised of:

- 295 Mt of oxide overburden from pre-stripping from each stage to expose the ore within the Fresh BIF rock units.
- 328 Mt of fresh waste rock will be mined over the life of the operation.
- 29 Mt of reject waste from the dry LIMS circuit based on 12.2% of the ore feed being rejected at a rate of 149 t/h. Specific gravity of this material is 3.5 t/m3 as stated in the PDC.

Waste dump capacities are shown in Table 16-5 and are based on a swell factor of 30%. The destination quantities shown were derived for design purposes to ensure sufficient overall capacity was available on the waste dumps with approximately 4% contingency. The final waste destination was determined using haulage simulation.

Table 16-5: Waste Storage Capacities

WRD Destination	WRD Capacity m^3 (000s)	Source (Stage#)	Mined Waste bcm (000s)	Loose Waste lcm (000s)	Total Waste Mined lcm (000s)	Spare Capacity	Spare Capacity
ROM	2,456	1	1,889	2,246	2,456	-	-
Roads & Stockpile pads	58	1	45	58	58	-	-
NWWD	103,043	1 (~80%)	14,162	18,410	97,459	6%	6%
		2	16,962	22,051			
		3	27,749	36,074			
		7 (~50%)	16,095	20,925		3%	
SEWD	31,333	1 (~10%)	1,788	2,325	30,281		3%
		4	21,504	27,956		2%	
SWWD	155,055	5	50,842	66,094	151,577		2%
		6	44,396	57,715			
		7 (~50%)	13,114	17,048			
		LIMS	8,247	10,271		**4%**	
TOTAL	**291,944**		**216,831**	**281,831**	**281,831**		**4%**

The WRDs were designed to a maximum of 60 m in height with a 15º final overall slope angle. The WRD locations were designed with a standoff distance from the pit using the greater of 100 m or the potential zone of failure projected from the intersection of the pit wall with the top of fresh rock surface at an angle of 25º plus 40 m. The shape of the dumps was also modified to ensure natural drainage channels remained clear and ponding at the toe of the dump was minimised.

Prior to commencement of mining the disturbed areas will be cleared and the topsoil removed and stored in various stockpiles around the site. These have been strategically located to minimise haulage distances both during stripping and when reclaimed for rehabilitation of the waste rock dumps. The topsoil locations are shown in the general site layout plan, Figure 16-7.

16.3.3. ROM Pad

The proportion of direct tip into the primary crusher is estimated to be approximately 60%. This is based on the disparity between the primary crusher and the excavator. The primary crusher is designed to operate at a nominal 1,265 t/hr for 21.7 hours per day which equates to approximately seven (7) loads per hour for the 180-t payload dump truck. The excavator is estimated to operate at 2,080 t/hr for 18 hours per day which equates to approximately 11.5 loads per hours.

Ore will be hauled to the ROM pad and direct tipped into one of two crusher pockets or placed on temporary finger stockpiles from a skyway for later rehandle using a FEL. Each of four fingers has been designed with a capacity of 96,000 dt ore, sufficient for 14 days of feed. The ROM pad has been designed to manage haul traffic with two-way traffic on and off the skyway and one-way traffic when direct tipping to the crusher as illustrated in Figure 16-6.

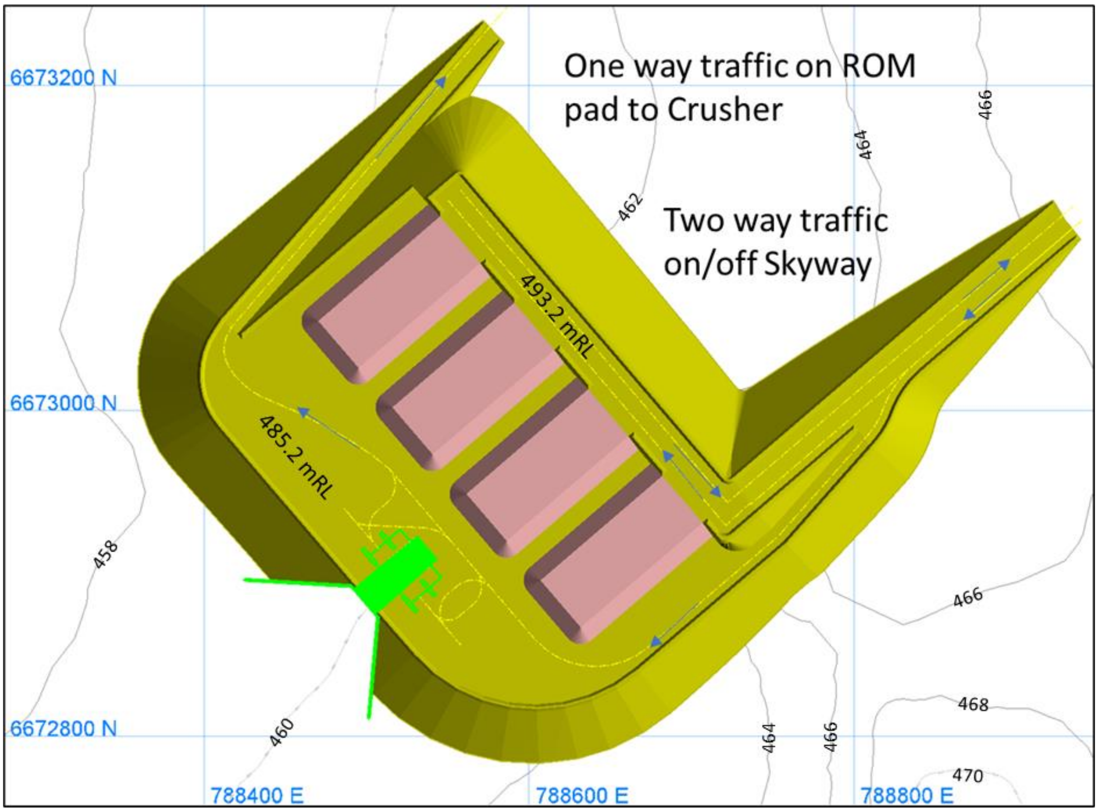


Figure 16-6: ROM Pad and Skyway traffic management

16.3.4. Ore Stockpiles

The blending strategy to manage silica levels reporting through to the concentrate requires stockpiling on long term stockpiles. All material above a DTR cut-off of 29% was categorised as primary feed. The material below 29% DTR was split into low and high silica categories as follows:

- Low Silica stockpile: SiO2 in concentrate < 6.7%; and
- High Silica stockpile: SiO2 in concentrate >= 6.7%

Two stockpiles were designed, each with a maximum capacity of 4.0 Mt. These are located adjacent to the ROM pad.

16.3.5. Mine Haul Roads

The overall strategy for haul road design was for a central road linking the Moonshine and Moonshine North pits. The ROM pad was located close to the centre of mass between the two pits. Access to the waste dumps branched of the main haulage corridor providing flexibility for dumping of waste material.

A temporary road was included that passed between Stage 1 and Stage 3 to provide access to the south-east WRD. This road has been designed with a cutting and effectively mines some material from Stage 5 earlier than required but eliminates the requirement for hauling up and down over the ridge that extends along the length of the Moonshine pit.

The design of the mine haul roads is based on an operating width of 35 m for two-way traffic with safety bunds on either side. Construction of the roads used an average fill depth of approximately 1.0 m with minimal cut. Outer slopes are battered down at an angle of 1:3.

A drainage channel crosses the main haul road between the north-west dump and the process plant. The haul road has been elevated in this area over a series of eight 1200 mm spiral culverts of 40 m in length. The number of culverts was determined for draining the maximum monthly rainfall of 178 mm from the 9 km^2 catchment area to the east of the road, based on a flow rate of 150 L/min.

The haul road will be constructed in phases as summarised in Table 16-6.

Table 16-6: Length and timing for haul road construction

	Description	Length (m)	Construction Period
Road 1	Stage 1 to ROM pad & SE WRD	2,028	Jan 2025
Road 2	Stage 2 to ROM pad & NW WRD	2,057	Nov 2026
Road 3	Stage 5 to join Road 1 & SW WRD	768	Q4 2032
Road 4	Stage 6 to join Road 3	1,446	2036
Road 5	Stage 7 to join Road 1	274	2044

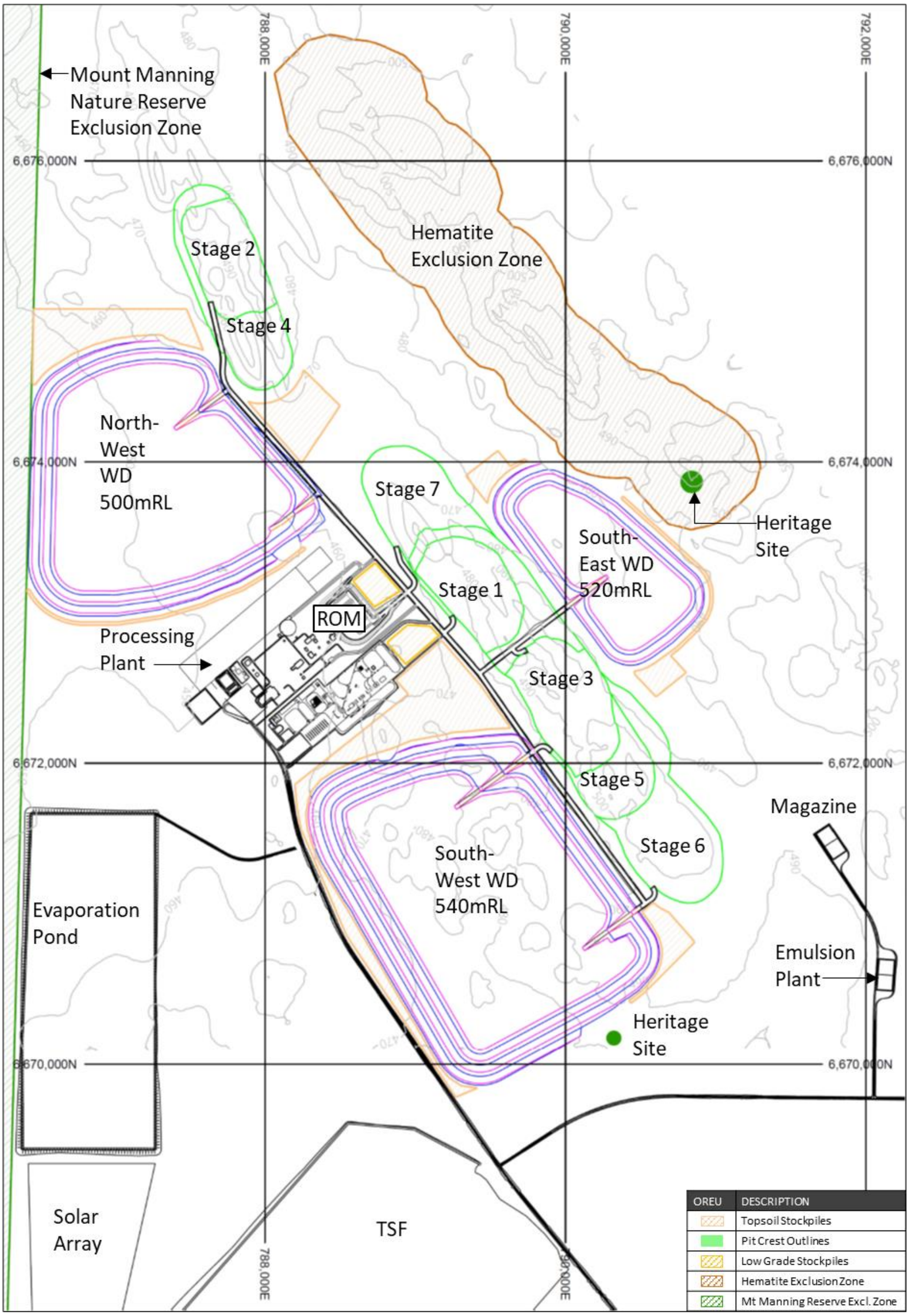


Figure 16-7: General Site Layout plan

16.4. Production Schedule

16.4.1. Production Schedule Optimisation

Mine production scheduling is the process of assigning tonnes and grades of ore and waste to time periods, with the objective of developing a production plan that is operable and meets the production objectives and grade criteria.

The mine production schedule was developed using the Maptek® Evolution™ scheduling package (EVO). The module used for schedule optimisation was Evolution™ Strategy which operates at the bench level and maximises NPV using cut-off grade theory and stage sequencing. The setup parameters are similar to the pit optimisation with the key difference being that the results provided apply practical mining constraints to bench turnover, number of mining areas and a mining rate that is balanced with the overall plant feed requirements.

The mining and process production schedule was developed to maximise NPV for a process rate of 9.68 Mt (dry) ore to produce 3.0 Mt (dry) of magnetite concentrate. Evaluation using EVO indicated that this could be achieved with an average mining rate of approximately 40 Mt/year as the Base Case. Optimisation of the production schedule included an assessment of the bottlenecks for the process plant and logistics for product delivery to market. The options are summarised in Table 16-7.

Table 16-7: Bottleneck analysis

Case	Description	Rail (Mtpa)	Crusher (Mtpa)	Mine (Mtpa)	Change in NPV (excl. added Capex)
SC1	Base Case	3.0	9.7	40.0	-
SC2	Increase Rail Limit	3.3	9.7	40.0	3%
SC3	Increase Rail & Crusher limits	3.3	10.6	40.0	4%
SC4	Increase Rail, Crusher & Mining Limits	3.3	10.6	44.0	6%

The analysis showed small improvements in NPV for a 10% increase in the rail limit and further increases for crusher limits and mining limits. However, it was evident that as the downstream objectives were increased, the production of concentrate became less likely to meet the increased production target without an increase in the mining rate as illustrated in Figure 16-7. When the capital cost for de-bottleneck the process plant was taken into consideration the overall increase in NPV was expected to be negligible.

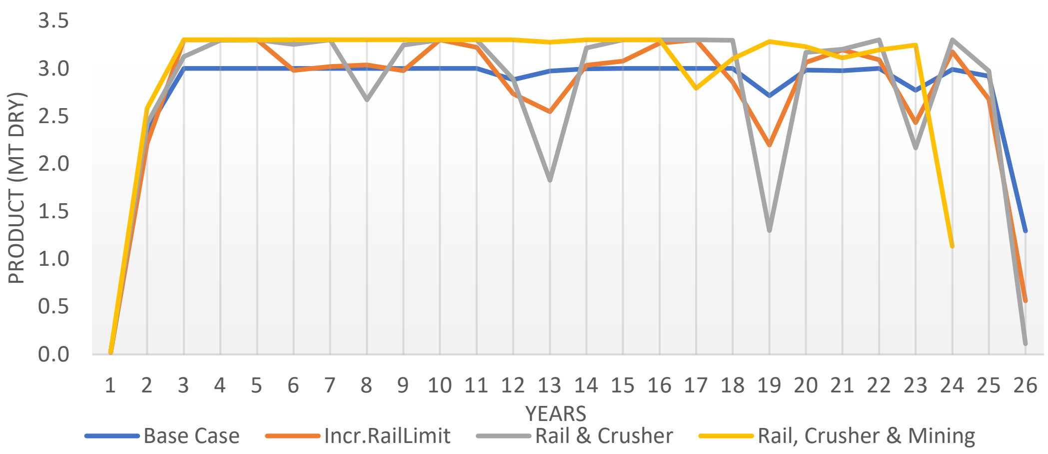


Figure 16-8: Effect on Magnetite concentrate production for change in constraints

The base case was adopted as the go-forward case and used as a guide to produce the final schedule in Evolution™ Origin which is a block-by-block meta-heuristic scheduler that uses evolutionary algorithms to find tactical solutions for multiple objectives. This provides sufficient detail for scheduling in monthly periods and permits blending constraints for manage of contaminant levels using stockpiles.

16.4.2. Mine Production Schedule

The detailed LOM production schedule was completed in monthly periods for the first four years of the mining operation followed by quarters for seven years and annually thereafter.

The objective of the LOM plan was to maximise discounted operating cash flows subject to the following constraints:

- Production target of 3.0 Mt of dry concentrate for export per year
- Crusher feed limit of 10.0 Mt (dry) at full production
- Plant ramp-up of 12 months from wet commissioning in December 2026
- Blending for the silica in concentrate grade with an upper limit of 7%
- Annual movement rate limited to 3 primary loading units based on 16.9 Mt/year in Oxide material and 13.7 Mt/year in Fresh rock
- Mining ramp up to full production over a period of 6 months with excavators delivered at one-month intervals
- Managing pre-strip to minimise pre-production costs prior to plant commissioning
- Limiting the vertical advance rate to 100 m in bulk waste and 60 m in ore zones
- Excavators limited to one move per month; and
- Setback of 100 m when mining more than 1 bench in a stage.

The overall mining strategy presented by the strategic scheduling identified targeting the higher DTR ores from Moonshine North early in the schedule and required blending with ores from Moonshine to balance the silica grades. Opening both pits at the same time required removal of 22 Mt of overburden from Stage 1 and 26 Mt of overburden from Stage 2. The trade-off between higher DTR grades, reduced pre-stripping and the silica limits showed that mining the initial stage in Moonshine provided better value than mining Moonshine North first.

The final schedule is summarised in Figure 16-9. Pre-production required a total of 28.1 Mt mined over an 11-month period comprising mostly oxide waste from Stage 1 with 164 Kt ore stockpiled for processing. After 14 years the mining rate was reduced to two excavators for 7 years before reducing to a single excavator for the final 5 years of operation.

Figure 16-9 illustrates the oxide pre-strip and fresh waste movements compared to the ore mined to the ROM pad for processing or to stockpile for blending.

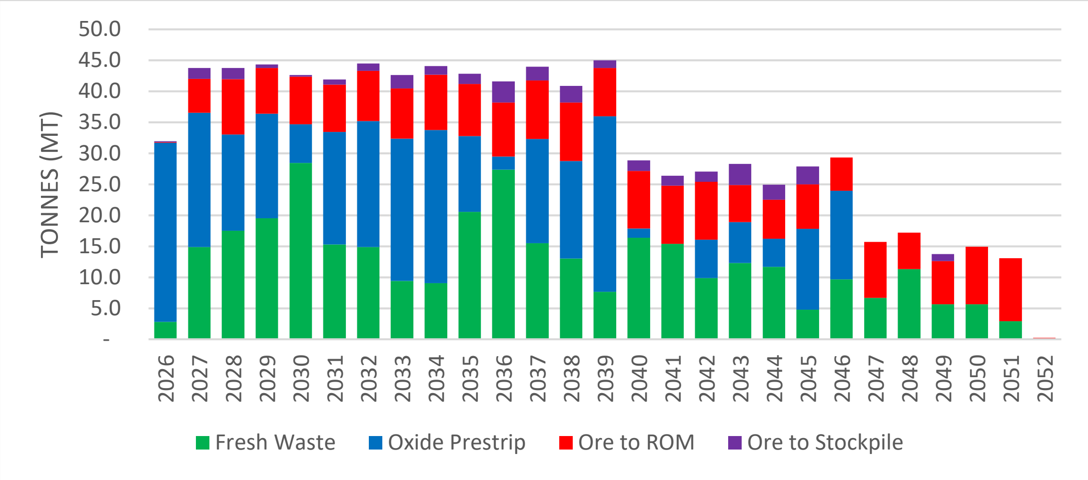


Figure 16-9: Material mined by year showing the mining sequence

Table 16-8: Annual Mine Production schedule

	Ore	Fe	SiO2	Al2O3	P	S	LOI	DTR	Waste	Total
Year	**Mt**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**Mt**	**Mt**
2026	0.2	29.5	50.0	1.20	0.044	0.95	1.94	26.2	31.7	32.0
2027	7.3	29.2	50.0	1.22	0.047	1.03	1.47	31.3	36.5	43.8
2028	10.7	27.8	50.2	1.76	0.046	1.41	1.41	31.1	33.0	43.8
2029	7.9	30.0	47.8	1.38	0.045	1.28	1.70	32.4	36.4	44.3
2030	8.0	34.5	38.3	1.91	0.056	1.88	3.88	34.6	34.7	42.7
2031	8.5	33.2	39.3	1.57	0.055	1.39	3.72	36.0	33.4	41.9
2032	9.3	32.5	40.8	1.62	0.054	1.37	3.30	35.5	35.2	44.5
2033	10.3	29.3	47.9	1.35	0.049	0.95	2.25	32.2	32.3	42.6
2034	10.3	28.0	49.2	1.40	0.048	0.94	2.02	31.5	33.8	44.1
2035	10.0	28.0	50.6	1.32	0.047	0.87	1.47	31.4	32.8	42.8
2036	12.1	28.2	51.1	1.17	0.047	0.91	1.84	30.1	29.5	41.6
2037	11.7	27.2	52.2	1.07	0.047	0.83	1.53	30.6	32.3	44.0
2038	12.1	26.2	52.5	1.41	0.047	0.89	1.46	29.9	28.8	40.9
2039	9.0	26.9	52.4	1.35	0.045	1.06	1.18	31.3	36.0	45.0
2040	11.0	27.2	51.8	1.36	0.045	1.05	1.11	31.1	17.9	28.9
2041	11.0	28.1	51.5	1.15	0.046	1.03	1.22	31.2	15.4	26.4
2042	11.0	27.9	51.5	1.33	0.045	1.17	1.21	31.3	16.1	27.0
2043	9.4	26.8	52.5	1.30	0.046	0.90	1.51	30.0	18.9	28.3
2044	8.7	26.9	51.9	1.45	0.047	0.88	1.56	30.5	16.2	24.9
2045	10.1	27.1	53.1	1.07	0.044	0.82	1.36	31.1	17.8	27.9

2046	5.4	27.7	51.7	1.26	0.047	0.91	1.31	32.2	24.0	29.3
2047	9.0	26.9	52.2	1.44	0.046	1.08	1.51	30.0	6.7	15.7
2048	5.9	26.9	51.7	2.01	0.045	1.45	2.12	28.3	11.3	17.2
2049	8.1	27.4	51.2	1.76	0.045	1.15	1.62	31.4	5.7	13.8
2050	9.3	27.1	52.5	1.73	0.044	1.08	1.56	30.0	5.6	14.9
2051	10.2	26.9	53.6	1.53	0.044	0.81	1.29	29.5	2.9	13.1
2052	0.2	25.1	57.0	1.01	0.046	0.44	0.89	27.8	0.0	0.2
TOTAL	**236.6**	**28.2**	**50.1**	**1.42**	**0.047**	**1.07**	**1.75**	**31.3**	**624.9**	**861.5**

Initially the blending stockpiles carry minimal tonnes as most of the low DTR material is diverted to the process plant to manage the silica grade. The blending stockpiles reach a peak capacity in 2046 with a total of 15.4 Mt which is reclaimed over the following 10 years (refer Figure 16-10).

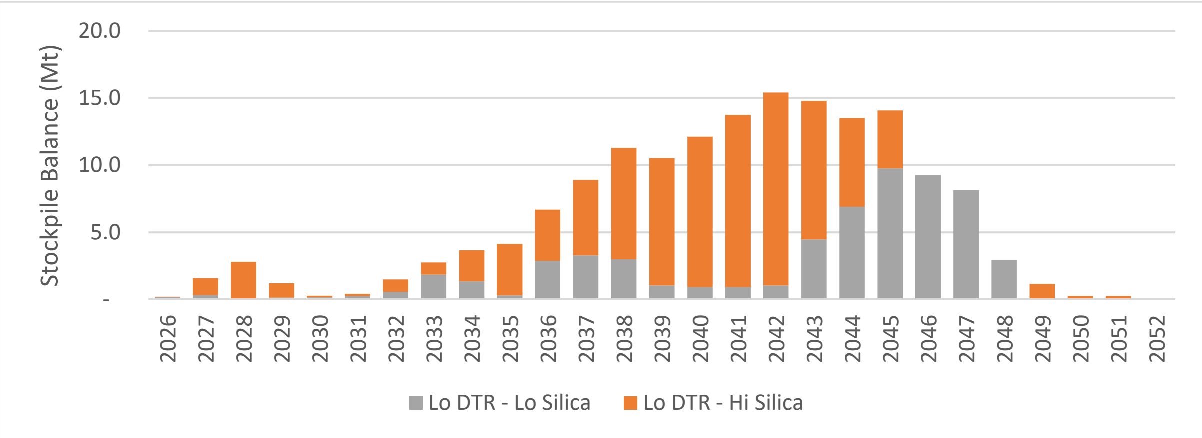


Figure 16-10: Material mined by year showing the mining sequence

The mining sequence and vertical advance rate for each stage is presented in Table 16-9 below. The highest vertical advance rate of 65 m occurs in Stage 1 driven by pre-strip required to expose ore from processing in December 2026. Once the ore is exposed in Stage 2, the vertical advance rate is low due to the need to balance mining ore and pre-stripping the next stage.

Table 16-9: Annual vertical advance by Stage (metres)

Year	Stage 1	Stage 2	Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
2026	65	10					
2027	40	35					
2028	45	25	40				
2029	50	25	35				
2030		30	40	20			
2031			35	20			
2032			40	35			
2033			20	25	40		
2034			20	30	25		
2035			10	45	25		
2036				10	35	10	
2037					25	15	
2038					25	10	
2039					15	25	
2040					20	10	
2041					20	15	
2042					65	15	35
2043						25	10
2044						25	5
2045						25	15
2046						15	25
2047						25	10
2048						5	25
2049						25	15
2050						10	35
2051							65
2052							10

16.4.3. Mill Production Schedule

The process production schedule is presented in Table 16-10 with the concentrate production results shown in Table 16-11. The schedule is based on a construction completion date by the end of September 2026 with 8 weeks dry commissioning and first ore in December 2026. The 12-month process ramp up is illustrated in Figure 16-11.

The process feed schedule varies in throughput rate depending on the DTR grade of the ore. Figure 16-12 shows the DTR initially trending up over the first six years trending down over the remaining mine life. Material is reclaimed from stockpile to manage the mass yield and silica level in concentrate which remain below 7% in most periods.

Following the ramp up periods, the annual concentrate production remains constant at 3.0 Mt (dry) each year to 2051 as shown in Table 16-10. Any remaining stockpiled material is processed in the first month of 2052.

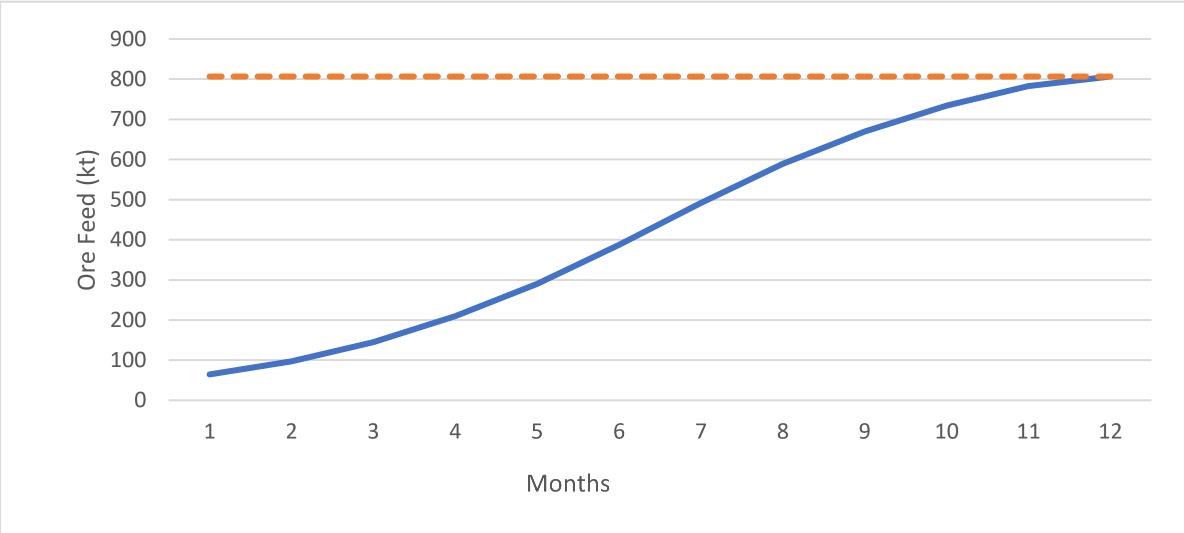


Figure 16-11: Plant production Ramp up

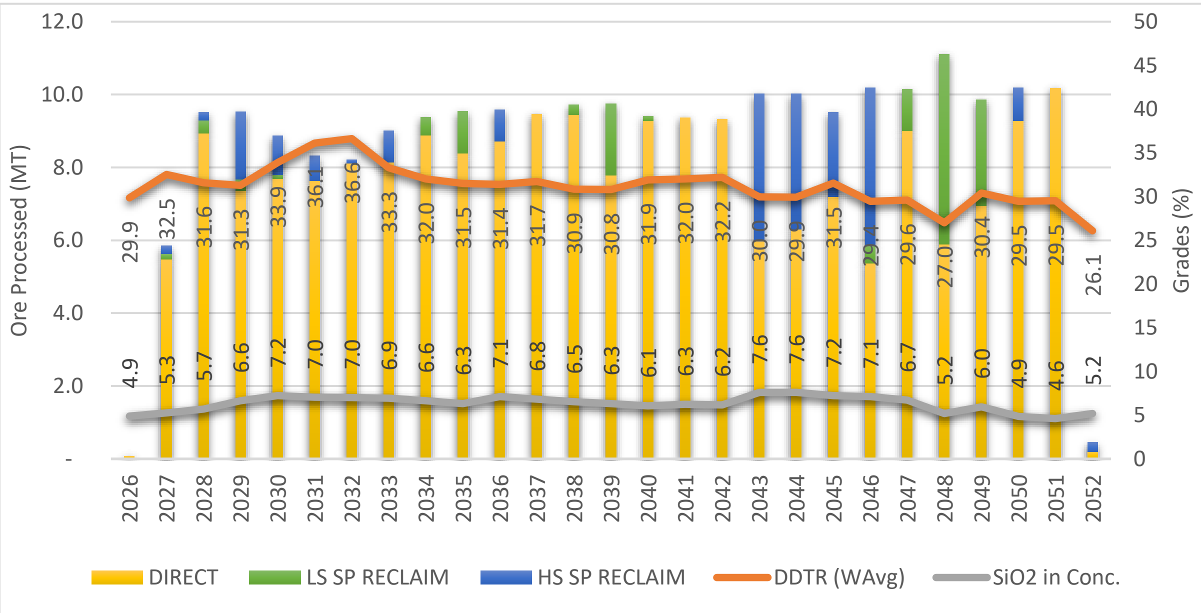


Figure 16-12: Process Feed showing mass recovery based on DTR grade

Table 16-10: Annual Process plant feed

	Ore Feed	Fe	SIO_2	Al_2O_3	P	S	LOI	DTR
Year	**Mt**	**%**	**%**	**%**	**%**	**%**	**%**	**%**
2026	0.07	27.1	50.9	1.63	0.044	0.88	1.76	29.9
2027	5.8	29.4	50.0	1.19	0.047	0.90	1.32	32.5
2028	9.5	28.0	50.2	1.69	0.046	1.34	1.41	31.6
2029	9.5	29.7	48.2	1.40	0.046	1.35	1.69	31.3
2030	8.9	33.6	39.8	1.91	0.054	1.25	3.56	33.9
2031	8.3	33.3	39.2	1.59	0.055	0.89	3.71	36.1
2032	8.2	33.0	40.4	1.59	0.054	0.74	3.16	36.6
2033	9.0	30.2	46.2	1.40	0.050	0.65	2.41	33.3
2034	9.4	28.4	49.2	1.36	0.048	1.07	1.96	32.0
2035	9.5	27.9	50.8	1.35	0.047	1.35	1.51	31.5
2036	9.6	28.7	51.0	1.00	0.047	0.80	1.63	31.4
2037	9.5	27.7	51.8	1.02	0.048	1.11	1.46	31.7
2038	9.7	26.4	52.2	1.44	0.047	1.50	1.47	30.9
2039	9.7	27.1	52.1	1.45	0.046	1.49	1.38	30.8
2040	9.4	27.5	51.6	1.34	0.044	1.41	1.05	31.9
2041	9.4	28.3	51.3	1.14	0.046	1.18	1.21	32.0
2042	9.3	28.2	51.3	1.30	0.046	1.30	1.18	32.2
2043	10.0	27.2	51.4	1.27	0.046	1.35	1.83	30.0
2044	10.0	26.9	52.3	1.30	0.046	1.62	1.50	29.9
2045	9.5	27.5	52.7	1.00	0.045	1.21	1.30	31.5
2046	10.2	27.0	52.2	1.34	0.046	1.72	1.40	29.4
2047	10.1	26.6	52.3	1.45	0.046	1.51	1.57	29.6
2048	11.1	25.8	52.3	1.87	0.046	1.66	2.03	27.0
2049	9.9	26.8	52.2	1.52	0.045	1.44	1.51	30.4
2050	10.2	26.8	52.4	1.85	0.044	1.70	1.68	29.5
2051	10.2	26.9	53.6	1.53	0.044	1.34	1.29	29.5
2052	0.5	24.6	54.4	1.72	0.044	2.47	2.07	26.1
Total	**236.6**	**28.2**	**50.1**	**1.42**	**0.0**	**1.3**	**1.8**	**31.3**

Table 16-11: Annual concentrate production

	Concentrate	Fe	SiO_2	Al_2O_3	P	S	LOI
Year	**Mdt**	**%**	**%**	**%**	**%**	**%**	**%**
2026	0.02	67.2	5.2	0.06	0.016	0.63	0.09
2027	1.9	67.4	5.2	0.08	0.013	1.04	0.08
2028	3.0	66.9	5.8	0.13	0.017	1.30	0.05
2029	3.0	66.1	6.4	0.19	0.024	1.27	-0.05
2030	3.0	64.7	7.1	0.34	0.047	1.85	-0.33
2031	3.0	64.9	7.0	0.29	0.059	1.43	-0.59
2032	3.0	65.2	7.0	0.24	0.046	1.34	-0.71
2033	3.0	65.8	6.8	0.16	0.028	1.00	-0.78
2034	3.0	65.8	6.8	0.16	0.022	1.00	-0.69
2035	3.0	66.1	6.6	0.13	0.014	0.90	-0.45
2036	3.0	66.1	6.9	0.12	0.016	0.79	-0.47
2037	3.0	66.1	7.0	0.10	0.016	0.77	-0.39
2038	3.0	66.2	6.8	0.11	0.015	0.87	-0.27
2039	3.0	66.1	6.8	0.14	0.014	1.03	-0.35
2040	3.0	66.6	6.1	0.12	0.012	0.96	-0.18
2041	3.0	66.6	6.3	0.13	0.012	0.95	-0.11
2042	3.0	66.6	6.2	0.12	0.012	1.11	0.00
2043	3.0	66.1	6.5	0.12	0.017	0.93	-0.12
2044	3.0	65.9	6.8	0.12	0.016	0.92	0.12
2045	3.0	66.2	6.7	0.10	0.013	0.83	-0.05
2046	3.0	66.0	6.7	0.15	0.015	1.20	-0.07
2047	3.0	65.9	6.9	0.14	0.015	1.07	-0.12
2048	3.0	65.9	6.8	0.14	0.015	1.22	0.10
2049	3.0	66.3	6.6	0.13	0.012	1.01	0.01
2050	3.0	67.5	4.9	0.09	0.011	1.16	0.28
2051	3.0	67.9	4.6	0.06	0.010	0.81	0.23
2052	0.1	67.4	5.1	0.07	0.011	1.62	0.34
TOTAL	**74.1**	**66.2**	**6.5**	**0.1**	**0.020**	**1.1**	**-0.2**

16.5. Waste Requirements for Civil Work

The initial site establishment in the mining area will require approximately 950 kt of waste rock for construction of the ROM pad, stockpile pads and haul roads as follows:

- ROM pad – 900 kt of waste rock for bulk fill.
- Haul roads – 30 kt for road base initially and a further 40 kt over the life of mine; and
- Stockpile pads – 20 kt for sheeting.

16.6. Mine Operations and Equipment Selection

16.6.1. Mine Operations Approach

The Moonshine and Moonshine North pits will be mined using conventional open pit mining methods based on 350-400 t class hydraulic excavators loading 180 t class rear dump trucks. The operation is proposed using experienced mining contractors with Macarthur maintaining orebody definition, quality control and medium to long term mine planning functions and management.

Four of seven mining service contractor contacted submitted unit rates for the mining services including:

- Supply of personnel, equipment and mining infrastructure required for the mining services excluding diesel fuel which is to be supplied by Macarthur.
- Mobilisation of buildings, equipment, and personnel.
- Clearing and stripping of suitable material from all disturbed areas into discrete stockpiles.
- Construction of haul roads and light vehicle service roads in the mine area.
- Construction of the ROM pad and skyway using bulk waste.
- Grade control drilling.
- Drilling and blasting of ore and waste on 10 m benches.
- Load and Haul utilising 350-400 t class excavators and 180 t class haul trucks mining on 5 m high flitches.
- Hauling ore to the ROM pad where it will be direct fed to the crusher ore placed onto a finger from skyway of stockpile adjacent to the ROM pad.
- Rehandle of ore from Rom fingers or adjacent stockpiles.
- Ongoing pit dewatering from in-pit sumps.
- Rehabilitation of waste dumps and roads.

A shadow estimation based on contractor mining was also developed in parallel with the Contractors estimate. The final operating cost estimate was derived using the Contractor's rates applied to the final schedule physicals.

16.6.2. Roster Schedules

The Contract mining operation will be conducted with two 12-hours shifts per day. Both Owners and Contractors management, technical and support personnel will work a 10-hour day shift. All personnel will be sourced from either Perth or Kalgoorlie on fly-in fly-out basis. Details of the rosters used for the estimation of manning for accommodation and flights is shown in in Table 16-12.

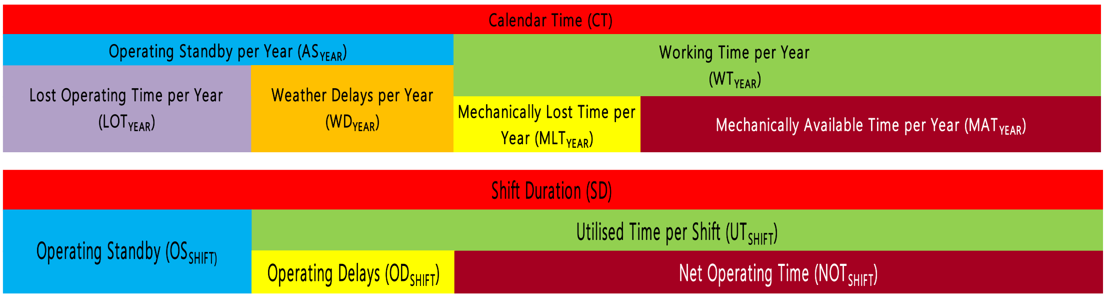
Table 16-12: Workforce Rosters

Roster Details		Shift 1	Shift 2	Shift 3
Type	**Unit**	**Management**	**Ops Day**	**Ops Shift**
Shifts per Day	#	1 x 10 hrs	1 x 10 hr	2 x 12 hrs
Roster	Days ON/OFF	9/5	14/7	14/7
Number of Crews	#	1	1	3
Annual Leave & Public Holidays per Year (Days)	Days	20	20	20
Days Worked per Year	Days	221	229	229
Annual work hours	hrs	2211	2293	2752
Annual Leave Cover Allowance	%	9.0	8.7	8.7
Personal Leave Allowance	%	4.5	4.4	4.4

16.6.3. Equipment Usage Model Assumptions

The equipment time model used for the cost estimate is based on the time model as shown in Table 16-13.

Table 16-13: Time Model Definitions



	Description	Definition
Annual Definitions	Calendar Time (CT)	Days per Year
	Lost Operating Time per Year (LOT_{YEAR})	Planned total operation shutdown (i.e., public holiday)
	Weather Delays per Year (WD_{YEAR})	Total operation shutdown due to inclement weather
	Operating Standby per Year (OS_{YEAR})	$LOT_{YEAR} + WD_{YEAR}$
	Working Time per Year (WT_{YEAR})	$CT - OS_{YEAR}$
	Mechanically Lost Time per Year (MLT_{YEAR})	

	Mechanically Available Time per Year (MAT_{YEAR})	$MAT_{YEAR} = WT_{YEAR} - MLT_{YEAR}$
Shift Definitions	Available Time per Shift (AT_{SHIFT})	Available time per shift defined as the total shift duration
	Operating Standby per Shift (OS_{SHIFT})	Time not operating with engine off (i.e., lunch, fuelling etc.)
	Utilised Time per Shift (UT_{SHIFT})	$UT_{SHIFT} = AT_{SHIFT} - OS_{SHIFT}$
	Operating Delays (OD_{SHIFT})	Time not productive with engine on (i.e., shift change over, manoeuvring etc.) This is divided into general delays and those specific to loading and drilling units
	Net Operating Time (NOT_{SHIFT})	$NOT_{SHIFT} = UT_{SHIFT} - OD_{SHIFT}$

Table 16-14: Equipment Availability and Utilisation

Time Component					Unit	Truck	Shovel	FEL	Drill	Dozer	Grader	Water Cart	Wheel Dozer
Scheduled time	Working Time			Shift Duration	(hrs)	12	12	12	12	12	12	12	12
	Operating Standby (Engine Off)			Lunch Break	(mins)	60.0	60.0	60.0	60.0	60.0	60.0	60.0	60.0
				Equipment Inspection	(mins)	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0
				Fueling/Service	(mins)	20.0	0.0	10.0	0.0	10.0	10.0	10.0	10.0
				Safety Meeting / Briefing / Ad Hoc Meetings	(mins)	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
				Equipment Specific Operating Standby (ESOS)	(mins)	1.0	1.0	1.0	0.8	0.8	0.8	0.8	0.5
				Operating Standby per shift	(Work hrs.)	1.6	1.3	1.5	1.3	1.4	1.4	1.4	1.4
				Time Equipment Operating per shift	(Work hrs.)	10.38	10.72	10.55	10.72	10.55	10.55	10.55	10.56
				Gross Operating Time per Year (GOT)	(Work hrs.)	7,330	7,565	7,448	7,567	7,450	7,450	7,450	7,454
				Utilisation (U = GOT/WT)	(%)	86.5%	89.3%	87.9%	89.3%	87.9%	87.9%	87.9%	88.0%
	Gross Operating Time	Mechanical Availability		Planned Mechanical Availability	(%)	90%	90%	88%	83%	83%	85%	83%	83%
				Unplanned Breakdown Allowance	(%)	3%	3%	3%	3%	3%	2%	3%	3%
				Downtime during Operating Delays	(%)	0%	0%	0%	0%	0%	0%	0%	0%
				Total Mechanical Availability	(%)	87.0%	87.0%	85.0%	80.0%	80.0%	83.0%	80.0%	80.0%
				Mechanically Available Time per Shift	(Eng. hrs.)	9.03	9.32	8.97	8.58	8.44	8.76	8.44	8.45
				Mechanically Available Time per Year (MAT)	(Eng. hrs.)	6,377	6,582	6,331	6,054	5,960	6,184	5,960	5,963
		Utilised Time	Operating Delay	Shift Change/Start-Up	(mins)	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0
				In-Cab Shift Breaks	(mins)	20.0	20.0	20.0	20.0	20.0	20.0	20.0	0.0
				Blasting Delay	(mins)	6.4	6.4	0.0	6.4	0.0	0.0		0.0
				Inter-bench Movement	(mins)	3.9	3.9		5.9				
				Inter-stage Movement	(mins)	2.0	2.0						
				Face Preparation - Shovel	(mins)	15.0	15.0						
				Operating Loss	hrs/shift	1.12	1.12	0.67	0.87	0.67	0.67	0.67	0.33
			Net Op. Time	Net Operating Time per shift	(Op. hrs.)	7.91	8.20	8.30	7.70	7.78	8.09	7.78	8.11
				Net Operating Time per Year (NOT)	(Op. hrs.)	5,780	5,966	5,979	5,614	5,629	5,840	5,629	5,798
				SMU Factor (SMU = NOT/MAT)	(%)	90.6%	90.6%	94.4%	92.7%	94.4%	94.4%	94.4%	97.2%

16.6.4. *Mine Management and Technical Services*

The Mining Manager will hold the statutory position of the Quarry Manager with the Mining Superintendent as the nominated person when the Mining Manager is off site.

The Mining management and technical services team will total 36 personnel during full production. The organisation structure of the team is presented in Figure 16-3. The team will reduce to 32 in Year 15 when the mining rate is reduced to two excavators and 22 when the mining rate is reduced to one excavator.

At cessation on mining, the technical service team will reduce to four personnel to manage the reclaim of ore from stockpile and rehabilitation of waste dumps and haul roads.

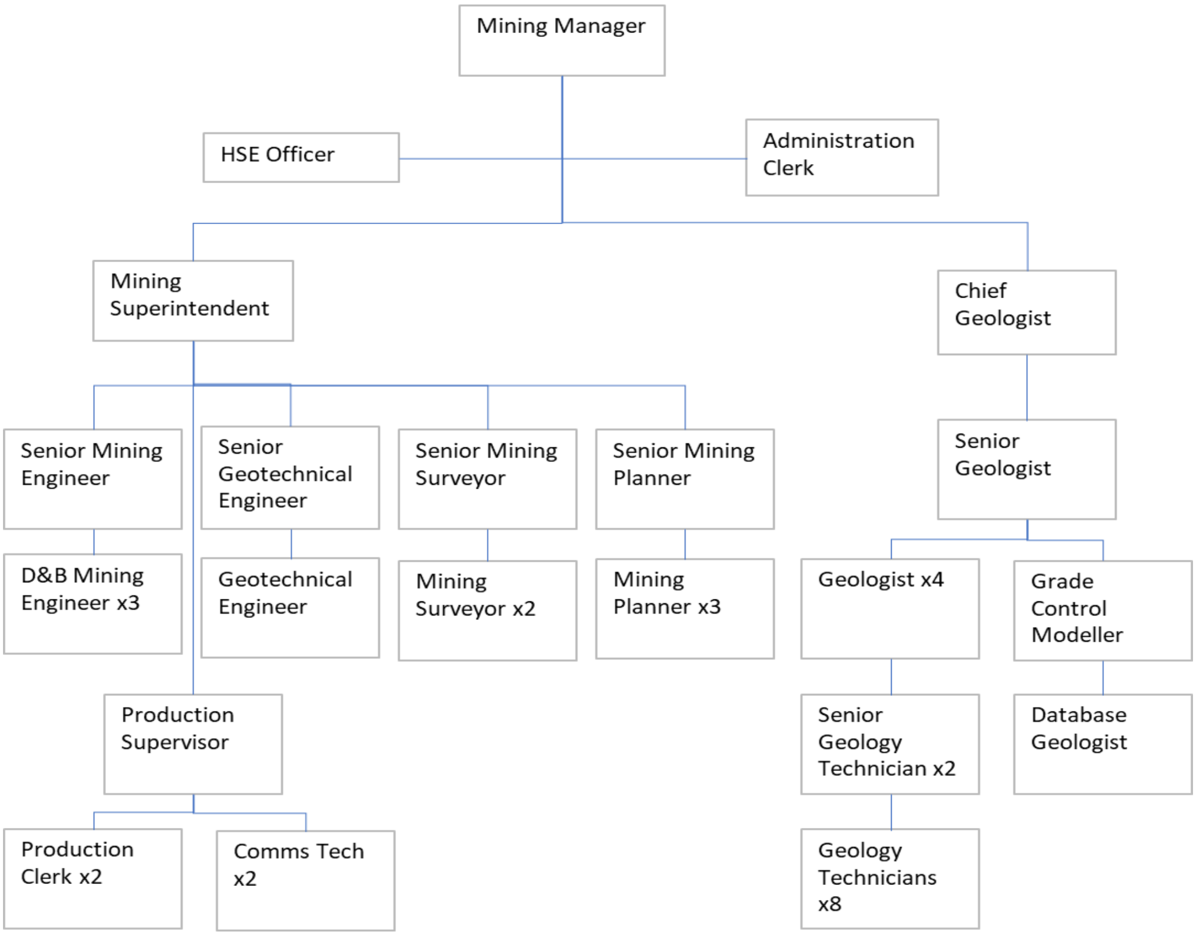


Figure 16-13: Technical Mining Services Organisation Structure

16.6.5. *Mine Maintenance and Support Equipment*

Supply of support equipment and maintenance of the mining equipment is the responsibility of the Mining contractor with all costs included in the rates. The Contractor has included the following in the list of support equipment for the operation:

- Mobile pumping Unit (MPU) truck for explosive delivery x1
- Integrated tool carrier x1
- Lighting towers x12
- Light vehicles x12

- Heavy Service vehicle x1
- Mobile crusher for stemming and road base x1
- Bus x1; and
- Telehandler x1

16.6.6. Fleet Management

The Mining Contractor will be responsible for the fleet management and dispatch systems. It is assumed that the Owner will have access to sufficient data from the Contractors system to track material movements between source and destination. An allowance is included in the Owners Technical Service team for analysis and

16.6.7. Pit Slope Monitoring

An allowance for pit wall monitoring has been included in the overheads for the Macarthur Technical services team. It has been assumed that the latest GPS systems will be employed.

16.6.8. Dewatering

Groundwater inflows anticipated by PSM are either from shallow inflows near the base of oxidation, or within the relatively tightly jointed fresh rock mass. The low in flow of groundwater will be managed using in-pit sumps excavated below the level of the lowest operating bench in each stage. The mining contractor is responsible for incidental water flows into the pit up to 5 L/sec using trailer mounted pumps pumped into a watercart where it will be dispersed on to the haul roads and working areas for dust suppression.

16.6.9. Aggregate Plant

The Mining Contractor is responsible for supply of aggregate and has include mobilisation of a mobile crushing plant for this purpose.

16.6.10. Ore Control

The grade control plan will be managed by Macarthur with drilling conducted using dedicated drill rigs supplied by the Mining Contractor. Grade control drilling will be angled at approximately 60 degrees from the horizontal to intersect the mineralisation approximately perpendicular to the dip. Reverse circulation (RC) drilling will be used for quality control and to maximise sample recovery. Holes will be campaign drilled at 140 mm diameter and will target coverage of multiple benches where possible to minimise interruptions to other pit operations.

Samples will be assayed at the on-site laboratory for both head grade and Davis Tube Recovery (DTR). The results will be utilised by the geology team to model the ore boundaries and silica contaminant levels for blending.

Blast movement will be incorporated into the ore blasts with one added hole per 8 blast holes drilled 102 mm diameter for installation of blast movement monitoring sensors. Ore boundaries will be adjusted on each flitch after determining blast vector adjustment and supplied to the contractors for excavation using high precision GPS systems installed in the mining equipment.

Wherever possible. the mining direction will be managed to face up the ore cleanly by mining across the strike of the orebody from the hanging wall side.

16.6.11. Production Drilling and Blasting

The fresh rock at Lake Giles is very hard and massive and will require significant explosive energy to deliver suitable fragmentation for both excavation and processing of the ore. A blasting analysis and optimisation study was undertaken using the rock mass data supplied by PSM with the assumption that dewatering was completed in advance of mining to provide dry blasting conditions. The criteria for fragmentation, for waste was based on bucket dimensions and ore based on criteria supplied by Engenium for the crusher. The resultant fragmentation curves are shown in

Figure **16-14**.

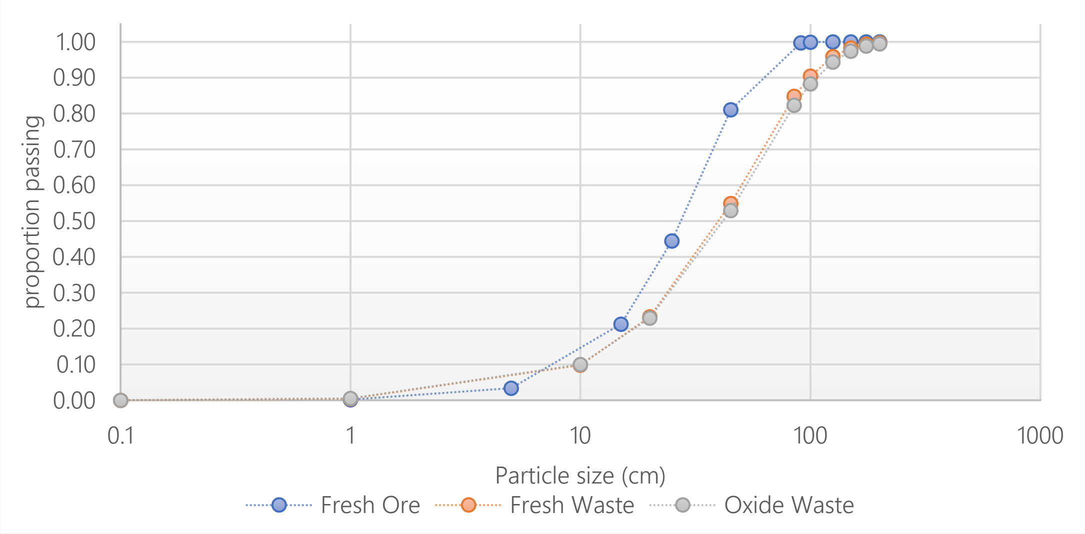


Figure 16-14: Blasting fragmentation curves for Ore and Waste

The drilling parameters used in the blasting analysis were derived using the web-based Atlas Copco calculator for a down-the-hole hammer rig with air delivery of 41 m3/min at 24 bar. The penetration rate for the selected hole size of 203 mm bit size was estimated at 29.1 m/hr with an overall rate of 23.1 m/hr after including delays for setup and manoeuvring between holes. The hole size of 203 mm was larger than the 165 mm hole size for the rates provided by the contractor, therefore, the unit rates were increased by 5% based on the proportional increase for an equivalent change in hole size derived within the shadow estimate. Total annual productivity for the primary production drill was estimated at 189,000 m/year thereby requiring 4 drill rigs to meet the production requirements. The contractor is expected to use GPS enabled equipment to control drilling accuracy.

Blasting activities will be undertaken by the mining contractor with bulk explosives supplied as an in-hole service. Macarthur will retain design and oversite of the blasting operations to quality control for dilution and wall protection. The loading and firing will be undertaken during daylight hours using electronic blasting systems. Like the drilling rate, the selected powder factors were higher than those submitted by the Contractor, therefore the blasting costs were increased by 5% and 30% for waste and ore respectively based on the proportional increase in powder factors derived within the shadow estimate. The selected blasting parameters are summarised in Table 16-15.

Table 16-15: Blasting Parameters

Blast design parameter	Oxide Waste	Fresh Waste	Fresh Ore
Drilling bench height (m)		10.0	
Hole Diameter (mm)	203	203	203
Burden (m)	6.3	6.0	4.2
Spacing (m)	7.25	6.9	4.85
Sub-drill (m)	1.5	1.5	1.5
Stemming length (m)	3.0	2.5	2.5
Design Powder factor (kg/m3)	0.78	0.91	1.83
Explosive type		Pumped Emulsion	
Explosive in hole density (g/cc)		1.3	
Relative Weight Strength (RWS)		1.18	
Surface delay Timing (ms/m)	18	17	14

16.6.12. Pre-Split

Presplit drilling is planned for fresh rock only. Presplit design is based on single bench heights using 102 mm drill diameter holes spaced at 1.2 m intervals. The holes will be charged using packaged explosives with a total charge length of 9.0 m. Drilling will be undertaken using a dedicated rig with an estimate penetration rate of 25 m/hr.

16.6.13. Loading

Excavation and loading of trucks will be undertaken using a 350-t hydraulic excavator with a 22 m^3 bucket. Excavators of this size are suited for mining a bench of up to 6 m with a total digging depth of 8 m which matches the flitch height of 5 m with allowance for heave. The estimated productivity of the units is presented in Table 16-16. The contractor has specified Hitachi EX3600-7 excavators with a total of three units required to meet the production rate.

Table 16-16: Loading unit Productivity

Description	Unit	Oxide	Fresh
Loading Unit bucket size	m^3	22.0	22.0
Bucket fill factor	%	85%	80%
Calculated Max. Bucket Capacity	m^3	19.6	18.5
Loose Wet Density	wmt/m^3	2.28	2.27
Rated Lift	t	39.6	39.6
Calculated Lift	t	44.8	42.0
Possible Bucket Payload	wmt	39.6	39.6
	m^3	17.4	17.4
Shovel De-rating Factor	%	100%	100%
Bucket Cycle Time	minutes	0.67	0.83
Tray Fill Factor	%	98%	98%
Dump Truck Rated Capacity (incl. FF)	m^3	105.8	105.8
Dump Truck Rated Capacity	t	183.0	183.0
Passes per truck theor.	#	4.6	4.6
Actual # Passes	#	5.0	5.0
Actual Truck Payload	wmt	183.0	183.0
Final Truck Payload	wmt	183.0	183.0
	dmt	177.7	180.3
First Bucket Drop Time	minutes	0.17	0.17
Loading spot time	minutes	0.33	0.33
Total load Time	**minutes**	**3.33**	**4.17**
Loading Unit Productivity (53 min/hr)	dmt / Eng. Hr	2,561	2,079
Utilised Engine Hours	Eng. Hr/year	6,582	6,582
	Mdmt/year	16.9	13.7

16.6.14. Hauling

Hauling of ore and waste will be undertaken using 180-t payload rear dump trucks matched to the 350-t excavator for a 4-5 pass loading cycle. The selected contractor has specified Caterpillar 789D mechanical drive trucks.

Haulage simulation was undertaken in EVO software and benchmarked using Talpac by RPM Global. The EVO haulage simulation estimates the cycle time from each block in the scheduling model to a waste block using the shortest path. Both systems use rimpull and retard curves from the original equipment manufacturers to calculate the travel times with load times as per Table 16-16 and other delays as speed limits as per Table 16-17. The haulage analysis also produced fuel consumption estimates for the haul profiles.

Table 16-17: Haulage analysis parameters

Cycle component	Speed Limit (km/hr)	Rolling resistance	Delay (sec)	Description
Loading			20	Queue time
Pit Floor	30	3.0%		
Ramp Up	Rim-pull	2.5%		
Surface road	50	2.5%	40	Intersections
Dump / Crusher	30	3.0%	120	Spot, turn, tip, lower tray
Ramp down	30	2.5%		

The cycle time and fuel burn rates were compiled into weighted averages for oxide, trans and fresh for both ore and waste on each bench by pit stage. The final cycle times reported in Table 16-18 were determined using truck productivities and the number of trucks required for each bench by pit stage, as follows:

- rounding up for up any proportion of a truck over 0.25 to keep the loading unit trucked up and adding queue time to the truck cycle time; or
- rounding down below 0.25 and derating the loading unit with no added queue times for the truck cycle times.

The haulage simulation was utilised to estimate the number of trucks required and the timing of mobilisation and demobilisation summarised in Figure 16-15.

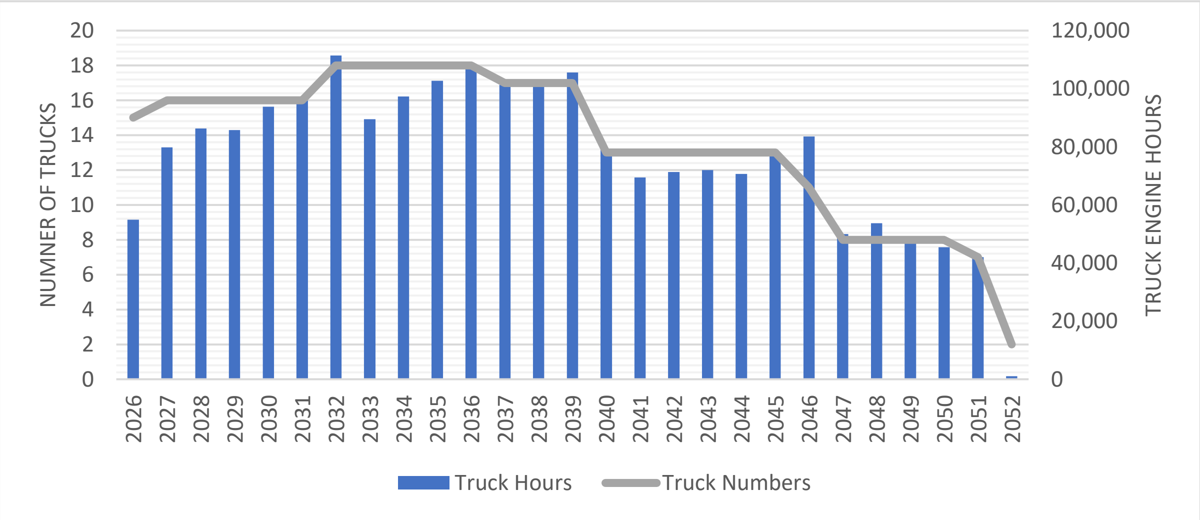


Figure 16-15: Haul Fleet and estimated engine hours

Table 16-18: Haulage cycle time analysis results (minutes)

Bench toe	Stage 1		Stage 2		Stage 3		Stage 4		Stage 5		Stage 6		Stage 7	
	Ore	Waste	Ore	Waste	Ore	Waste	Ore	Waste	Ore	Waste	Ore	Waste	Ore	Waste
500		10.8		11.5		12.7		12.4		10.4		10.6		13.5
490		10.6		11.6		12.8		12.3		11.2		11.2		13.8
480		10.9		11.8		12.5		13.3		12.3		12.7		14.0
470		13.3		12.2		12.3		15.4		13.8		13.7		15.4
460		15.4		13.3		12.3		17.3		15.4		15.4		16.3
450		16.4	20.8	14.0		12.9		18.5	28.7	16.8		16.1		18.5
440	21.9	18.5	21.3	15.3		14.0	33.8	19.7	23.6	18.4	23.9	15.6		18.6
430	19.5	18.8	23.1	16.8	22.5	15.4	21.7	21.4	28.1	19.8	21.2	16.8		20.3
420	17.2	20.6	25.0	18.0	23.6	16.7	21.8	23.0	23.2	21.3	20.4	17.9	18.1	21.9
410	17.4	21.3	25.0	19.0	24.4	17.8	23.1	23.9	23.4	23.0	21.3	19.2	20.1	22.6
400	19.3	23.1	26.1	20.8	24.1	19.2	23.1	24.7	24.0	23.1	22.0	20.0	20.1	23.6
390	19.4	23.5	26.9	21.3	24.9	19.9	23.8	27.0	24.7	23.7	23.1	21.0	19.7	24.8
380	19.9	24.2			27.0	21.4	24.7	27.1	25.5	24.7	24.0	21.9	20.0	25.6
370	20.9	25.2			27.4	23.1	25.6	28.0	25.8	25.0	24.9	23.1	21.0	27.0
360	23.1	27.0			28.1	23.3	27.0	29.0	27.0	26.0	27.0	23.9	21.6	27.0
350	23.1	27.1			28.7	23.9	27.5	30.8	27.9	27.0	27.0	24.8	23.1	28.0
340	23.4	27.8			29.1	24.3	28.3	31.2	28.6	27.7	27.9	25.6	24.1	29.7
330	24.3	28.7			29.8	24.8	25.2		29.3	28.4	28.7	27.0	24.7	30.8
320	25.5	29.7			30.8	25.6			30.8	29.5	29.5	27.4	25.2	31.0
310	27.0				31.4	27.0			30.8	30.8	30.8	28.3	27.0	32.1
300					32.1	27.7			31.5	31.3	31.5	29.0	27.0	32.9
290					33.0	28.1			32.5	32.0	33.0	29.7	28.0	34.7
280					33.7	29.3			33.0	32.6	33.7	30.8	29.2	35.3
270					34.7	30.8			33.6	32.9	34.7	31.4	28.9	34.7
260									34.7	33.0	35.1	32.3	27.0	
250									35.4	34.9	35.9	32.8		

16.6.15. Ore Handling

Once the ore is loaded into the tray of a dump truck, the ore will be hauled to the destination assigned by the Owners technical mining team. The ore destinations are:

- ROM Pad where a traffic light system or similar will direct the truck to tip the load directly into a crusher pocket or head to the skyway where the load will on a finger.
- Long-term blending stockpiles located adjacent to the ROM pad.

The ROM pad has four fingers which will be used to manage the blend. The two central finger stockpiles with store the primary feed material with the outer fingers used to store the high and low silica material. It is intended that these fingers are constructed in using end tipping and recovered from the side to assist in mixing the ore when reclaimed using a FEL.

The long-term stockpiles will be constructed initially using paddock dumping before establishing a ramp once the base is filled. Reclaim from the long-term stockpiles will be conducted using either a loader or the backup excavator loading two rehandle trucks. The trucks will haul directly to a crusher pocket to minimise rehandle.

16.6.16. Dry LIMS Reject Rehandle

The processing method uses Low Intensity Magnetic Separation (LIMS) to scalp off the non- and low-magnetic material as the first step in production of magnetite concentrate. The reject material, approximately, 12.2% of the ore feed, will be discharged from a conveyor at a rate of ~ 149 t/h to a cone with about 24 hours capacity. The reject material will be removed on a shift-by-shift basis by the mining contractor using a FEL and rehandle trucks. The rejects will be hauled to the SWWD and co-mingled with the mine waste.

16.6.17. Road and Dump Maintenance

Waste rock from each of the pits will be hauled to the waste rock dumps and placed using haul trucks end-tipping in lifts of 20.0 m vertical height with standard dozing management practice. The waste dump tip heads will be managed by the Mining Contractor using a fleet of Caterpillar D10T dozers.

ROM finger and ore stockpiles will be managed using a Caterpillar 854K wheel dozer which provides greater mobility than a tracked dozer for this task.

Road maintenance will be the responsibility of the Mining Contractor using a fleet of two Caterpillar 16M graders, and two Caterpillar 777F watercarts.

16.6.18. Mine Equipment Requirements

The contractor will mobilise the main mining fleet to site in the first quarter of 2026 and add to the haulage fleet over the following years to meet the peak fleet requirements in 2023 through to 2036. As the mining rate reduces the fleet size will reduce and equipment will be demobilised from site. The fleet changes over the life of mine are shown in Table 16-9.

Table 16-19: Contractor supplied fleet requirements

Unit	Make	Model	2026	2026-31	2023-36	2037-39	2040-46	2047-50	2051	2052
Primary Excavator	Hitachi	EX3600	3	3	3	3	2	1	1	1
Haul truck	Caterpillar	789C	15	16	18	17	13	8	7	2
Dozer	Caterpillar	D10T	5	5	5	5	3	2	2	1
Grader	Caterpillar	16M	2	2	2	2	1	1	1	1
Watercart	Caterpillar	777C	2	2	2	2	1	1	1	1
Support Excavator	Hitachi	EX1200	1	1	1	1	1	1	1	1
Wheel Dozer	Caterpillar	854K	1	1	1	1	1	1	1	1
Primary Drill	Sandvik	DR410i	4	4	4	4	3	2	2	1
Presplit drill	Sandvik	D1650i	1	1	1	1	1	1	1	1
FEL	Caterpillar	Cat993K	2	2	2	2	2	2	2	2
Rehandle truck	Caterpillar	Cat785D	2	2	2	2	2	2	2	2
GC drill	Sandvik	RC Drill Rig	1	1	1	1	1	1		

16.6.19. Mine Manpower Requirements

Mine manning numbers were estimated for the life of the operation as presented in Figure 16-16. Equipment operators and maintenance personnel are linked to the equipment hours and management and technical services personnel reduce with the number of excavators. Peak manning levels are reached in 2029 at 307 people.

The Contracting strategy provides flexibility in the manning levels for the mining operation. The manning ratio between equipment operators and maintenance staff is approximately 2:1.

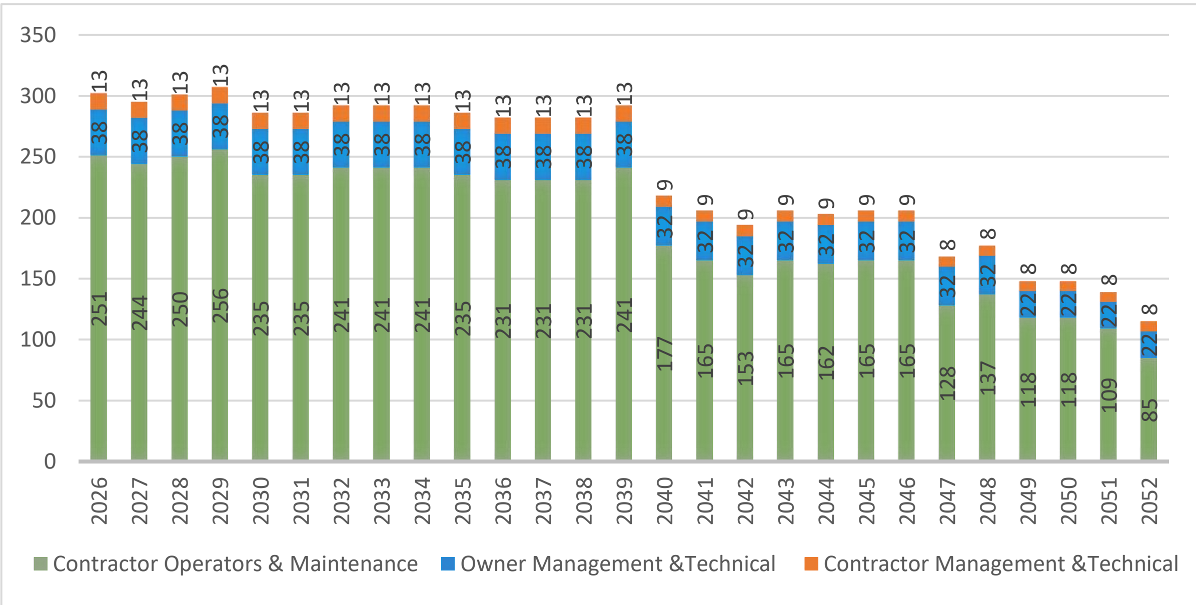


Figure 16-16: Mining workforce

17 Recovery Methods

17.1. Ore Processing

The development of the concentration process for the Project is influenced by several key elements. These include conservation of water, minimum power consumption, the competent and abrasive nature of the ore and addressing the sustainability and heritage of the project. Whilst addressing all of these issues the processing plant must also achieve efficient and economic recovery of the contained magnetite.

In order to produce 3.0 dMtpa concentrate, assuming a weight recovery of 31%, 10 Mtpa of feed to the process would be required. Two stages of conventional crushing would crush the ore to a size suitable for feed to a High-Pressure Grinding Rolls (HPGR) unit. The fine ore grinding section contains two streams in parallel each containing two stages of mills, with Low Intensity Magnetic Separation (LIMS) units after each stage. This is followed by reverse flotation and a final LIMS stage. The final concentrate moisture is reduced by pressure filtration allowing stockpiling and transport by truck.

A flowsheet for the operation is shown below.

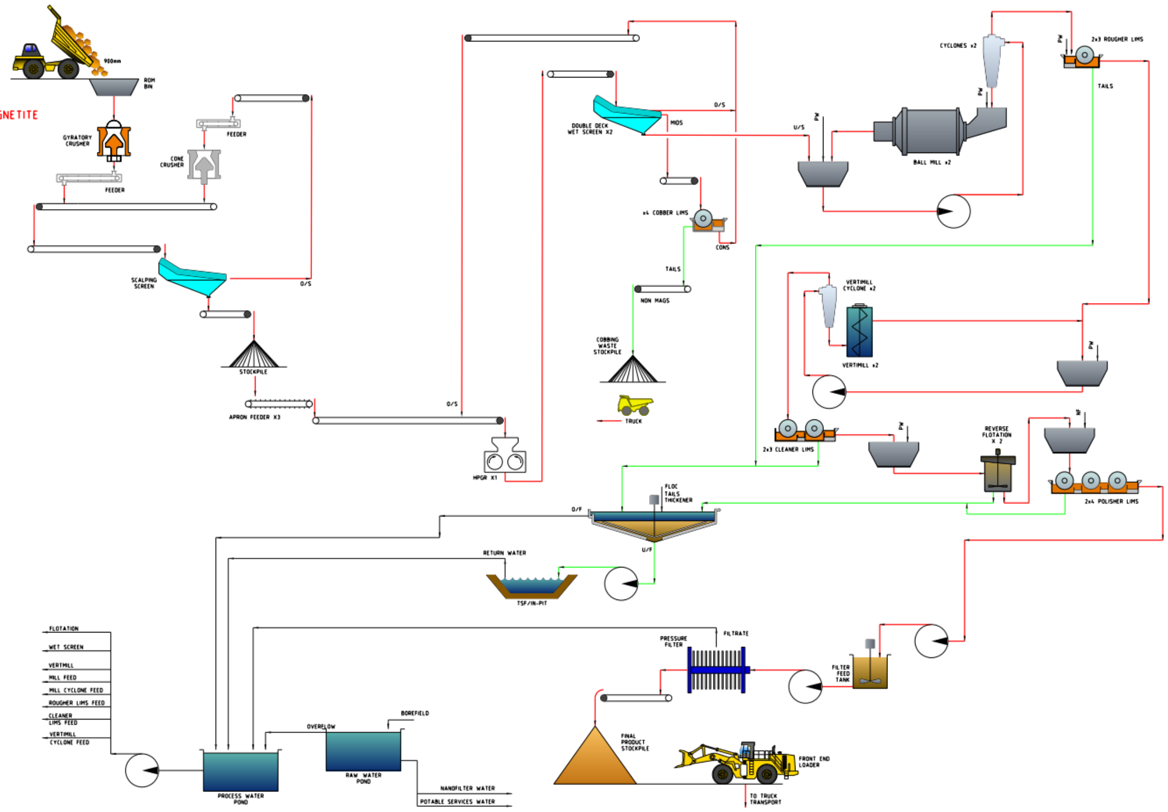


Figure 17-1: Conceptual Project Flowsheet

17.2. Primary Crusher

Ore from the mine will be delivered by dump trucks to the 50” by 65” gyratory crusher, which has two loading points. A rock breaker, installed adjacent to the crusher, breaks up oversize rocks in the feed as required. A discharge feeder and conveyor transfers the crushed ore to the 447 tonne secondary scalper feed bin. A 20-tonne crane is installed for maintenance of the crusher.

Figure 17-2: Gyratory Crusher

17.3. Secondary Scalping Screen and Stockpile

The secondary scalper feed bin is discharged by a belt feeder to distribute the material across the width of the 3.6 m by 8.5 m double deck scalping screen. The double deck screen passes both coarse fractions to the secondary crusher bin feed conveyor, while the minus 40 mm undersize material, from the bottom deck, passes to the HPGR Feed Stockpile.

17.4. Secondary Crushing

The oversize material from the scalping screen is stored in a feed bin before feeding, at a controlled rate, to the MP1000 secondary cone crusher. Monitoring the level in the crusher cavity maintains the choke feeding arrangement. If the bin level gets low then the feed stops and the crusher coasts until the bin refills sufficiently. The crushed ore returns to the scalping screed feed conveyor.

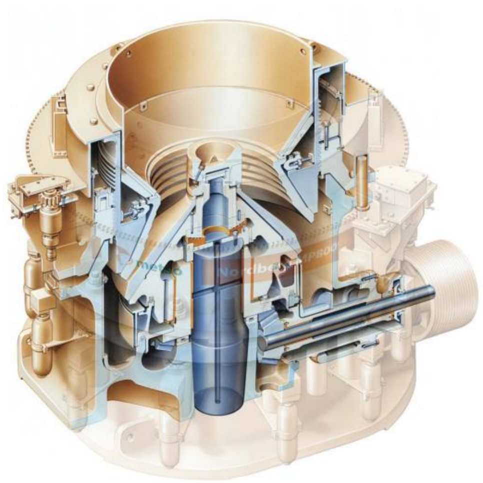


Figure 17-3: Cutaway View of Cone Crusher

17.5. HPGR and Dry LIMS

Three under-stockpile apron feeders discharge the ore from the HPGR feed stockpile and drop the material onto the HPGR feed bin conveyor. The bin discharge passes through a metal detector and then is choke fed to the HPGR feed chute.

Following size reduction by the 2.4 m by 1.65 m wide HPGR, the material would be wet screened, in parallel, on two 4.2 m by 8.5 m double-deck screens splitting at 12 and 3 mm. The underflow is split to allow parallel processing from this point forward The -3 mm material passes through the screen via a dedicated primary ball mill cyclone cluster for milling in a dedicated Ball Mill.

The mid-sized -12 mm, +3 mm material would be subject to Dry LIMS separation to remove a barren stream to reduce the milling requirement. The magnetic fraction from the Dry LIMS and the screen oversize would recirculate back to the HPGR feed conveyor.

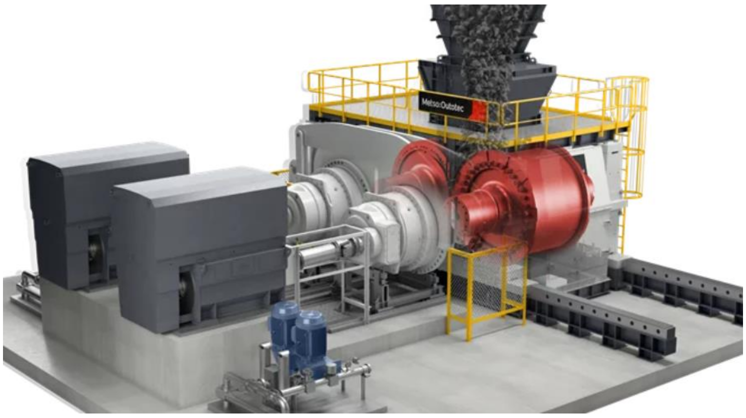


Figure 17-4: HPGR

17.6. Magnetite Processing

The beneficiation plant would operate in two parallel streams, to allow the effects of maintenance and breakdowns to be minimised as the other half of the plant can maintain operation.

17.7. Primary Grinding and Rougher LIMS

Primary milling would be by two parallel 6.1 m by 10.2 m EGL Ball Mills, fitted with 6,700 kW motors, grinding in closed circuit with their dedicated cyclones to produce a cyclone overflow size of 80% passing 106 microns to feed the 1.2 mD x 3.6 mL Rougher Wet LIMS. The roughing stage rejects barren material while maintaining a high level of magnetite recovery. The tails produced at this stage pass to the tailings section. The cyclone underflow passes directly down a launder and pipe to the Ball Mill feed.

Figure 17-5: Ball Mill

17.8. Secondary Grinding and Cleaner LIMS

Rougher concentrate needs further size reduction by two parallel VTM3000WB Vertimills®. These operate in closed circuit with cyclones to give a cyclone overflow size 80% passing 38 microns; the underflow returning to the Vertimill®. The cyclone overflow becomes the feed stream for the cleaner LIMS stage. These are 1.2 mD by 3.6 mL double drum units. The cleaner LIMS tailings stream flows to the tailings section while the concentrate will require further upgrading, by reverse flotation.

Figure 17-6: Cutaway View of Vertimill®

17.9. Reverse Flotation and Polisher LIMS

Flotation is a physical separation that uses chemicals to make one species of the mineralisation (either ore or gangue) hydrophobic so it will attach to air bubbles and flow upwards to overflow out of a designed tank. Conventional flotation removes valuable material in the overflow stream, while reverse flotation removes gangue material from the overflow whilst the valuable concentrate stream flows out the bottom discharge of the tank. The Project flowsheet includes a reverse flotation stage to reduce the concentrate silica to an acceptable level.

This reverse flotation section is, again, operated in parallel, with each bank being fed by a cleaner LIMS unit. The froth layer material from flotation is silica rich and pumped to the tailings section.

The reverse flotation concentrate flows from the base of the cell and is subject to the 1.2 mD by 3.6 mL triple drum polisher LIMS stage with a low salinity water wash on the final drum to reduce alkali metal content in the concentrate. Polisher tails pass through a polishing tank to collect any entrained fine concentrate for intermittent return to the process, before flowing to the tailings section.

17.10. Filtration and Stockpiling

Concentrate from the polisher LIMS is fed to two parallel 10 m diameter x 10 m high filter feed tanks. Low salinity water will be added to the tanks to maintain a density suitable for pressure filtration. The tanks each feed one pressure filter with an additional filter on standby. The filter cake produced transfers by conveyor to the plant concentrate stockpile prior to loading into trucks by front-end loader for transport to the rail siding.

17.11. Tailings

The tailings section includes a collection box that flows to feed the 30 m diameter tailings thickener. The thickened tails pass to the Tailings Storage Facility (TSF) for long-term storage.

The current development proposal for the Project is for open cut mining of the Moonshine and Moonshine North deposits at the rate of 9.68 Mtpa, yielding 3 dMtpa of magnetite ore over a period of 25 years.

Of the 6.68 dMtpa of waste material, 1.18 dMtpa will report as dry tailings from the LIMS process for trucking to the mine waste stockpiles and 5.5 dMtpa will be report as wet slurry tailings pumped to the TSF. Therefore, 137.5 dMt of tailings will need to be stored at the tailings storage facility during the mine's 25-year life.

17.12. Utilities

The plant will include a reagents facility for flotation and thickening related chemicals as well as raw, process, fire and potable water, plant air and instrument air facilities.

18 Project Infrastructure

18.1. Introduction

Several alternate options were identified for the Project's non-process infrastructure encompassing water supply, logistics, power, communications, tailings storage and other supporting facilities. For conciseness only the optimal solution is presented unless required to understand the ultimate decision.

The study has identified the preferred logistics option of hauling the product by private road to a rail loop on the existing Eastern Goldfields Railway (EGR), transport by rail to Esperance and then loading onto cape class vessels for export. Apart from the existing EGR and ship loader at the port, all infrastructure has been designed and costed by Engenium. Where applicable, the Company has elected to develop several facilities under a build-own-operate (BOO) model funded and managed by interested third parties. Such facilities include the laboratory and power station with pricing treated as an operating cost over the term of the proposed contract.

Product will be transported from the mine by road to a rail loop, east of Mt Walton station, approximately 93 km south of the Project

Figure **18-1**and then by rail to the Port of Esperance for export. The logistics chain includes road haulage along a private haul road utilising triple road-trains with side tip trailers, stockpiling at the rail siding, rail transport with rotary tippling wagons to the Port of Esperance, unloading by a rail car dumper, stockpiling in a covered shed, reclaim by FEL and loading onto ships via the number 3 berth ship loader. The following section describes the logistics path in more detail.

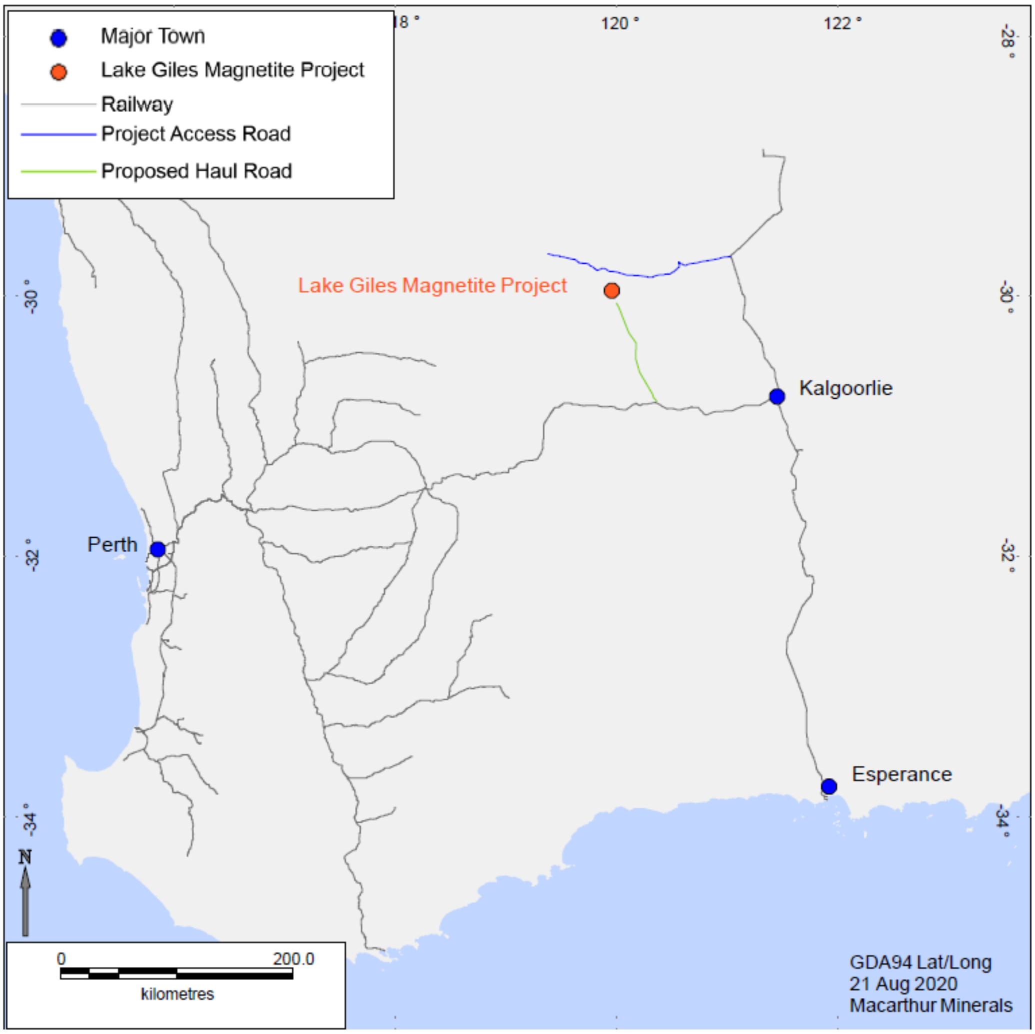


Figure 18-1: Lake Giles Iron Project Logistics Route

18.1.1. Road Haulage

Road haulage for the Project will be along a dedicated 93 km sealed haul road to be constructed from the mine product stockpile to the rail loop adjacent to the Perth-Kalgoorlie rail line near Mt Walton station.

The triple trailer side tipping road trains of 210 tonne payload will be loaded at the mine product stockpiles via Caterpillar 988H size or equivalent front-end loaders. A typical road train configuration is shown in Figure 18-2.

There may be potential to access the sealed haul road owned by another iron ore producer at the Carina mine, south of the project. Should this be possible, only 45 km of haul road would need to be constructed. Similarly, it may be possible to negotiate access to the Carina rail load-out which is currently not being used. These options are subject to negotiation with other parties.

Figure 18-2. Typical Side Tipper Road Train

18.1.2. Long-Haul Services

The responsibility for transporting product from the mine product stockpile to the rail siding will be contracted out to a specialist long haul contractor with the benefits of offsetting capital expenditure, better haulage efficiencies and reduced operational costs. The services include all vehicles, plant, equipment and offices necessary for the provision of the services. The contractor will also have responsibility for stockpile management and train loading at the rail siding.

The Long-Haul Contractor facilities include:

- administration office
- workshops
- refuelling
- services such as power, communications, IT, potable water and sewage
- turkeys nest for storage of water for dust suppression activities, and
- wash down bay including oil-water separator.

18.1.3. Rail Loading

A dedicated rail loop and product loading facility is planned south of the Project to manage the transfer of product from road transport to rail transport. An overview of the loop is shown in Figure 18-3 with loading facilities shown in Figure 18-4.

The proposed 4.6 km rail loop ties into the Eastern Goldfields Railway (EGR) at the 541 km chainage mark on the main line. The rail loop consists of a 1:12 turnout off the EGR mainline, single track (standard gauge) up until the 1:9 turnout for the 300 m radius balloon loop. It is proposed that trains will travel clockwise on the rail loop to be loaded with concentrate. The rail loop has been designed to accommodate a single consist (up to 1800 m long) at any one time however the single track could be duplicated to accommodate two consists if an expansion is required in the future.

Train loading will be conducted by two front end loaders loading at a rate of approximately 2000 tph. Lighting has been allowed for night operations to meet rail operational requirements as the train schedule will be confirmed closer to implementation. The two loaders will draw on three linear stockpiles 13.33 kt each (approximately three days storage). The stockpile length is 600 m long and designed to minimise the number of train repositioning movements required whilst also minimising the length of the rail loop and hence the upfront capital required.

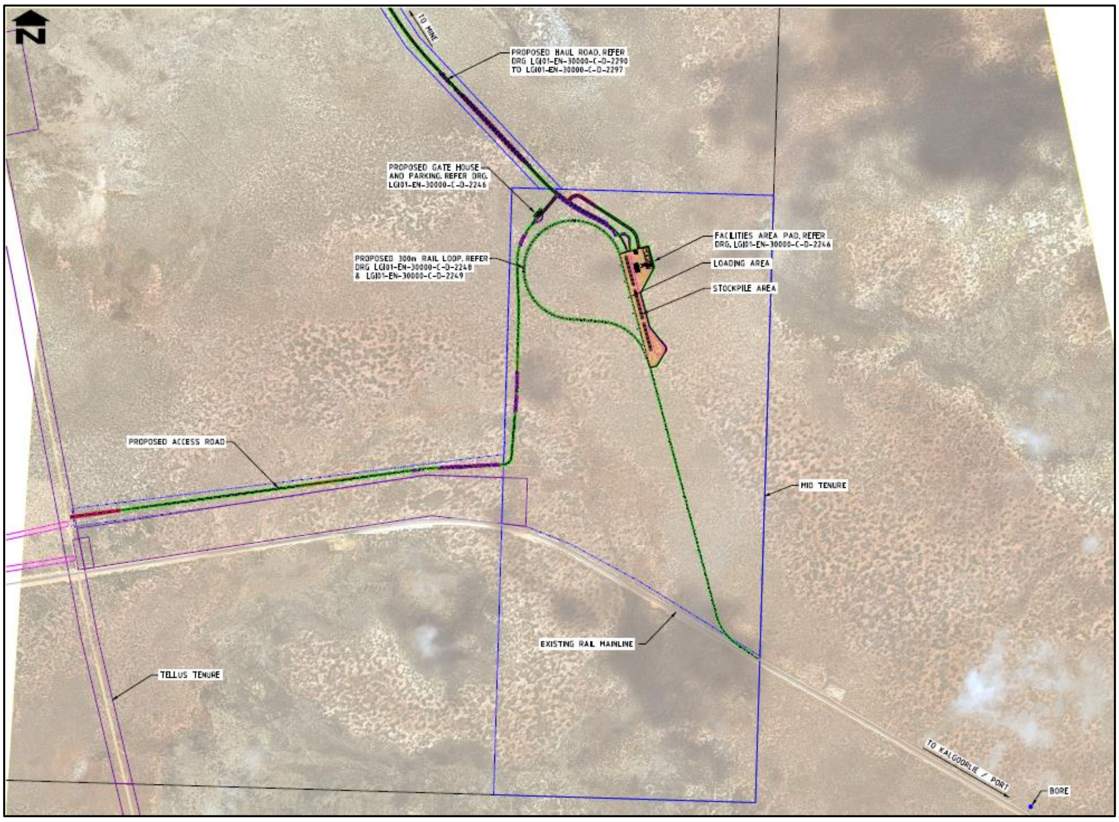


Figure 18-3: Rail Loop

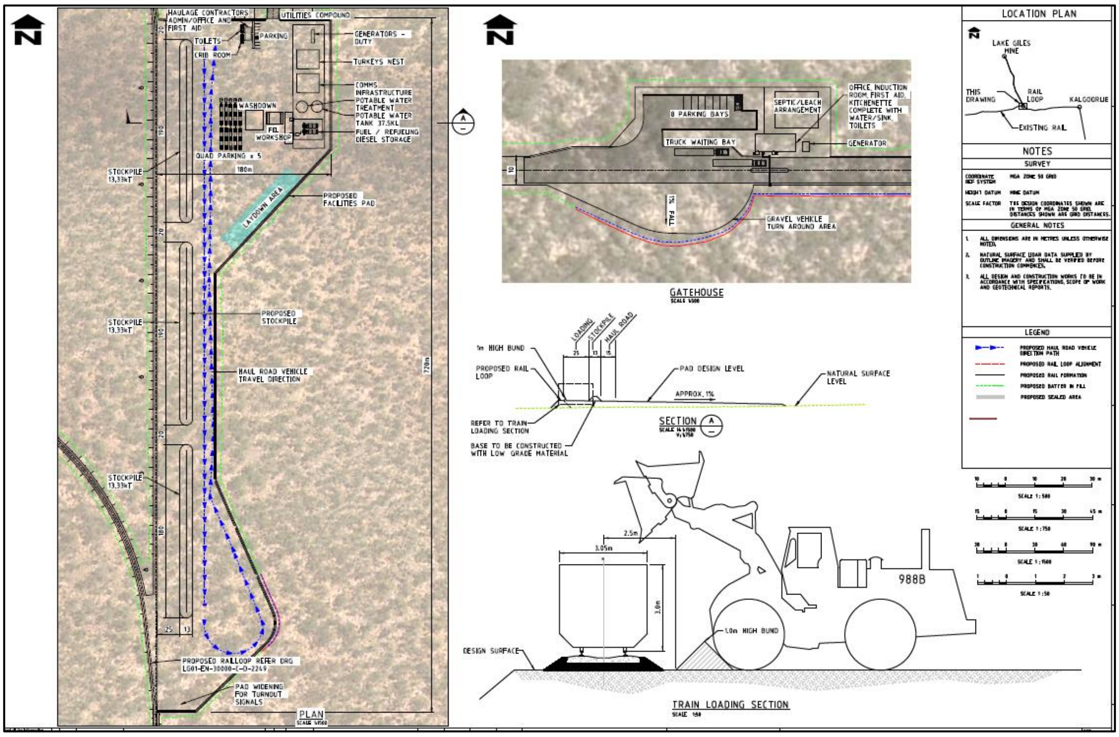


Figure 18-4: Rail Loading Facility

Although the front-end loaders will have load cells to weigh the amount of concentrate being loaded into the wagon, overload detection may be a requirement by Arc Infrastructure to ensure the maximum capacities are not exceeded on the rail line to Esperance. Space for the installation of wagon overload detection as depicted in Figure 18-5 below has been allowed for should it be required. In the event an overload is detected, excess material will be removed from the wagon with the use of a front-end loader.

Figure 18-5: Wagon Overload Detection

In consideration of dust emissions and loss of product over the 488 km journey to Esperance, the application of a polymer veneer is proposed. A single application can last the life of the journey to Esperance; therefore, a second application will not be required.

The polymer veneer will be applied to the concentrate in the rail wagons and applied with an overhead rail gantry similar to that shown in Figure 18-6.

Figure 18-6: Application of Veneer

18.1.4. Rail Logistics

A specialist consultant was engaged to evaluate the operational, technical and third-party pricing for the rail logistics component of the Project.

Below Rail

The rail line from the Mt Walton rail siding to Esperance is approximately 488 km of standard gauge rail suitable for bulk ore wagon transport.

The rail line from the siding to the Port of Esperance is managed by Arc Infrastructure (Arc) under a lease agreement with the Western Australia Government. Arc Infrastructure is owned by global asset management company, Brookfield Infrastructure Partners and operates the rail under an open access regime.

Enquires were made with Arc with regards to capacity along the proposed rail route. At the time of the enquiry, additional capacity on the Kalgoorlie to Esperance line was not available. Based on the production rates the rail network operator will be required to construct a series of passing loops. These upgrades require a lead time however there is sufficient time to construct these passing loops during the construction period of the mine facilities.

Indicative pricing for rail access has been provided by Arc inclusive of capital upgrades. Should capacity become available due to a decrease in rail paths by other operators, updated pricing will be obtained that should deliver a reduction in rail tariffs.

Above Rail

Proposals were sought from above rail operators to transport the magnetite concentrate from the rail loop to the car dumper at the Esperance port.

At the time of enquiry there were no suitable wagons available for the project and as such wagons will need to be purchased. Currently the most likely source for wagon manufacturing is China with the proposed wagon for the Project being a rotary wagon produced by CRRC as shown in Figure 18-7. This rotary wagon has a maximum payload of 75.5 tonnes.

Macarthur could purchase these rail wagons and provide them to an above rail operator under a “hook and pull” arrangement or the above rail operator can provide them under a “full service” arrangement.

The preferred solution for the Project is a “full service” arrangement. Under this arrangement the rail operator will be responsible for providing the rail consists (providing the locomotives and procurement of the wagons), rail operations and rail maintenance (inclusive of storing spare wagons). To meet the production rate for the project the rail operator will need to provide a maximum of seven services a week based on a train configuration of three locomotives and 126 wagons (9,387 T payload). This will require a purchase of 264 wagons which includes 12 spare wagons (5% of the fleet size).

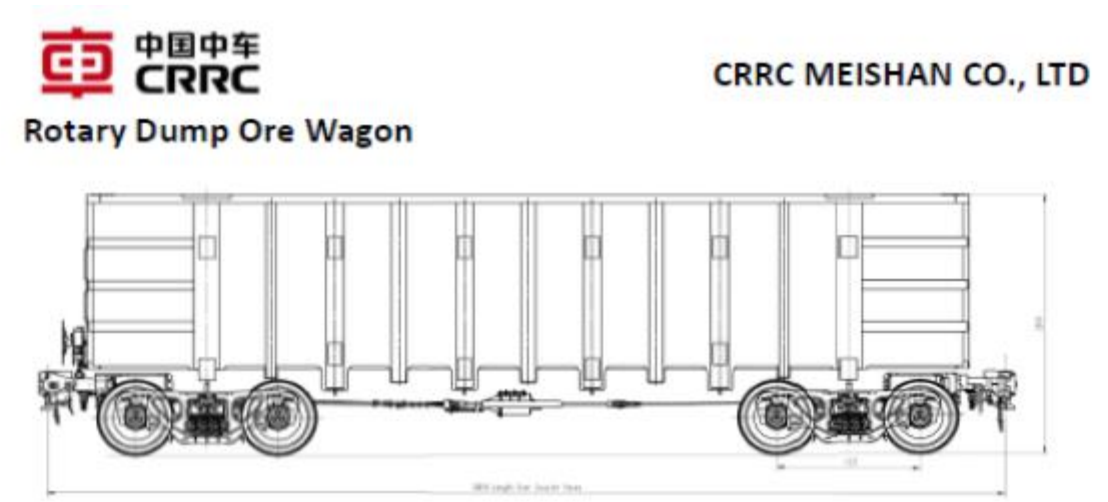


Figure 18-7: Proposed Wagon

18.1.5. Port

The Project is centrally located between a number of ports in southern Western Australia. The preferred port, with a direct connection to the existing Perth-Kalgoorlie railway, is the Port of Esperance operated by Southern Ports Authority (SPA). Refer to figure 18-8.

Berths (Red Numbers)	Storage Facilities (Yellow Numbers)
1. Berth No. 1 – Grains 2. Berth No. 2 – Mineral Concentrate, Fertiliser, Fuel 3. Berth No. 3 – Iron Ore	1. Shed 1 - Iron Ore 2. Shed 2 - Iron Ore 3. Shed 6 - Mineral Concentrate 4. Shed 7 - Mineral Concentrate 5. Shed 5 - Mineral Concentrate 6. CBH Operations 7. Summit Fertilisers 8. Gas Fired Power Station 9. Shed 3 - Iron Ore 10. Shed 4 - Spodumene 11. Shed 10 - Sulphur 12. Container Storage Area
General Infrastructure (Green Numbers)	
1. Rotary Car Dumper 2. Smith Street Level Crossing 3. Potential Shed Storage Area	

Figure 18-8: Port of Esperance Aerial View

With the completion of a $54 million port upgrade project in February 2002, the port became the deepest port in southern Australia, with Berth 3 capable of handling Cape class vessels up to 200,000 dwt, plus fully loaded Panamax class vessels up to 75,000 dwt. The design depth of Berths 1 and 2 is 14.6 m, Berth 3 and the inner channel are 19.1 m, the middle channel is 19.5 m and the outer channel is 19.9 m.

Berth 3 is currently utilised for all iron ore shipments. Existing iron ore export capacity through the Port of Esperance is 11.8 Mtpa, with export in the 2020/2021 financial year around 10 Mtpa. The port is also a major grain exporting hub and handles bulk imports such as fuel, sulphur and fertilisers.

To facilitate export from the Port of Esperance, new storage infrastructure is required to store the magnetite concentrate. Existing rail unloading facilities, expected to have adequate capacity, will be utilised to unload rotary car wagons and direct product to the storage shed. Reclaim of material from the storage shed will be managed by SPA contractors with ship loading via the existing outload circuit and Berth 3 ship loader.

18.1.6. Ship Loading

Iron ore loading rates, at Berth 3, of up to 4,500 tph are obtained by a travelling ship loader with an outreach suitable for vessel beams of up to 47 m. The berth is 230 m long with a depth alongside of 19 m and it can accommodate ships with a maximum LOA of 290 metres and a draft of 17.8 m. Figure 18-9 shows an iron ore vessel moored to the Berth 3. The design export vessel capacity is 175, 000 dwt.

Maximum ship loader capacity is expected to be 3500 tph after consideration of down time in operations such as ship hatch change requirements. There will be capability for direct ship loading from the car dumper.

Figure 18-9: Iron Ore Loading at Berth 3

18.1.7. Site Infrastructure

The Project will comprise a fully serviced remote area mining and processing hub that will be supported by a fly in fly out (FIFO) work force supplemented by regionally located personnel.

18.1.8. Power

Power Generation and Reticulation

40 MW of power supply will be required for the Project inclusive of the magnetite process plant and supporting non-process infrastructure. Macarthur engaged power supply analyst firm, Veckta to review and determine the most cost-effective power solution for the main power plant located at the mine. The results of the study by Veckta showed the lowest cost of electricity was achieved by significant renewables penetration.

The lowest power cost involved a combination of wind, solar, battery storage and natural gas reciprocating engines however due to land tenure constraints and capital cost of wind turbines, the preferred option for the project is a hybrid solution of solar, battery storage and natural gas reciprocating engines.

Enquiries were made to vendors to determine the cost of power, optimal level of renewable penetration and the preferred contracting strategy. Three proposals were received, and the recommended solution was a BOO contracting strategy with installed capacity of 48 MW (24 x 2 MW – including redundancy) of natural gas reciprocating engines, 40 MW Battery Energy Storage System (BESS) and 60 MW of solar panels resulting in a renewable penetration of 33%. The location of the solar farm in relation to the plant is shown in Figure 18-10 and the source of fuel for the main power plant (trucked LNG) is discussed in more detail in Section 18.1.9.

The proposed hybrid power solution will save approximately 107,684 tonnes of CO_2 emissions per annum when compared to 100% diesel power generation. An added benefit of the hybrid power solution is the opportunity to create large-scale generation certificates (LGCs) based on the amount of electricity generated from the power plants solar farm. These LGCs can be sold on the market to reduce the effective cost of power for the Project.

The accommodation village, aerodrome, tailings storage facility, ANFO facility, rail loop and bore fields will be powered by diesel generators. Permanent power by overhead transmission from the main power station was reviewed for these facilities however this option was eliminated due to the distance from power station and hence capital cost.

The power options for the village and other remote infrastructure will be reviewed in the subsequent value improvement works to consider localised renewable options, or use of similar fuel as the main power facility.

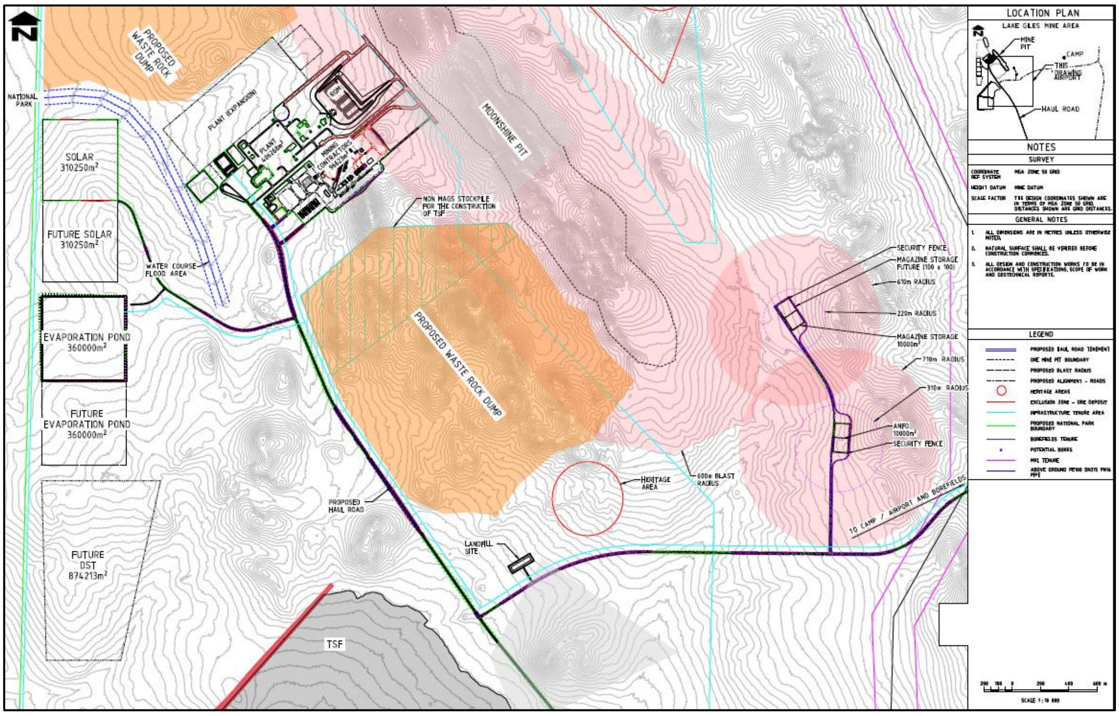


Figure 18-10: Solar Farm

18.1.9. Fuel Facilities

The Lake Giles Project requires the storage of two types of fuel. Liquified natural gas (LNG) for the power plant and diesel for refuelling of heavy mobile equipment and light vehicles.

LNG fuel facility

The LNG fuel facility will provide the fuel required to power the main power plant (See Figure 18-11). LNG was chosen as the preferred fuel for power generation as it has lower greenhouse gas emissions compared to alternative fuel sources. The LNG facility consists of 6 x 350 kL (780 tonnes) storage tanks to provide approximately 7 days of on-site storage. LNG fuel deliveries will be conducted by LNG road tankers with a payload of 30 tonnes. It is expected that the power plant will have a consumption of 114 tonnes per day therefore approximately 4 deliveries will be required per day from Perth.

Figure 18-11: Typical LNG Fuel Facility and LNG Road Tanker

A gas pipeline was considered but was deemed to not be the most feasible option for the requirements of the Project due to the distance from the pipeline, tenure constraints and a higher capital cost.

Diesel fuel facility

Two diesel fuel facilities are required for the project. One facility will be located at the mine site with the second facility to be located at the rail loop.

The facility at the mine site has a capacity of 550 kL to service the mining fleet, light vehicles, mobile plant and fuel trucks utilised to refuel diesel generators at the bore fields and tailings storage facility. To meet the requirements of the project diesel fuel will be delivered daily with a double road train (78,000 L). This allows for 7-9 days of storage in the event of any road closures due to inclement weather. The 550 kL fuel facility consists of 5 x 110 kL self-bunded tanks (similar to the tank shown in Figure 18-12) that are interconnected to provide the required capacity.

Two high flow (400 litres per minute) bowsers will be available for the mining fleet and a single bowser (80 litres per minute) will be provided for light vehicles.

The facility at the rail loop is a self-bunded packaged facility Figure 18-12 with a capacity of 100kl to service the road trains carrying concentrate from the mine and the front-end loaders responsible for loading product onto the trains. This fuel facility will have automatic tank gauging and a fuel management system to co-ordinate fuel deliveries with the fuel supplier and allow for the tracking / management of fuel. A high flow bowser will be available for road trains and FELs alongside a single bowser for light vehicles.

The proposed fuel facility is supplied complete with fuel management equipment and systems that allow tracking and management of fuel consumption per consumer or groups of consumers. Automatic electronic re-ordering from the fuel supplier is possible, and the system facilitates contracts management during construction in the event that fuel is free issued to construction contractors.

Figure 18-12: Self-Bunded 100 kL Tank

18.1.10. Water Supply

The total annual water requirement for the Project is estimated to be 4 Gl, supporting a mineral processing facility operating at a nominal 3.0 Mtpa run rate along with all associated non-process infrastructure (excluding the port) and dust suppression.

To develop an understanding of both the hydrogeology and water quality within the area, a specialist hydrogeological consultant, Rockwater was engaged to undertake an initial desktop assessment. This was followed by an airborne electromagnetic geophysical survey to further assess groundwater availability and quality within the identified paleodrainage systems adjacent to the Project.

The Rockwater studies concluded that, subject to completing a comprehensive drilling and testing program, water supplies for the Project should be available from the paleodrainage systems in the Project area.

A bore field will be constructed to source water to supply the Project's construction and subsequent process and potable water requirements. Project estimates indicate 26 fully equipped bores will be required to meet the Project water demand of 466 kL/h, based on nominal flow rates of 5 L/sec. Macarthur is currently progressing an application for tenure across this area to complete the required drilling activities.

The proposed bore field, pipeline alignments and key storage locations are depicted Figure 18-12. The bore field supply line will feed the bore water tanks from the tanks a transfer pump station will pump the bore water to the raw water pond. There are three bore water tanks with a residence time of 12 hours and a nominal capacity of 2000 m^3. The aboveground piping will be white co-extruded high-density polyethylene with a nominal diameter of 315 mm.

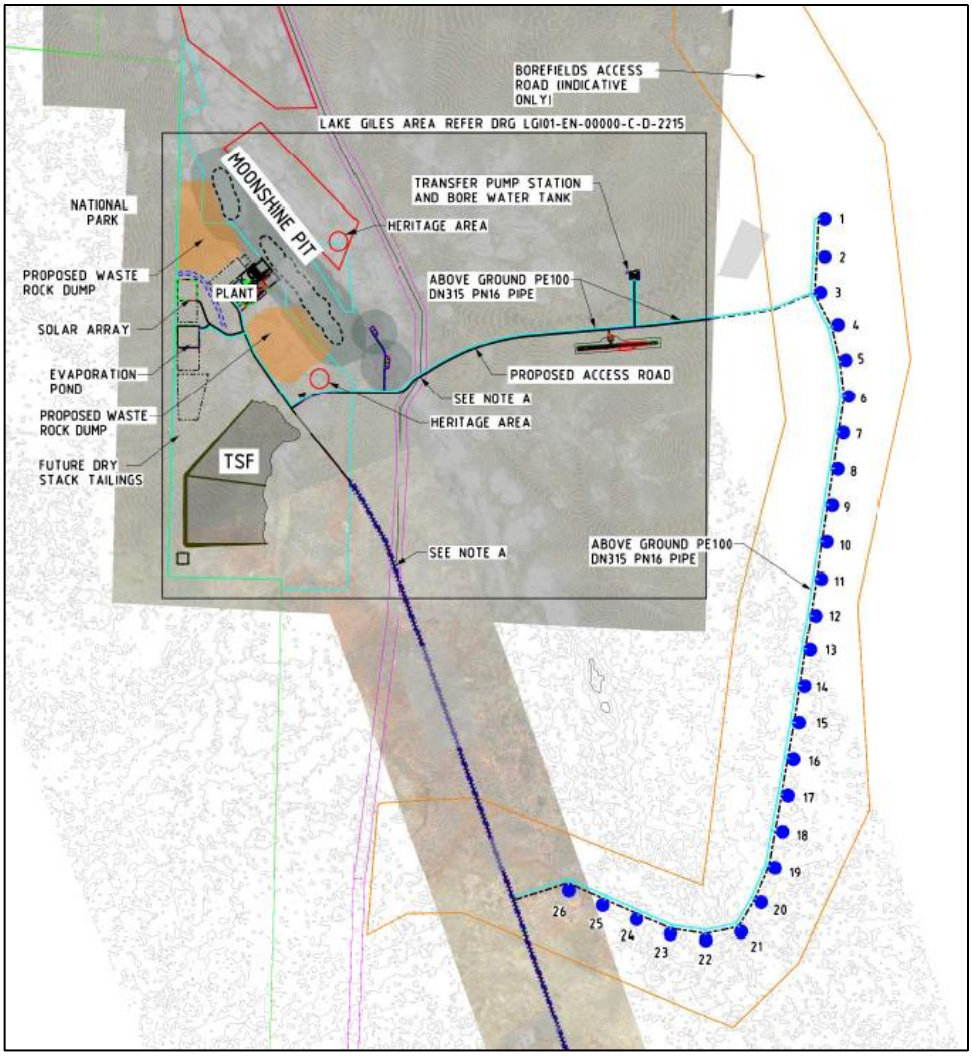


Figure 18-13: Bore field Layout

For provision of raw water during the construction phase, initial bores will be constructed, pumping to a series of turkeys nests along the access road. A desalination plant would be required to treat the raw water and provide some potable water for construction use, for example in concrete batching and upper pavement construction.

Hydrogeology

Project water supplies will require development of groundwater sources adjacent to the site. Numerous studies have been conducted in the area however little information is available on groundwater occurrences encountered via drilling.

According to Rockwater (2010), a paleochannel aquifer was inferred to exist beneath the Rebecca paleodrainage system west of the Project area. The aquifer was considered a potential source of large supplies of groundwater, with potential bore yields of 3.4 L/s, at 5,000 to 20,000 mg/L TDS.

Rockwater was unable to identify any large quantities of low salinity groundwater within the project area and recommended further investigations to support the Project water requirements.

Subsequently, Macarthur commissioned Rockwater (2020) to review palaeovalley water supply options targeting tertiary palaeovalley aquifers, and utilising outcomes of a gravity survey (Haines, 2014). Utilising a digital elevation model, Rockwater considered that the headwaters of the Lake Ballard palaeovalley system comprising three main northerly draining palaeovalley aquifers, Scorpion East, Scorpion Central and Scorpion West, would present the best opportunity for the Project. Airborne electromagnetic surveys along with air core exploration drilling along transects of the aquifer were recommended to better assess the locations and quality (salinity) of the three Scorpion palaeovalley aquifers.

Due to a lack of tenure, Macarthur commissioned Rockwater (2021) to map the palaeovalley aquifers utilising TEMPEST AEM, an airborne electromagnetic geophysical survey.

The findings indicated two major northerly draining paleovalleys to the east of the Project deposit site. Rockwater advised that drilling along the Evanston-Menzies Rd into the Scorpion East palaeovalley returned salinity figures of some 100,000 mg/L TDS.

The Scorpion West palaeovalley was considered as the best opportunity to supply the project water requirements. Accordingly, Rockwater provisionally recommended 38 production bores at 1.5km spacings, subject to installation of several transects of air-core holes drilled across the palaeovalley to assess width, depth, permeability and salinity of the aquifer.

Due to tenure limitations, it was also recommended to undertake a targeted passive seismic survey to define profiles of the bedrock depth at several points along the aquifer.

The Rockwater studies indicate that, subject to completing a comprehensive drilling and testing program, water supplies for the Project should be available from the paleodrainage systems in the Project area.

Further, as the Lake Giles deposit lies within the Goldfields Groundwater Area, a DOW 26D licence is required to construct bores, and a 5C licence is needed to take water. Water bore drilling contractors are required to hold a water well drillers licence to construct bores. Approval for a Programme of Works (POW) is needed from the Department of Industry and Resources (DOIR) to clear areas for drilling.

Water Requirements

A water balance has been determined for the processing plant, supporting non-process infrastructure, mining contractor requirements and at the railhead (see Figure 18-14).

The categories of water described below are:

- raw water drawn from the bore fields
- treated water (reverse osmosis) for potable water supply
- process water which is recycled within the plant and topped up by the raw water
- treated water (nanofiltration) for process washing requirements
- brine from the water treatment plants, and
- separate railhead bore and potable water supply.

STREAM		61	63	64	65
Parameter	Unit	Southern Central Borefield Supply	Southern Borefield Supply	Typical bore	Borefield Supply
PFD		2113	2113	2113	2113/2107
Liquid density	t / cu.m	1.035	1.035	1.035	1.035
% solids	% w/w	0	0	0	0
Volume of liquid	cu.m / hr	234	234	18	466
Number bores	no.	13	13		26

STREAM		66	67	68	69	70	71	72	73	74	75	76	77	78	79	80
Parameter	Unit	Nanofiltration Plant Brine Discharge	Raw Water Pond Pump Supply	Seal Water	Nanofiltration Plant Permeate	Permeate RO Potable Water	Village Potable Supply	Airport Potable Supply	Fixed Plant Workshop Potable Supply	Process Building Potable Supply	Shared Services Potable Supply	Main Administration Potable Supply	Mining Contractor Potable Supply	Haulage Contractor Potable Supply	Eye Wash Supply	Powerstation
PFD		2107	2107	2107	2107	2107/2110	2110	2110	2110	2110	2110	2110	2110	2110	2110	2110
			Note 9	Note 12			Note 2	Note 3	Note 4	Note 4	Note 5	Note 4	Note 3	Note 6		Note 7
Liquid density	t / cu.m	1.053	1.035	1.008	1.008	1.0006	1.0006	1.0006	1.0006	1.0006	1.0006	1.0006	1.0006	1.0006	1.0006	1.0006
% solids	% w/w	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Volume of liquid	cu.m / hr	83	390	15	153	43	3.33	0.16	0.016	0.016	0.00	0.016	0.16	0.08	1.50	2.50

STREAM		81	82	83	84	85	86	87	88	55	56
Parameter	Unit	Nanofiltration Plant Feed	RO Plant Feed	Mine Dust Suppression	Cloth Flushing	Filter Manifold Washing	RO Plant Brine Discharge	Mining Contractor Supply	Cooling Water Top Up	Flocculant Dosing	Flocculant Dilution
PFD		2107	2107	2107	2107/2105	2107/2105	2107	2107	2107	2107/2106	2107/2106
		Note 10	Note 11	Note 8				Note 15	Note 14		
Liquid density	t / cu.m	1.035	1.035	1.053	1.0006	1.033	1.053	1.053	1.008	1.008	1.033
% solids	% w/w	0	0	0	0	0	0	0	0	0	0
Volume of liquid	cu.m / hr	236	107	20	35	20	64	60	0	4	14

STREAM		90	91	92	93	95	96	97	98
Parameter	Unit	Railhead Bore Supply	Railhead Raw Water Pump Discharge	Railhead Dust Suppression Standpipe	Railhead Vehicle Washdown	Road Tanker Offloading	Railhead Potable Water Pump Discharge	Railhead Potable Water Pump Discharge	Railhead Potable Water Pump Discharge
PFD		2111	2111	2111	2111	2111	2111	2111	2111
Liquid density	t / cu.m	1.005	1.005	1.005	1.005	1.005	1.005	1.005	1.005
% solids	% w/w	0	0	0	0	0	0	0	0
Volume of liquid	cu.m / hr	12	25	20	5	20	0.10	0.05	0.05

Figure 18-14: Water Balance

Raw Water

Raw water is drawn from the paleochannels bore fields as discussed in 18.1.10. Raw water is stored in a pond with a residence time of 48 hours, or a nominal volume of 22,500 m^3. Raw water pond transfer pumps feed a raw water tank with a residence time of 12 hours, a volume of approximately 5,600 m^3.

The raw water pond provides an overflow top up to the process water pond. Raw water is also pumped from the raw water tank to the nanofiltration (NF) and reverse osmosis (RO) plants and is reticulated to the mine for plant dust suppression.

Potable Water

To support the Project potable water requirements, water is pumped from the raw water pond to a RO plant with a maximum permeate total dissolved solids (TDS) of 600 mg/L. This potable water is reticulated for use in the mine area buildings and workshop, for pressure filter cloth flushing and is also pumped to the village and the airport. The potable water at both the village and the airport requires additional disinfection with sodium hypochlorite due to their distance from the mine in line with the Australian Drinking Water Guidelines (ADWG).

Potable water requirements have been modelled on the following demands:

- Aerodrome – 20 L / person / day (flight days only)
- Accommodation village – 200 L / person / day; and
- Mine – 20 L / person / day.

Potable water tanks less than 50, 000 L in size will be polyethylene. Potable water tanks greater than 50, 000 L in size are to be steel bolted and lined.

Process Water

Process water is to be stored in a pond with a minimum of two hours residence time or 7500 m^3. To minimise entrained particles, the process water overflows and is drawn from a needle tank with the same required capacity. The pond and tank are filled by the bore fields supply via the raw water pond overflow in a first fill or start up scenario.

Along with an overflow top up from the raw water pond, the process water incorporates recycled water flows. The majority of the water that enters the process is recycled to the process water pond in the form of thickener overflow, decant return from the tailings storage facility and concentrate filtrate.

The main process water users are the wet screening and milling (ball mill and Vertimill) circuits. Process water is also used for reverse flotation, pressure filter manifold washing and flocculant dilution.

Nanofiltration Water

Treated water is required for a final wash at the polisher LIMS in order to achieve an acceptable alkaline salt level in the concentrate.

A nanofiltration (NF) plant, fed by the raw water tank pumps, is utilised to desalinate the water to a permeate TDS of 8000 mg/L.

The NF water is also used for flocculant dosing, seal water and equipment cooling. The initial supply of fire water for storage and top up as required would also be sourced from the NF plant.

18.1.11. Brine

The brine from the RO and NF plants would be pumped to a brine evaporation pond sized to allow storage of the remaining salt over LOM. Brine storage requirements and the Project raw water use will be decreased by using the brine for mine site road dust suppression. The brine will also be reticulated to the mining contractor for mining contractor supply and as dust suppression in the mine. A tie in for alternate supply from the bore fields transfer pumps to the mining contractor is also provided.

18.1.12. Railhead

Railhead raw water supply will be by a single dedicated bore at the railhead. The raw water will be pumped from the bore to be stored in a turkeys nest. Raw water pumps will draw from the storage and reticulate the water to the dust suppression standpipe and vehicle washdown.

Potable water requirements at the railhead will be from a potable water tank with chlorination unit, supplied by a potable water tanker. This potable water will be reticulated to the gatehouse and haulage contractor.

18.1.13. Communications

During the construction phase, it is anticipated that trailer mounted VSAT broadband units would be utilised to establish voice and data communications via a satellite network. This solution is inherently flexible and can be adapted to the changing requirements during initial site establishment and construction.

During operations a conventional VHF radio system would allow communications coverage for the entire site. These VHF radio systems will be mounted on five of 20 m communications towers located at the mine site, the tailings storage facility, the village and two along the haul road leading to the rail loop (Figure 18-5).

Up to ten channels will be allowed for operations which allow channels including but not limited to an emergency channel, mine operations, contractor's channel and process plant. All vehicles shall have at least one (1) hard wired VHF radio.

The same communications towers will provide internet services for the mine site and the village via a microwave link from a communications carrier (such as Telstra). The overall proposed configuration of the communications is shown in Figure 18-16. Internet access will be provided to site for mine operations and contractors which will allow communications to head office locations such as remote monitoring of plant and voice services. This internet access will also be used to provide live video feed from the process plant and rail siding back to the Macarthur head office.

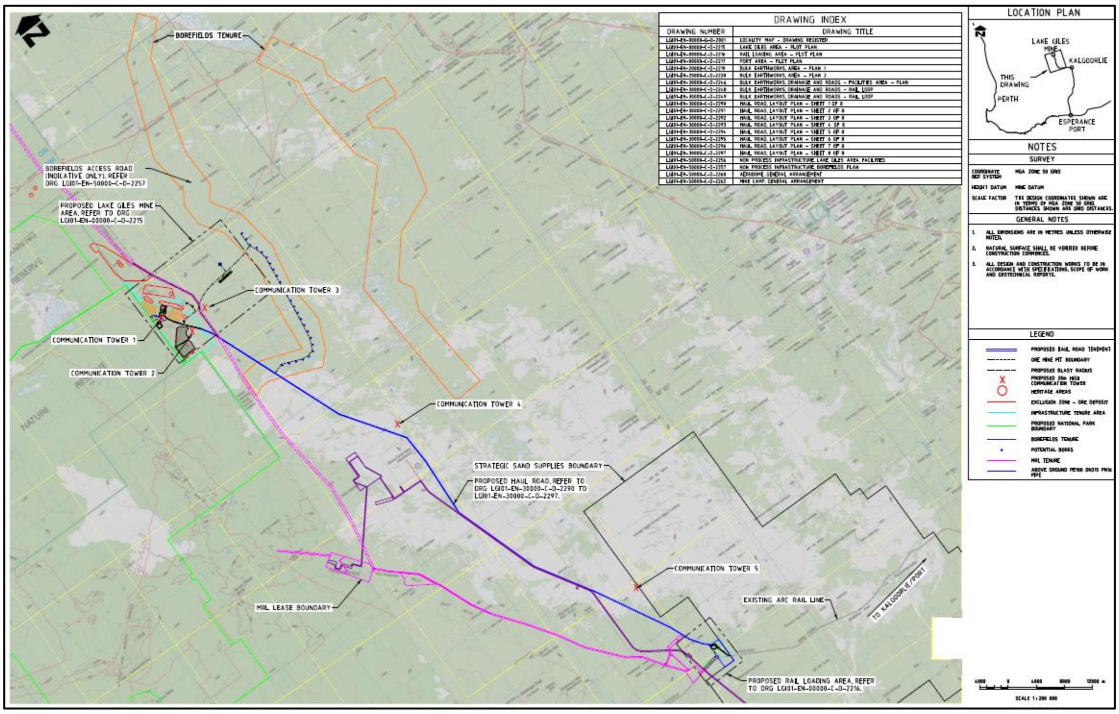


Figure 18-15: Communications Tower Locations

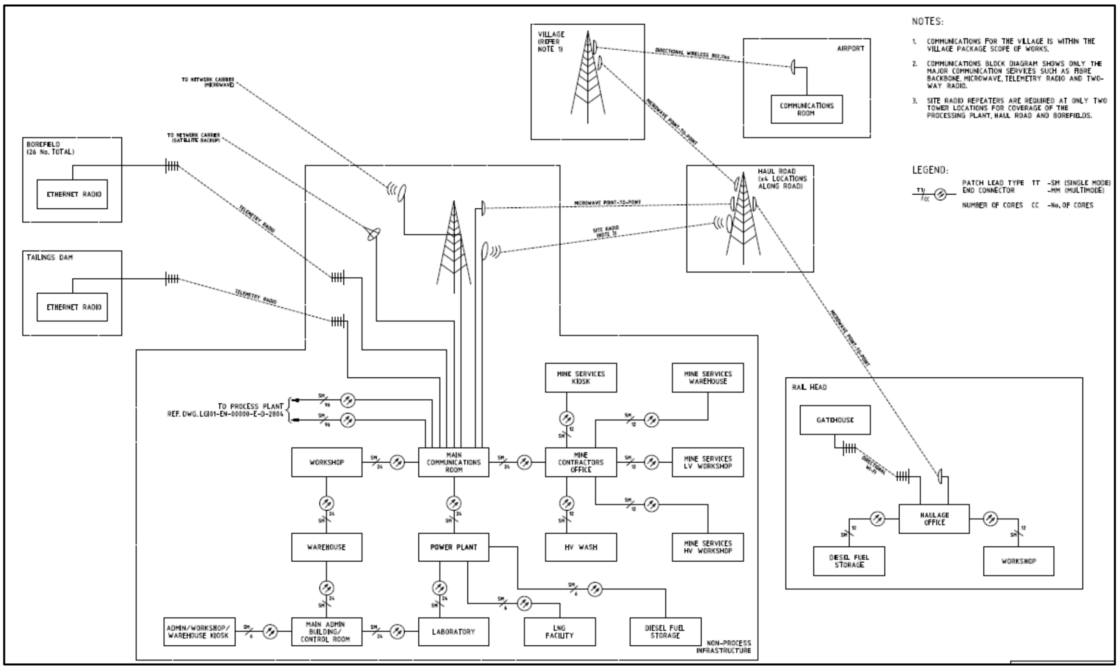


Figure 18-16: Communications Block Diagram

The camp will have its own dedicated communications for internet to allow entertainment services for residents. Internet will be distributed throughout the camp facilities by a combination of wired and wireless connections.

The airport will require internet access for airport operations (checking in passengers and airline operations). The weather station at the airport will also require internet access for communicating weather information to the Bureau of Meteorology. In addition to internet services, an aerodrome frequency response unit (AFRU) and pilot actuated lighting control (PALC) is required for pilots to communicate with airport operations and to activate the precision approach path indicator (PAPI).

The bore fields and the tailings storage facility will be connected via ethernet radio for remote monitoring and control purposes. It is envisaged the flowrates for the bores at the bore fields, monitoring bores at the TSF and the stability of the TSF will be monitored.

18.1.14. Access Roads and Plant Area Roads

The project can be accessed by heading approximately 130 km west from Kalgoorlie via the Great Eastern Highway, north approximately 45 km along the unsealed access road and then east after the Eastern Goldfields Railway (EGR) level crossing. The access road adjacent to the EGR follows the track towards Kalgoorlie before heading north towards the gatehouse. Once the mine is operational the gatehouse will be the only access into the Project. The product haul road intersects the access road immediately after the gatehouse and is utilised to access the mine / village and airport to the north and the rail loadout area to the south. The main access road is shown in Figure 18-17.

The product haul road is the main access to the process plant area. The plant access roads have been laid out to segregate heavy vehicles and light vehicles. As such the first intersection will be utilised by heavy vehicles to access the mining contractor’s area. The second intersection will be utilised for deliveries (warehouse, LNG fuel and diesel fuel) and will be the main access for operations staff (light vehicle parking and a bus drop-off bay). Lastly, the third intersection will be utilised by mobile equipment and light vehicles during shuts. Plant access roads and access to the haul road from the plant are shown in Figure 18-8.

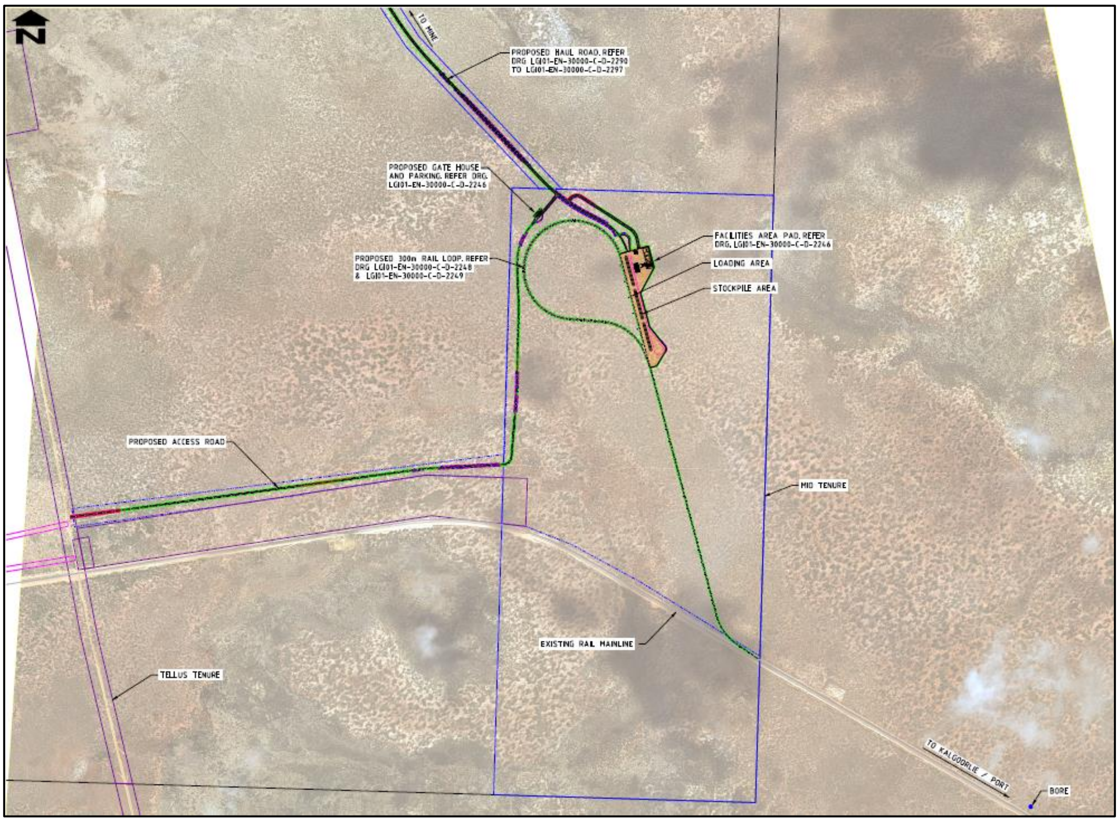


Figure 18-17: Main Access to Site

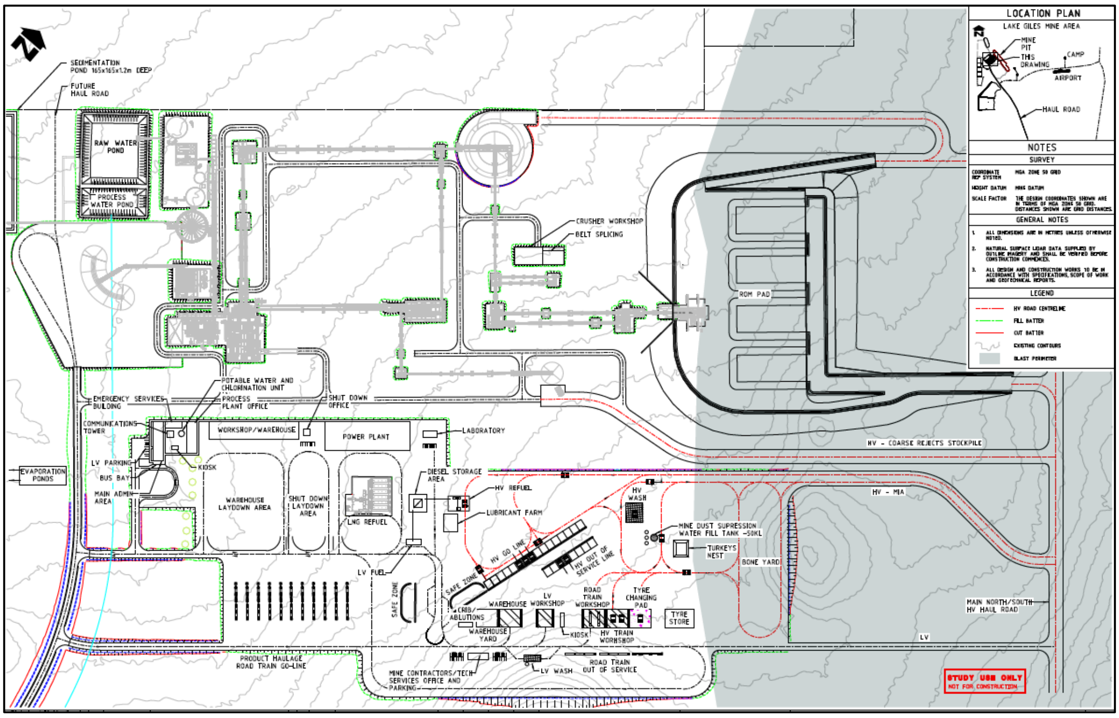


Figure 18-18: Plant Roads

18.1.15. Mine Administration Facilities

The main Mine Operations Centre will include the following:

- Main Administration Area:
  - Administration office including training room
  - First aid and fire building complete with vehicle parking for ambulance and fire truck. Building to include room for ERT equipment
  - Bus and LV parking; and
  - Two sea containers for geology equipment storage.
  - Mining Contractor Area:
  - offices
  - crib room and ablutions
  - heavy vehicle warehouse
  - oil storage building area
  - heavy vehicle workshop (dome shelters) and apron (four covered and two uncovered)
  - boilermaker's bay
  - heavy vehicle tyre change and laydown
  - bulk lubricant storage
  - gas bottle storage
  - heavy vehicle washdown, and
  - training room.
- Haulage contractor area:
  - haulage loader and truck workshop, and
  - haulage truck parking area.
- Fixed Plant area:
  - belt splicing building
  - workshop
  - gas bottle storage shed
  - battery storage shed
  - warehouse
  - lubricant dispensing system (20ft sea container)
  - day maintenance workshop for crusher area (shed), and
  - shutdown office.
- Process Buildings:
  - laboratory building, and
  - process plant building including the control room.

- Permanent laydown area
- Mining contractor facilities:
  - ANFO storage, and
  - magazine storage.

The Mine Operations Centre would largely comprise of modular buildings and dome shelters. Some of the larger facilities such as warehouses and workshops will be required to be constructed in-situ.

18.1.16. Accommodation

Due to the location of the Lake Giles project, accommodation is required to support the construction and operation of the mine. It is expected that the majority of workforce required from the project will be sourced from Perth with only a small percentage of the workforce commuting from Kalgoorlie. It is proposed to construct the camp approximately 10km east of the mine. It is proposed that the camp will be a “design and construct” package to be performed by a specialist construction contractor with the operation of the village to be performed by a specialist camp operator.

It is estimated that a 720-room camp would be the peak size required to support both the construction and operation of the Project. Initially, the camp would accommodate the mining contractor for pre-strip operations and the mine construction contractors for a period of approximately one year.

Based on availability at the time of construction of the village, it is envisaged that 280 rooms will be leased for the duration of the construction with the remaining 440 rooms owned by Macarthur.

Whilst the village is being constructed, temporary accommodation could be sourced in Menzies, by expansion of the existing exploration village 30 km north of the Project, and/or by hire of a camp at a neighbouring operation.

Once the construction of the mine is complete, mine operations will ramp up and the camp will be downsized to a capacity of 440 rooms.

The 280 rooms removed from the village will have services such as power, water and sewerage disconnected with allowance for reconnection in case there is a requirement for future expansion of the mine. It is estimated that 440 rooms provides sufficient capacity for mine operations, village operations staff and shutdown contractors.

The 720-room camp includes the following facilities with a layout as shown in Figure 18-19:

- wet mess / recreation area
- dry mess and kitchen
- administration building
- village laundry and linen store
- laundry buildings
- camp accommodation buildings (four ensuite units per building)
- camp maintenance building
- gym
- potable water treatment plant and storage
- fire water
- rubbish service centre – food waste/recycled/general
- waste oil receptacle

- wastewater treatment
- bus shelter and parking
- car parking
- outdoor multi-court (tennis / basketball)
- outdoor pool, and
- communications and village entertainment room.

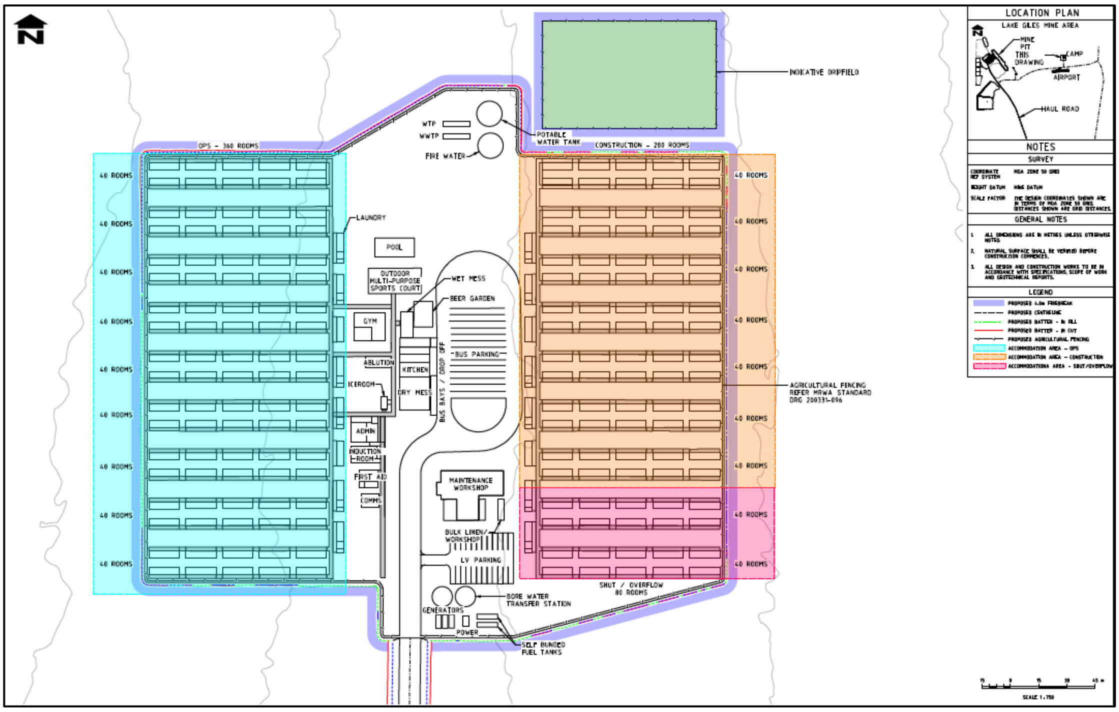


Figure 18-19: Proposed Camp Layout

18.1.17. Airport

With most of the workforce for construction and operations expected to be sourced from Perth (only a small percentage of the workforce is expected to commute from Kalgoorlie) it is proposed to construct an airport to reduce the commute times between the site and the closest public airport (Kalgoorlie-Boulder Airport).

The proposed airport is located approximately 2km south of the camp and was chosen as the preferred site due to geotechnical and hydrological conditions, topography, proximity to the camp / mine.

Based on the expected number of personnel required for construction and operations the design aircraft for the airport is a Fokker 100 (F100) which has a capacity of 100 seats. It is expected that three return flights will be required during construction and two return flights will be required during operations. Additional services may be required for shut contractors as required.

Figure 18-20: Fokker 100 (Virgin Australia Regional Airlines)

The Fokker 100 is an aircraft commonly used in the mining industry in Western Australia with most airline operators such as QantasLink, Virgin Australia Regional Airlines and Alliance Airlines utilising them in their fleet. The F100 however is an ageing aircraft and airline operators are commencing plans for their replacement in the 100 to 125 seat range such as the Embraer E190 (100 seats). The airport will be designed as a Code 3C compliant airport (1900 m long and 30 m wide runway) to meet the requirements of the F100 alongside other future Code 3C aircraft such as the Embraer E190 and the Boeing 717 (125 seats).

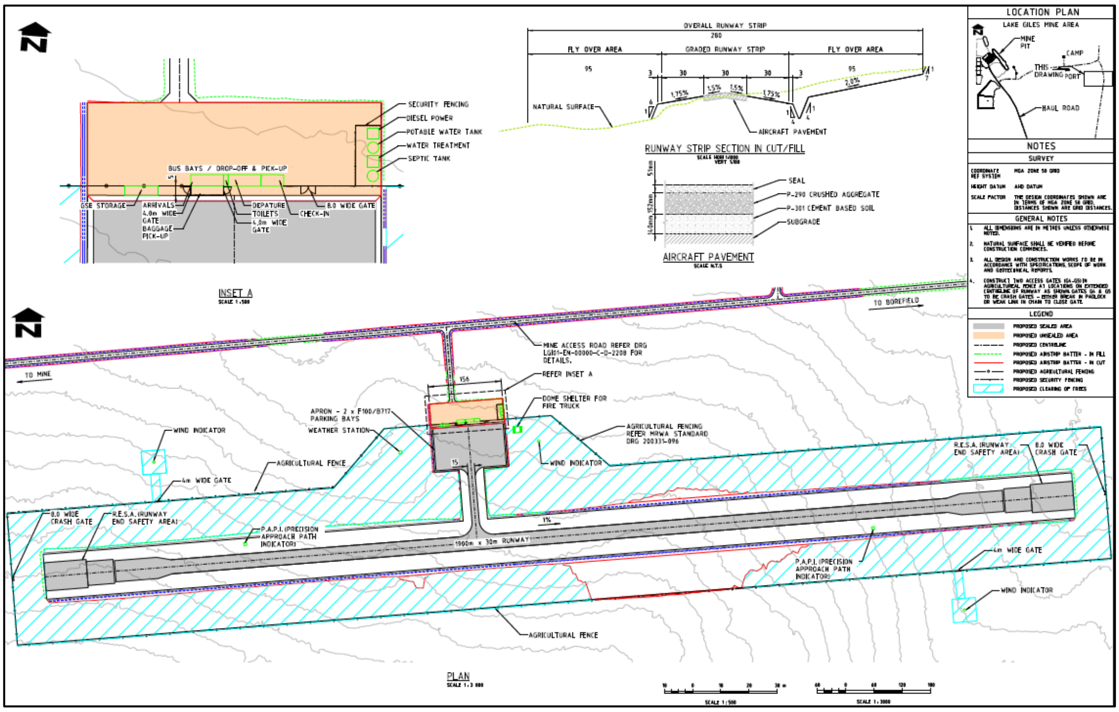


Figure 18-21: Proposed Airport

18.1.18. Tailings Storage

The current development proposal for the Project is for open cut mining of the Moonshine and Moonshine North deposits at the rate of 9.68 Mtpa, yielding 3 Mtpa (dry tonnes; dMtpa) of magnetite ore over a period of 25 years.

Of the 6.68 Mtpa of waste material, a total of 1.18 Mtpa will report as dry tailings from the low intensity magnetic separators (LIMS) process to be trucked to the mine waste stockpiles and 5.5 dMtpa will be report as wet slurry tailings pumped to the tailings storage facility (TSF). Therefore, a total of 137.5 Mt of tailings (dry) will need to be stored at the tailings storage facility during the mines 25-year life.

Previous studies identified a suitable site for the tailings storage facility to minimise the volume of embankments by utilising a ridge as containment, maximise stored capacity by selecting a relatively uniform area, ensure seepage / runoff is contained within a single catchment and to ensure runoff from the catchment upstream can be diverted around the tailings storage facility. The location and size of the TSF is shown below in Figure 18-2.

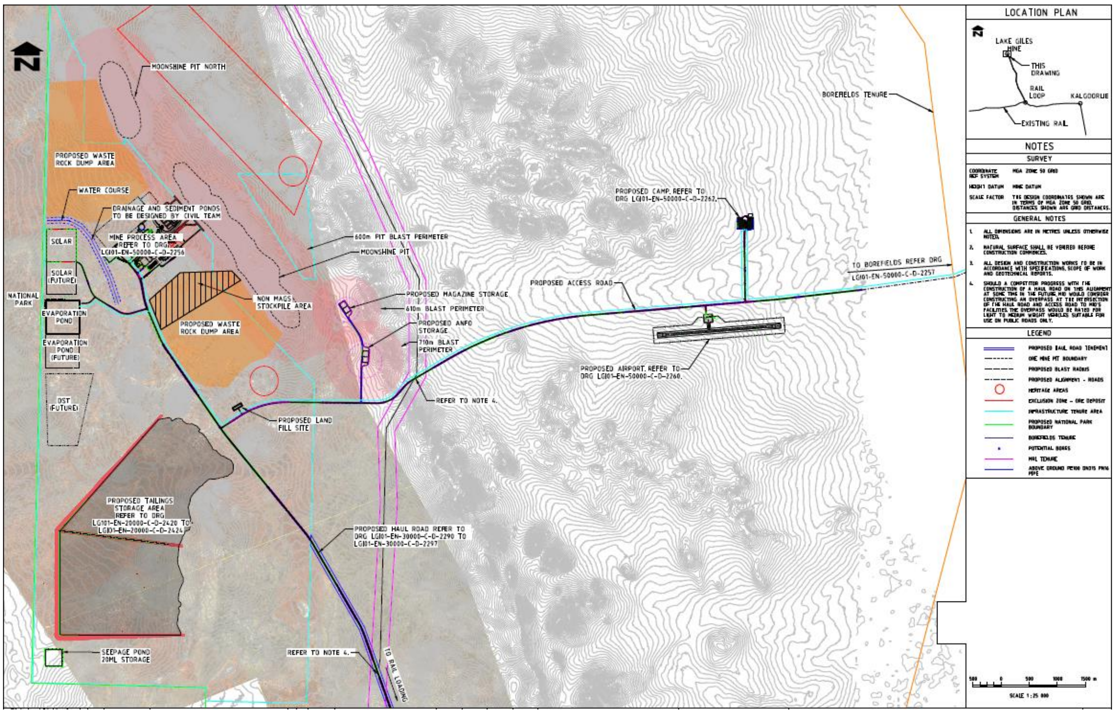


Figure 18-22: Location of the Tailings Storage Facility

The FS proceeded with the development of a design utilising the centreline construction method. The starter embankment for the tailings storage facility will be constructed to an elevation of 447 m AHD (See Figure 18-23). In the first years of operations a pontoon pump will be required for the decant water return to the process water dam until a tailings beach can be established. Once a tailings beach has been established, the pontoon will no longer be required as the decant tower can be utilised to pump water back to the process water pond.

The starter embankment will have sufficient capacity to store tailings until the third year of mining operations at such stage the embankment will need to be raised 5 m to an elevation of 452 m AHD. In the eighth year of operations the embankment will be raised to an elevation of 457m AHD and the final raise required for the life of the mine will be required in the fifteenth year to an elevation of 462 m AHD.

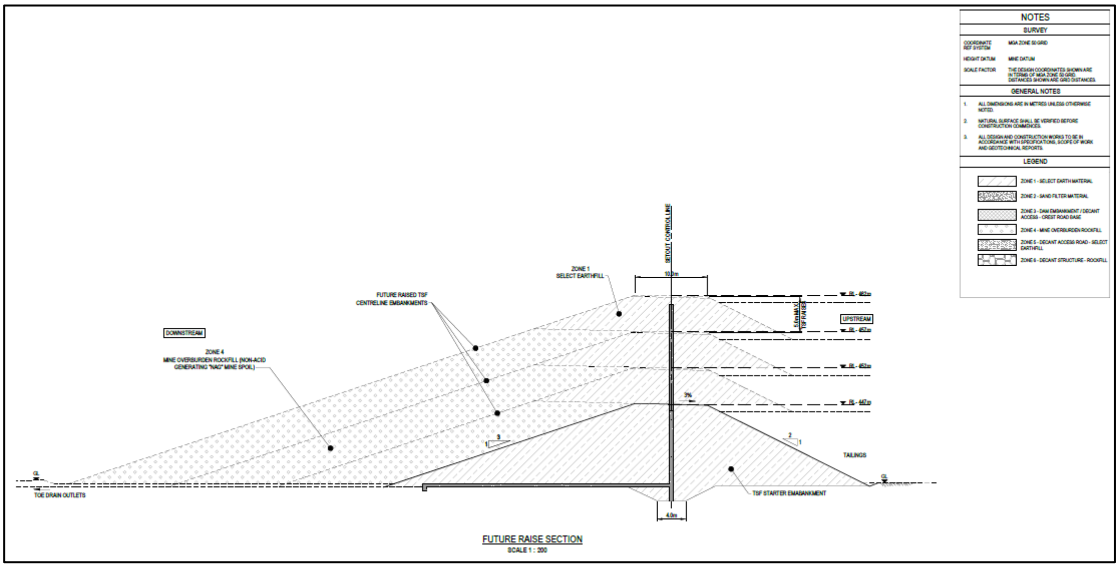


Figure 18-23: Tailings Storage Facility Raises

18.1.19. Landfill Facility

The landfill facility will be a Class II putrescible landfill (unlined). The landfill would take waste from village and mine operations only, such as food scraps, paper and low-density waste. It does not allow for construction or mine waste, which would be sent to the waste rock dump. The design allowance is for 1 kg of waste per person per day.

The landfill would be a 150m x 30m x 4m deep trench, a sufficient volume to cater for the expected waste over the LOM. The area would be surrounded by a fence for safety and to deter local wildlife from entering the landfill. To minimise construction and excavation costs, a relatively flat site with no rock was chosen, away from waterways.

18.1.20. Wastewater Management

Ablutions and kitchenette wastewater from the gatehouse, airport, haulage contractor, mining contractor and plant will be contained locally in septic tanks with a leach drain.

The village will include a dedicated wastewater treatment plant.

18.1.21. Laboratory

An onsite laboratory facility has been included for analysis of metallurgical control samples and plant samples, enabling grade and plant control. The onsite laboratory is preferred over sending samples offsite for testing as it will allow for faster turnaround of test results. As laboratory testing is a specialist task, it is proposed that the onsite laboratory facility will be a BOO arrangement with a specialist laboratory operator providing the transportable building, laboratory equipment and labour required to perform all the required tests.

18.1.22. Gatehouse and Security

The gatehouse will be situated on the access road before the intersection splitting off to the haul road and the rail loop loading area. Access to site will be controlled, with a boom gate allowing authorised personnel through.

The majority of workers accessing the mine area would initially travel by bus from the airport, then commute daily by bus from the village, all within the controlled area.

The gatehouse would comprise an office, induction room, first aid, kitchenette, and ablutions. Parking and a truck waiting bay are adjacent to the building. A gravel vehicle turnaround is provided, to minimise disruption to access while allowing unintentional visitors or tourists to exit the area. Security personnel will be able to maintain radio and internet contact with site and head office to authorise access and conduct inductions (see Section 18.1.13).

18.1.23. Explosives Storage

Explosives storage compounds for bulk ANFO and detonators have been located southeast of the Moonshine pit in compliance with relevant separation distances. Facilities will be fenced and bunded in accordance with statutory requirements.

The ANFO store and magazine containers for detonators & boosters will be supplied by the mining contractor responsible for drill and blast operations.

18.2. Port Development Option

In addition to the base case port operations utilising existing rail unloading facilities, the Feasibility Study also considered an alternate case should rail unloading capacity be unavailable at the time of development of the Project. Under this scenario a new rotary car dumper would be constructed at the port in addition to a rail loop to allow a full consist to enter the port. Engineering studies were completed and costs developed for this option. High level capital costs are presented but are not included in the financial analysis of the project.

During the course of 2021 Macarthur held discussions with the SPA to identify an export solution. SPA drafted a Port Masterplan in 2021 which identifies a development strategy for increased iron ore export. This includes an upgrade of Berth 3, including ship loader capacity increase and significant reconfiguration of the rail system within the Port, to support improved throughput capacity. There is also a future option for expansion of Berth 3 to support an additional vessel alongside.

Item		Scope	Trade / Sector	Timeframe
16) Rail loop (including storage, materials handing, road and Berth 3 upgrades)	16a) Rail loop	Rail loop and associated reclamation	**Iron Ore / Mineral Bulk**	Short term
	16b) New road	New road development around rail loop – link to item 16a		
	16c) Container hardstand	Relocate container hardstand area		
	16d) Berth 3 materials handling circuit	Adapt Berth 3 materials handling circuit for rail loop and road development		
	16e) Workshop relocation	Relocate existing workshop for new rail loop alignment		
	16f) Additional storage	Additional new storage shed developments to support 9.8 mtpa >18 mtpa storage requirement (market dependant)		
	16g) Light vehicle access (rail loop)	Horseshoe bridge to provide light vehicle access into new rail loop		
	16h) New train unloader	New multi-user train unloader (existing RCD to be replaced in approx. 2032, market dependant)		
	16i) Materials handling infrastructure	New materials handling infrastructure to tie into existing and new storage and outload circuits		
	16j) Berth 3 ship loader upgrade	Berth 3 ship loader upgrade to increase load rate (replacement required by 2026)		Medium term
	16k) Berth 3 upgrade for ship loader	Upgrade berth 3 to accommodate upgrade ship loader.		
	16l) Noise management infrastructure	Noise management infrastructure		Short term
17) Multi User truck unloading		Provision of a truck unloading facility to maximise storage loading efficiency	Mineral Bulk	Short term
18) Berth 3	18a) Mooring upgrade	Berth 3 mooring system upgrade	Iron Ore	Short term
	18b) Berth-side walkway	Berth 3 walkway to be made independent of main structure and Berthing Dolphin (BD) 3 to be structurally connected to BD2 and BD4 to stop pile fatigue from long period waves.		
	18c) Breakwater extension	Extension of the Berth 3 breakwater for mooring stability and siltation outcomes		Medium term
19) Reclad Sheds 2 and 4		Recladding of Shed 2 and Shed 4		
20) Shed 1 / 2 Trade Optimisation		Reconfigure Shed 1/2 to accept road based products in the event additional Iron Ore storage sheds constructed out on reclaim as per 16f		Medium term

Figure 18-24: SPA Masterplan Short- and Medium-Term Port Development Items

In December 2021, Macarthur presented to SPA a design allowing for a new iron ore circuit to align with the SPA Masterplan for a multi-user iron ore facility. The design allows for integration with existing operations with minimal disruption. Further criteria were:

- 100 year design life
- rail line extension and new rail loop to remove existing port rail constraints
- a new twin-cell rotary car dumper (RCD) with unload capacity of 4500tph (existing RCD to remain operational until new RCD in place)
- provision for three new storage sheds of approximately 250,000 t each
- direct unloading to ship or shed
- integration of circuit to existing iron ore and spodumene sheds, and
- land reclaim as required but minimised for environmental and capital reasons.

The proposed development excluding the concentrate storage shed is to be funded by a third-party infrastructure asset group. Under this scenario, the Company would be charged a tariff for material handled through the circuit, operated by SPA or the asset owner. The treatment of capital as an operating cost is detailed in Section 21.

Overall isometric and general plan views of the proposed design are shown in Figure 18-25 and Figure 18-26 and further detailed in the following sections.

Due to installation of the new rail loop, the existing ship feed conveyor would require modifications by way of a new elevated conveyor section and new elevated truss, as indicated in Figure 18-26.

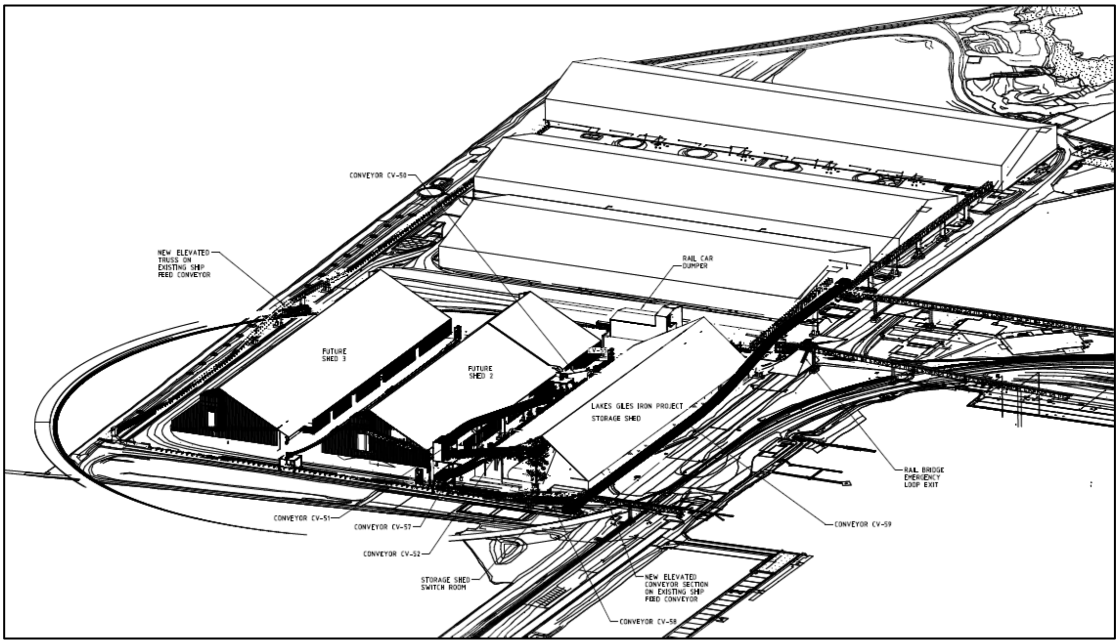


Figure 18-25: Port Area Site Plan Isometric

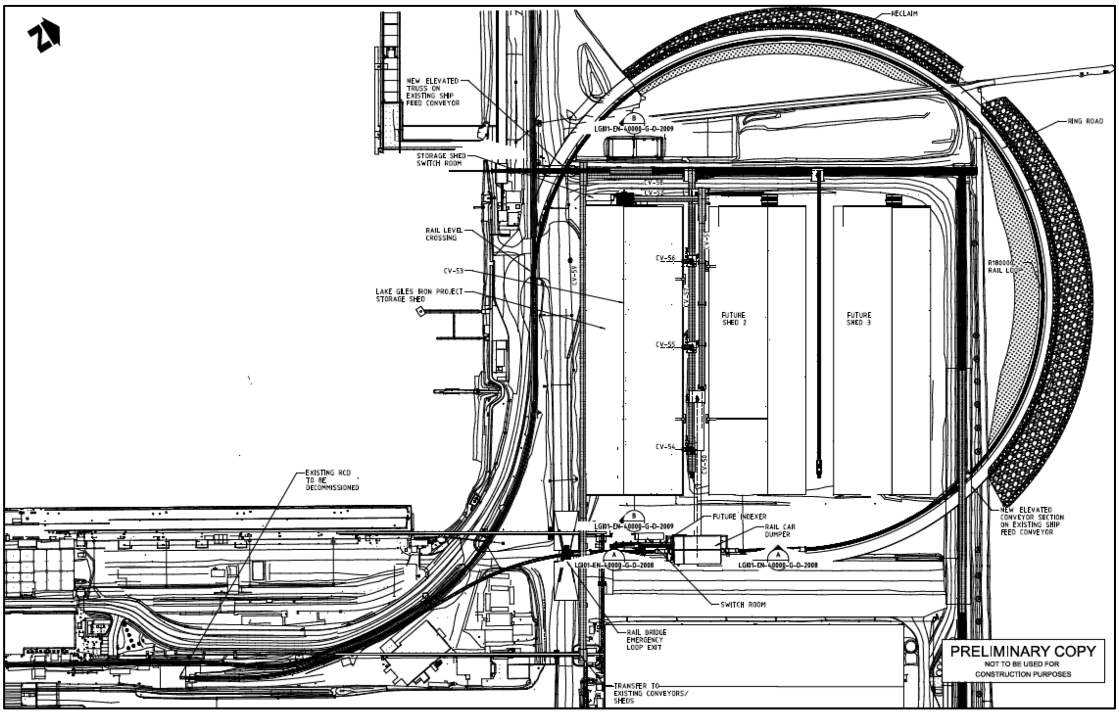


Figure 18-26: Port Area Site Plan General Arrangement

18.2.1. Rail Unloading

Port throughput capacity would be 18 Mtpa with 3 Mtpa required for Macarthur, plus 11 Mtpa by others. Macarthur's rail unloading at the port would operate 330 days per year, with three operational shifts unloading up to seven trains per week.

The RCD will allow for two rail wagons to be unloaded simultaneously enabling greater throughput compared to the existing single cell RCD.

Rail Loop

The designed rail extension and new rail loop facilitate the full 1800 m consist to enter the port. The new track works would enable the ore wagons to pass through the train unloader as a single rake without decoupling. This would remove unloading constraints created by an existing train breakup bottleneck outside of the port.

The rail loop consists of a spur off the existing rail line, to a standard gauge 180 m radius balloon loop. It is proposed that trains will travel anticlockwise on the rail loop to be unloaded. The rail loop has been designed with an outer ring road, with a rail level crossing for access to the inner area and a rail bridge providing alternate emergency exit.

Modifications to the existing ship feed conveyors would be required at crossover points with the new rail loop. These include a new elevated conveyor section and new elevated truss, as indicated in Figure 18-27.

18.2.2. Rail Loop Reclamation

The proposed design to provide the required additional area to support the rail loop and road, is a circular breakwater structure around the existing quay. The road is situated on the outside of the rail loop and is utilised in the protection and drainage design concepts. The breakwater is designed to reduce potential for overtopping during bad weather events and sea level rises, with a coastal infrastructure design life of 100 years.

Reclamation would be incremental, with land reclaim between the new breakwater and the existing quay edge deferred to a later date. This is considered to reduce upfront capital costs compared to the previous more traditional, straight edge (rectangular) reclaim area with an indicative distance from the wall edge to the new and existing infrastructure. Infill would be introduced gradually, as SPA conducts maintenance dredging of the channel.

Construction of the breakwater would be staged, with introduction of wall sections to reduce initial capital investment, commencing with seawall construction to support the rail loop. Membrane and drainage arrangements have been considered to reduce water ingress into the deferred reclamation area and avoid stagnation and bacterial build-up. A cross section of the design is shown in Figure 18-27. The final stage of construction would allow for potential future sea level rise by installation of a reinforced concrete adaptation headwall, with compacted material under the rail loop to be further stabilised with geogrid.

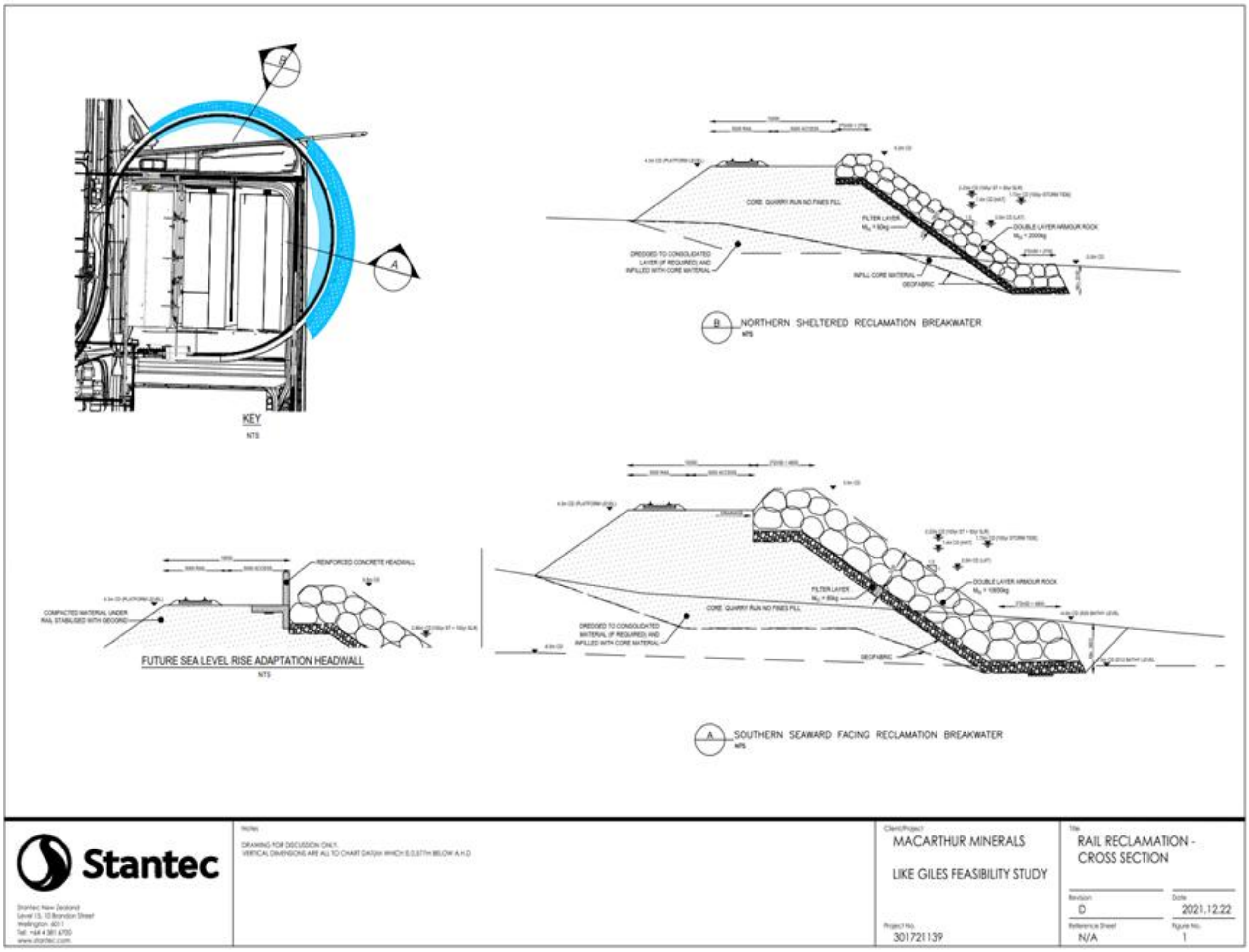


Figure 18-27: Port Rail Reclamation Cross Section

18.2.3. Rail Car Dumper

Unloading of the rail wagons at 4500 tph will be by a hybrid twin rail car dumper (RCD) with capacity for both rotary tippling and direct bottom dump. The new hybrid RCD is to be located on the rail loop with a switch room adjacent. The existing RCD could be decommissioned once the new RCD is operational. The new RCD will be housed inside a shed with a dust extraction hood fitted. Wheel grips are included in the design to allow for rotary tippler unloading, with allowance for a future indexer.

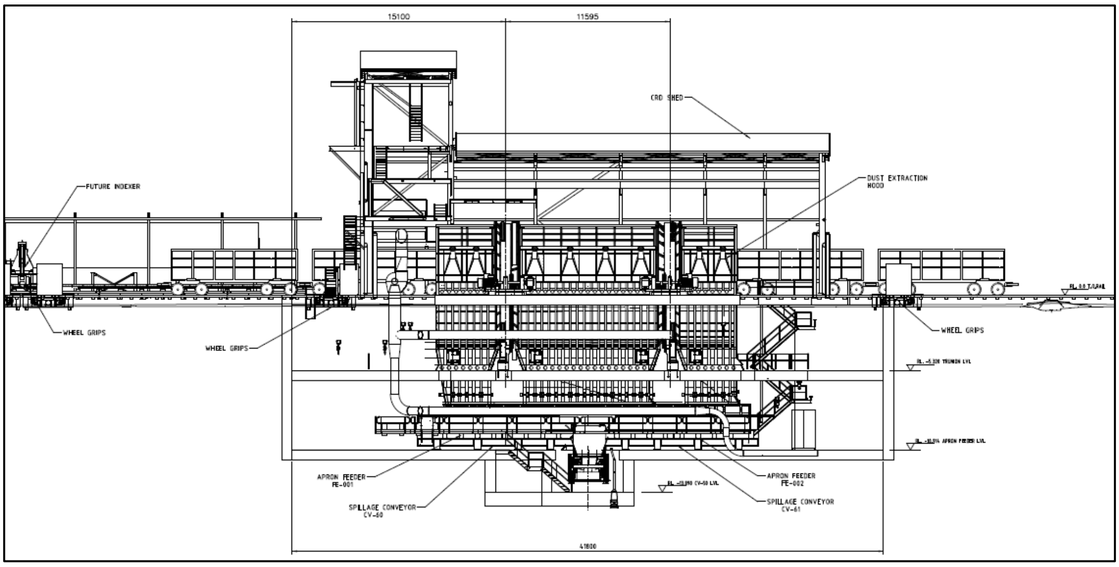


Figure 18-28: Rail Car Dumper Cross Section

18.2.4. Product Unloading

The wagons will discharge onto apron feeders, with spillage conveyors as required, to be directed onto a car dumper discharge conveyor. A diverter chute will direct the concentrate to either a transfer conveyor to the shed storage, or to the shed reclaim conveyor. The shed storage transfer has an allowance for transfer to future sheds, with the concentrate passing through a diverter chute before flowing onto the shed tripper feed conveyor.

The shed reclaim conveyor can also be fed by three shed reclaim belt feeders. The shed reclaim conveyor discharges into a divertor chute which directs the concentrate either to conveyors feeding the existing shed system, or to the existing ship loading system. The design allows for simultaneous unloading and ship loading of concentrate.

Belt and chute washing has been included in the design to reduce cross contamination due to the multiuser nature of the facility. A sump pump for the car dumper area will collect any runoff. A dust collection system has been included, with return to the car dumper discharge conveyor, to minimise losses to the surrounding area. The car dumper discharge conveyor and shed reclaim conveyor are both fitted with weightometers to allow online measurement of the concentrate unloading rate from the car dumper and car dumper/shed respectively.

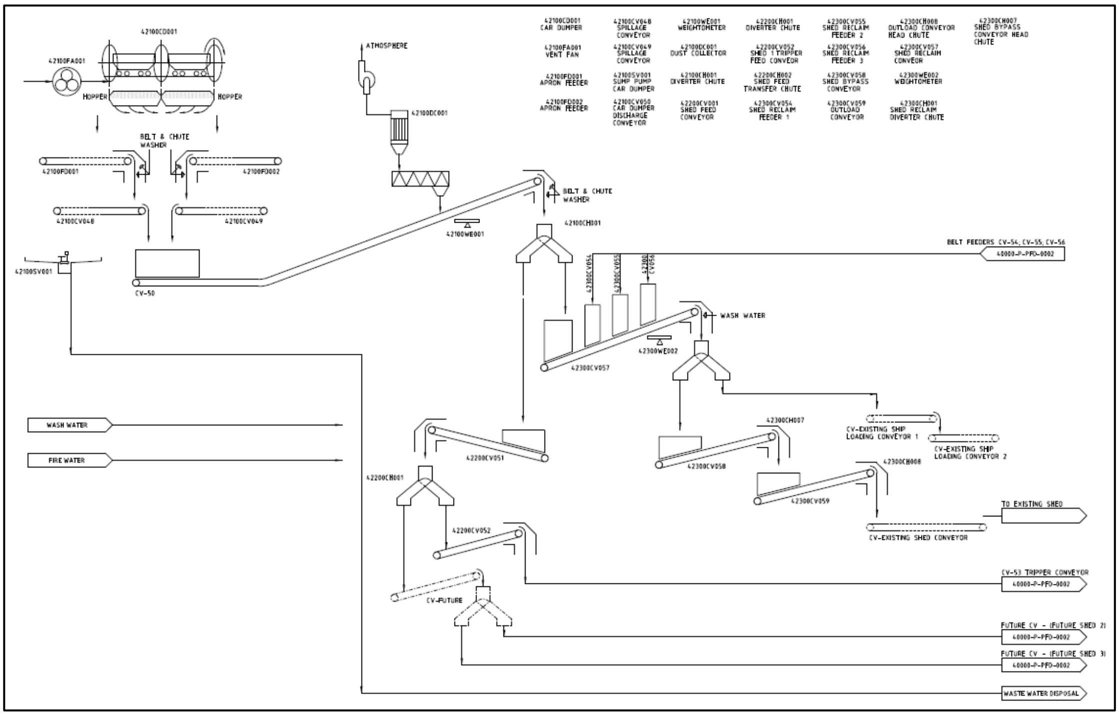


Figure 18-29: Port Unloading Process Flow Diagram

18.2.5. Product Storage

The environmental conditions of the Port's operating licence require iron ore to be stored in a sealed shed to minimise the impacts of dust. The Port has three sheds designated for iron ore storage but at present are owned or leased by another operator.

There would be land available at the port for the construction of up to three (3) new storage sheds inside the new rail loop (storage is equivalent to two cape size vessels). The design includes an initial single new storage with design capacity of 260,000 t. A cross section of the design is shown in Figure 18-30.

The shed would be fed by a tripper feed conveyor discharging onto the shed tripper conveyor. Future sheds would require their own dedicated tripper feed and tripper conveyors. The shed tripper conveyor forms a longitudinal stockpile in the shed of up to 25 m in height and a combined length of up to 200 m. A dust extraction and collection system has been included, with return to the shed reclaim conveyor, to minimise losses to the surrounding area.

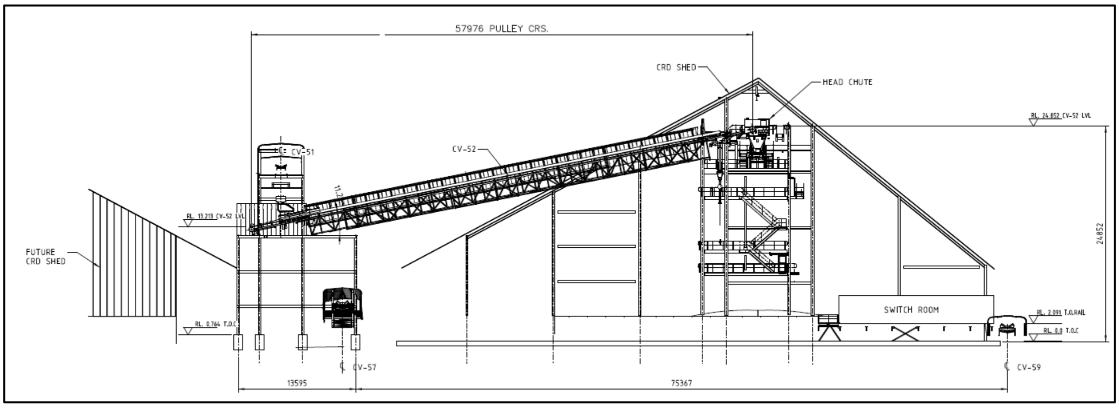


Figure 18-30: Port Storage Cross Section

18.2.6. Product Reclaim

Product reclaim from the storage shed will be by front end loader into one of three reclaim hoppers. Each hopper would discharge onto a dedicated shed reclaim feeder. The concentrate would flow from the feeders onto a shed reclaim conveyor.

The shed reclaim conveyor discharges into a divertor chute which directs the concentrate either to conveyors feeding the existing shed system, or to the existing ship loading system. The design allows for simultaneous unloading and ship loading of concentrate.

Currently SPA manages iron ore reclaim by FEL from existing sheds. Costs for reclaim by SPA have been allowed for, as detailed in Section 21.

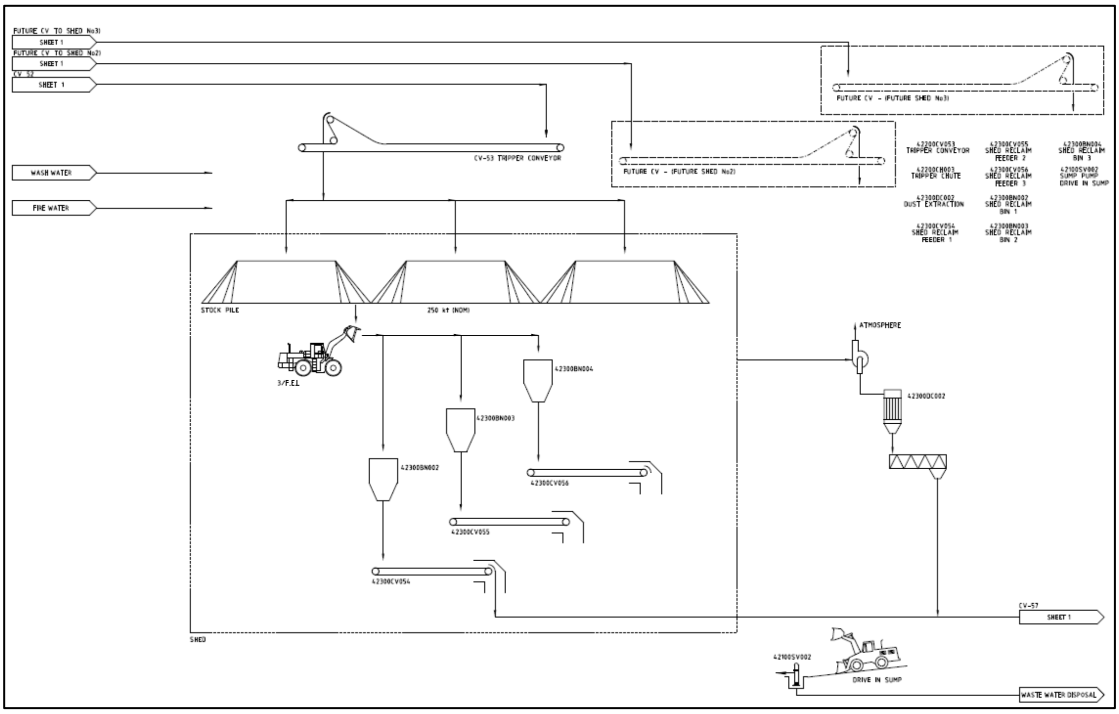


Figure 18-31: Port Reclaim Process Flow Diagram

19 Market Studies

19.1 Executive Summary

19.1.1 Approach to forward iron ore price forecast

The forward iron ore price adopted for the Lake Giles Iron Project in this Report is based on the Company's assessment of published consensus pricing, forecasts derived directly from steel mills, the various analyst reports described below and a comparison of historical analyst forecasts against actual pricing over time. The Company has then adopted an adjustment for grade using historical and projected premiums to arrive at a long-term price for the Lake Giles concentrate.

Information on current and forward product demand characteristics, product marketing and pricing were supplied by Glencore and were also derived from published research reports prepared by Wood Mackenzie and major global iron ore producers and marketers (such as BHP's published price and market forecasts).

Macarthur also engaged LFJ Consulting Pty Ltd (LFJ) to undertake an iron ore market and price analysis. LFJ's analysis has been considered and utilised in the preparation of this market studies chapter in conjunction with the other market data referred to above

A long-term CFR China sales price of US$131.40/dmt for Macarthur's 66.1% Fe concentrate product specification has been adopted, based on forecast pricing for 62% Fe CFR China of US$99/dmt through to 2050 with an adjustment for grade and a magnetite premium. This is expected to result a realised free on board (FOB) sales price of USD$120.30/dmt after shipping and marketing costs.

The pricing scenario is consistent with the Company's determination of current consensus price forecasts to 2050. The Company considers that the pricing scenario is appropriate for the Lake Giles concentrate product specification, and it has been normalised for the highs and lows experienced throughout early 2020 and into mid-2021 (which was a period that largely reflected the market response to the uncertainties of the Covid-19 pandemic).

The pricing analysis is based on iron ore industry knowledge, experience, and on information available from company, industry, trade, government and other sources that may be limited. The analysis, estimates or projections considered for this Study, together with other sources of input, are based upon information and upon assumptions that are subject to significant degrees of economic, commercial, market, industrial and other uncertainties.

19.1.2 Glencore Offtake Agreement

On 21 March 2019, Macarthur announced that it had entered into a binding Offtake Agreement with Glencore for the offtake and exclusive marketing rights to iron ore produced from the Project. The Offtake Agreement is binding and guarantees the purchase of Macarthur's product after it passes the ship rail. Under the agreement, Glencore is responsible for the marketing of all products, and Macarthur assumes no credit risk.

The key terms of the Glencore Offtake Agreement are as follows:

- Offtake and marketing rights apply to up to 4 million tonnes per annum for the first 10 years, with an option to extend for a further 10 years for all tonnes of future iron ore production from the Project.
- Glencore agrees to release up to 70% of their off-take volume if Macarthur secures project financing from a Strategic Industrial that also secures off-take of the product produced.
- Glencore will take possession of the iron ore once it has been loaded onto a vessel for export.
- Glencore is responsible for the marketing, shipping, delivery and associated freight insurances, which will be adjusted back against the sales price to the final buyer.

The terms and conditions of the Offtake Agreement were competitively negotiated, and at the time of execution, reflected strong forward iron ore demand.

19.2 Iron Ore Market Overview

19.2.1 Background to the market

Iron ore is the most commonly utilised metal worldwide and it is the fourth most abundant element. Global demand for iron ore is strong as it is arguably more integral to the global economy than most other commodities. It is the key component used in the steel making process and is therefore an essential mineral for the construction, engineering, automotive and machinery industries.

Steel is the most widely used metal in the modern economy and it is the primary building material and indicator for industrialization, urbanization and economic wealth. Population growth, the ‘infrastructure of decarbonisation’ and rising living standards are all expected to drive demand for metals like iron ore for the next several decades.

19.2.2 Current price trends

BHP’s Economic and Commodity Outlook FY 2022 Half Year predicted the global population to expand by 0.8 billion to 8.5 billion, and urban populations to also expand by 0.8 billion to 5.2 billion (BHP, 2022). At the same time, it predicts nominal global GDP to expand by $74 trillion to $161 trillion, and capital spending to expand by $14 trillion to $37 trillion. BHP’s analysis noted that “Each of these basic fundamental indicators of resource demand are expected to increase more in absolute terms than they did across the 2010’s.” Furthermore, it noted that “Comprehensive stewardship of the biosphere and ethical end–to–end supply chains will become even more important for earning and retaining community and investor trust.”

With a strong supporting narrative for future iron ore demand, BHP’s Economic and Commodity Outlook for the 2021 financial year also reported:

- Iron ore prices (62% CFR, Argus) were very strong over the second half of financial year 2021, ranging between USD$150/dmt and USD$236/dmt; and
- Whilst overall fines stocks increased by 7 Mt over first half of CY 2021, premium branded fines declined by 8 Mt. These trends resulted in widening discounts for lower grade and unbranded products, higher realisations for medium and high-grade branded products and very attractive premia for lump.

The price for 62% Fe CFR China pulled back to approximately USD$92/dmt in early November 2021, reflecting forced cuts to Chinese steel production and weaker demand for steel in China in the second half of the 2021 calendar year. However, there was a recovery of iron ore prices late in Q4 2021 and into Q1 2022 (with spot prices approaching USD$130/dmt CFR China towards the end of January 2022 and reaching around USD$150/dmt in February 2022; Wood Mackenzie 2022). On that basis, Wood Mackenzie raised its Q1 2022 forecast for 62% Fe CFR China to USD$125/dmt (previously USD$90/dmt) and lifted its full year forecast to USD$105/dmt.

19.2.3 Long-term price trends

In the short term, most analysts have forecast a downward price trajectory for 62% Fe CFR China from Q2 2022 onwards, although recent indicators suggest the decline may be shallower than previously thought and the starting point higher than previously thought.

The outlook over the longer term to 2050 suggests that a key challenge for suppliers of seaborne iron ore over the coming period is how to adapt to a market that is on the verge of going ex-growth but with increasingly stringent quality requirements to meet the needs of steel decarbonisation (Wood Mackenzie, 2021).

Changes to global environmental regulations on emissions and policies targeting net zero carbon outcomes is generating a preference for high-grade iron ore with lower impurities.

Consequently, a growing premium gap for high-grade iron ore is emerging and looks set to continue at around 20% or higher against 62% Fe benchmark prices over the next several decades. This trend is discussed further below.

19.3 Iron Ore Types and Products

19.3.1 Iron ore

Iron ore is generally found in the form of iron oxides. The most common forms are hematite (Fe_2O_3) and magnetite (Fe_3O^4). Other forms of hydroxide minerals including goethite (FeO(OH), limonite (FeO(OH).n(H_2O) (which are formed from the weathering of hematite), and carbonate minerals such as siderite ($FeCO_3$) are also mined.

Iron ore is typically categorised according to its iron-content grade and other physical properties. High grade iron ore products are generally above 63% Fe, whilst medium-grade iron ore products range between 58%-63% Fe and low-grade iron products grade below 58% Fe.

19.3.2 Iron ore types

The difference between hematite and magnetite is material. Hematite iron ore typically has a much higher in-situ iron content (which can be up to 65% Fe) and is referred to as "direct shipping ore" (DSO). This is because its production typically only involves basic crushing and screening of the raw ore to separate lumps from fines and/or minor upgrading to slightly increase the iron grade of the saleable product prior to being shipped. Because DSO is not more extensively processed, it typically contains much higher levels of impurities than magnetite ore. Significantly higher concentrations of impurities such as phosphorous, silica, sulphur, alumina and moisture can attract price discounts (which are effectively penalties in trade).

The current global seaborne trade in iron ore is overwhelmingly dominated by DSO. DSO fines currently make up more than an estimated 60% of world seaborne trade, while lump DSO accounts for approximately 15%, with concentrates (and pellet feed) having a similar share to lumps. Pellets make up the balance at less than 10%.

In contrast to hematite, magnetite iron ore has a comparatively lower in-situ iron content (which can typically be within a broad range of between 15% to 40% Fe). Additional processing is required to produce a high-grade (+65% Fe) product with low impurities. This generally involves crushing, screening, grinding, magnetic separation, filtering and drying. Consequently, magnetite projects often have capital and production costs that are higher than hematite projects. Processing of magnetite ore is also generally more energy intensive, and as a result the cost differential to the production of hematite ore is generally directly related to the difference in energy costs. Magnetite (Fe_3O_4) has an additional oxygen molecule in its composition when compared to hematite (Fe_2O_3). When this extra oxygen molecule liberates in the furnace, it has the potential to increase furnace productivity and decrease energy costs during the steel-making process. The difference in the typical processing requirements between magnetite and hematite is demonstrated at a high level in Figure 19-1.

The superior chemical and thermodynamic properties of magnetite make it an ideal product for use in electric arc furnaces (EAF) in combination with scrap steel. The gradual increase in adoption of EAF globally over the last several decades in preference to blast furnaces (particularly in North America) may allow the steel industry to eventually take the further step of using hydrogen as a reductant in EAFs, rather than coking coal. If the hydrogen is produced using renewable energy, then it is possible to produce what is being termed 'green steel' (i.e., steel produced with zero net CO2 emissions). The difference between the production of steel via the blast furnace route versus the EAF route (with an overlay of renewable energy and hydrogen as the reductant) is shown in Figure 19-2.

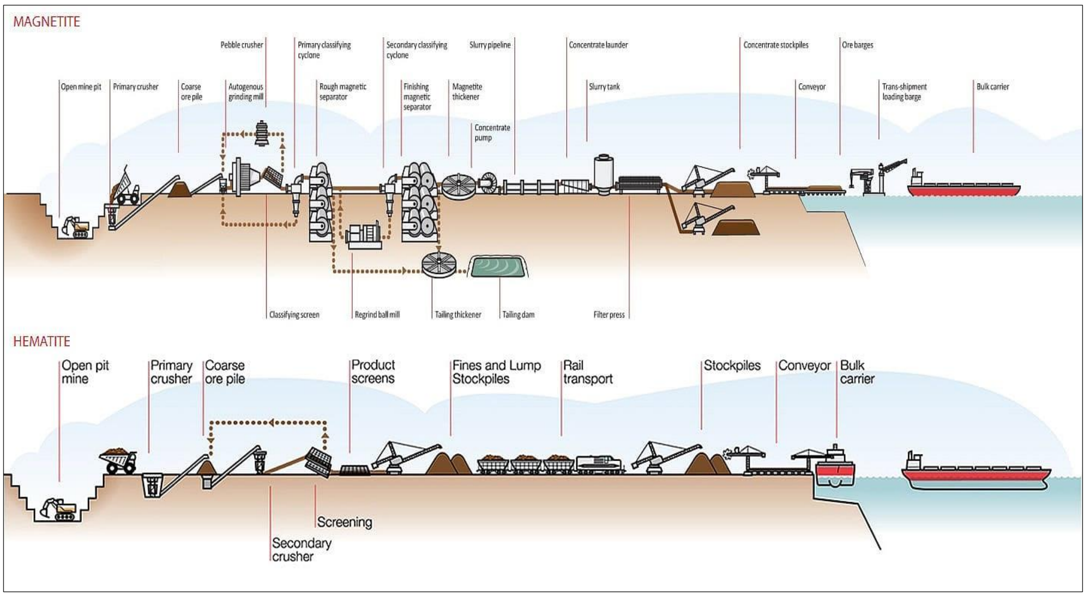


Figure 19-1: The iron ore production chain

Source: Ten Squared

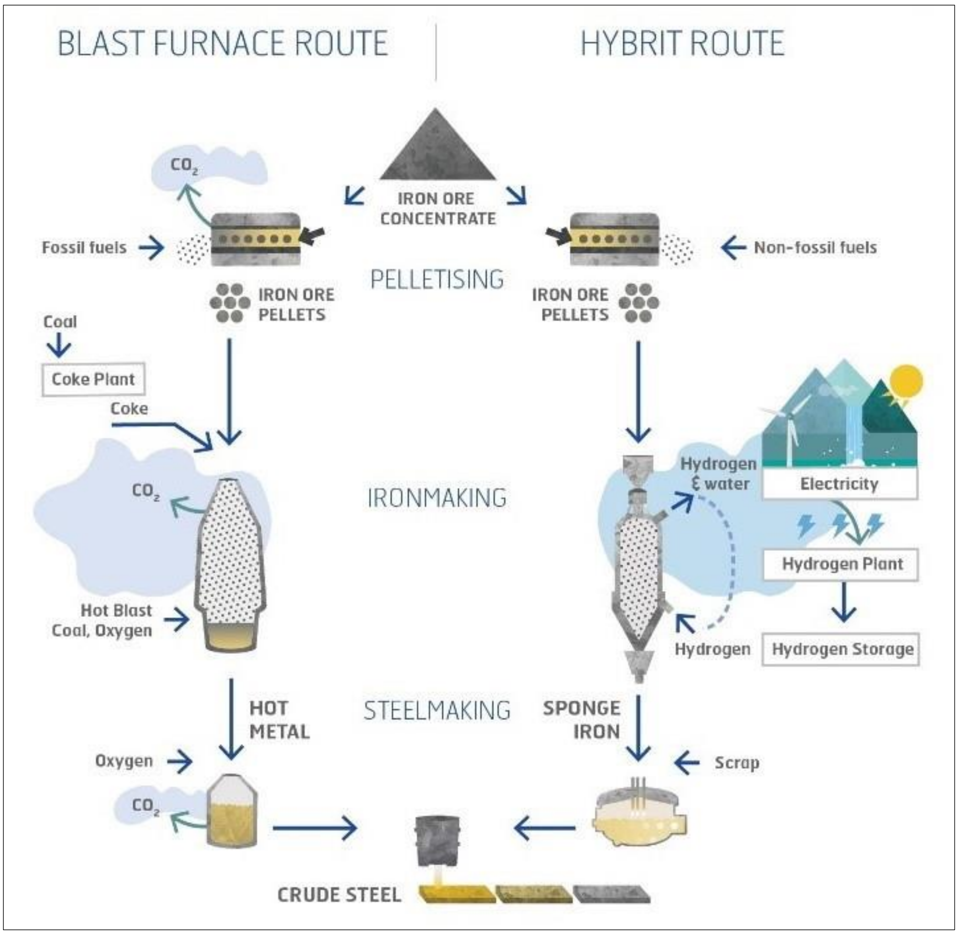


Figure 19-2: The steel production chain with hydrogen as a clean energy alternative

Source: Fuel Cell & Hydrogen Energy Association.

19.3.3 Iron ore products

Saleable iron ore products are typically sold in the following physical forms:

- Lumps – sized between approximately 6 mm up to 30-35 mm
- Fines – sized between ~0.150 mm to 6.3 mm (and sometimes up to 10 mm)
- Concentrates – intensively processed ore with particles less than 1 mm
- Pellet feed – fine concentrates with most particles typically less than 0.050 mm (50 microns); and
- Pellets – 6 mm to 18 mm balls made by the agglomeration of pellet feed.

Lumps and pellets products are generally referred to as “direct charge” products as they are suitable for directly charging into a blast furnace, whereas fines and concentrates first must be agglomerated into a lumpy material, either by sintering to form a clinker-like material or by pelletising it to form pellets.

19.4 Major Iron Ore Markets Size and Structure

The global market for iron ore was estimated at 2.1 billion metric tonnes in 2020 and is forecast to reach approximately 2.7 billion metric tonnes by the year 2026. This represents a forward growth rate at a CAGR of + 3.5% in the medium term.

The iron ore market in the United States of America was estimated to be approximately 32.7 million metric tonnes in 2021, accounting for around 1.5% of the global market. China, the world's second largest economy, is forecast to reach an estimated market size of 1.6 billion metric tonnes in 2026, trailing a CAGR of 4.1% between 2021 and 2026.

Japan and Canada are each forecast to grow at 1.6% and 1.9% respectively over the medium-term to 2026. Germany is forecast to grow at approximately 2.9% CAGR whilst the balance of the European market will reach 1.7 billion metric tonnes by the end of 2026. Generally between 2020 to 2035, the EU and UK’s share of global iron ore consumption is set to increase, 1%, the USA and Canada will increase by 2%, China will decrease by 3%, India will increase by 5%, Japan will decrease by 1% and the rest of South East Asia will increase by 7% (see Figure 19.3 below).

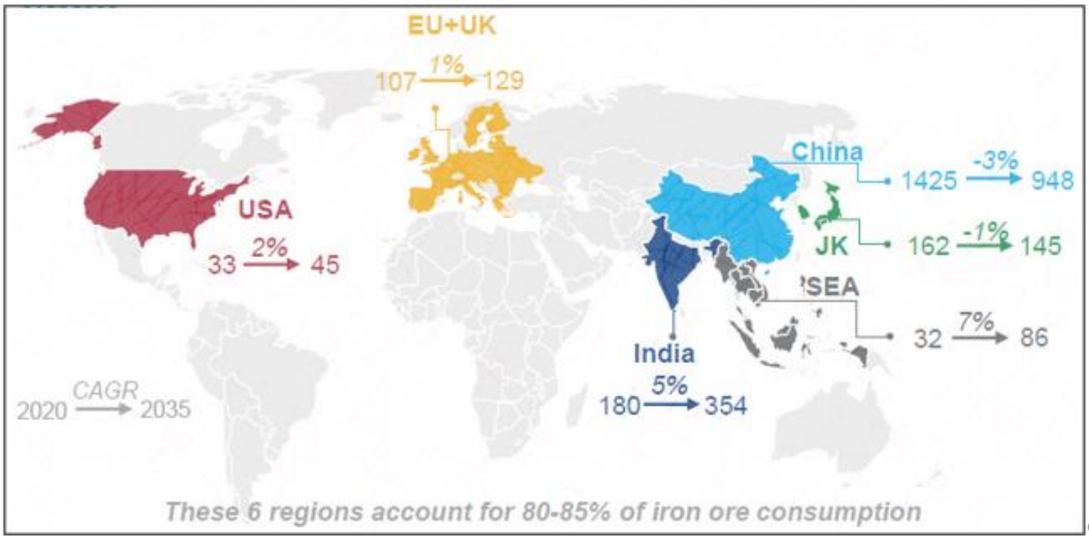


Figure 19-3: Key global iron ore consumption regions

Source: Wood Mackenzie

Traditionally, China has been the key driver to the growth in iron ore industry across the globe and this is expected to remain the case for the next several years. Any change occurring in China's steel production significantly influences global iron ore trade – a dynamic that was evident during the second half of CY 2021. Chinese demand for iron ore is primarily due to its rapid urbanization and industrialization and a robust GDP growth, which exceeds the growth rates of most western countries.

19.5 Iron Ore Supply and Demand

Assumptions about future steel demand and production scenarios are important to iron ore pricing forecasts and require a consideration of both future global economic growth and the projected intensity of future steel usage. Planned and potential iron ore supply scenarios are also relevant. These assumptions have considerable uncertainty attached to them over the longer term.

19.5.1 Steel demand and production

Wood Mackenzie currently forecasts a +3.5% compound annual growth rate in global steel production between 2021 and 2035 which bodes well for the iron ore industry. However, it also forecasts a number of 'big' changes to the market, particularly with regard to iron making 'routes' driven by decarbonisation. China currently dominates steel production and consumption with 56.5% of global steel production in 2020 (See Figure 19-4).

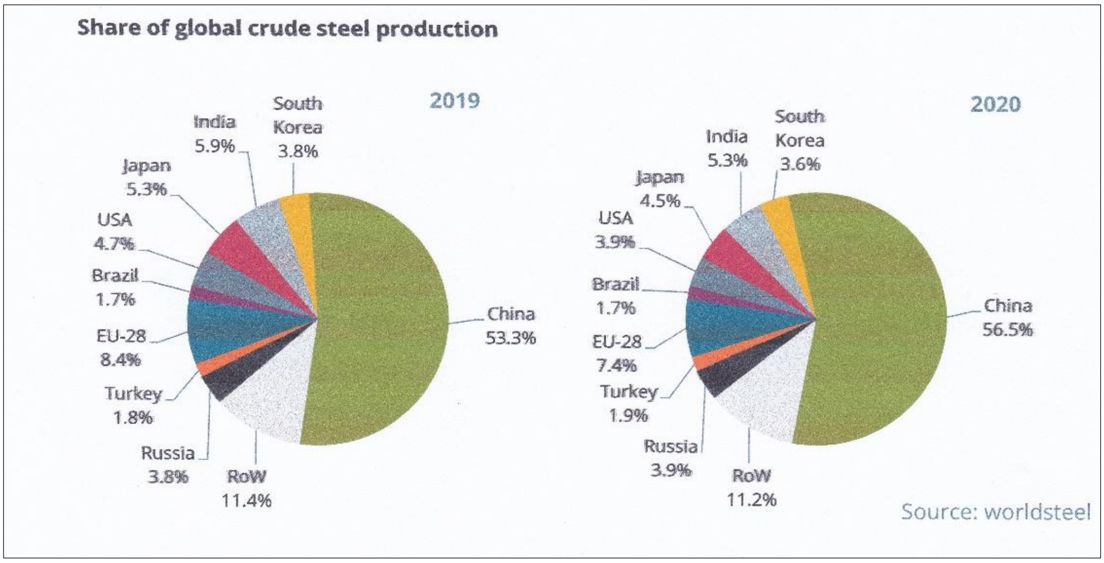


Figure 19-4: Share of global crude steel production. Source: worldsteel

Source: worldsteel

Starting from 2021, Chinese steel output is set to decline over the next several decades as strict production control measures designed to limit emissions continue.

The Chinese Central Government has recently confirmed through "guiding opinions" issued for its iron and steel industry that it plans to achieve peak carbon emissions "before 2030" (Carbon Brief, 2022) See Figure 19-5.

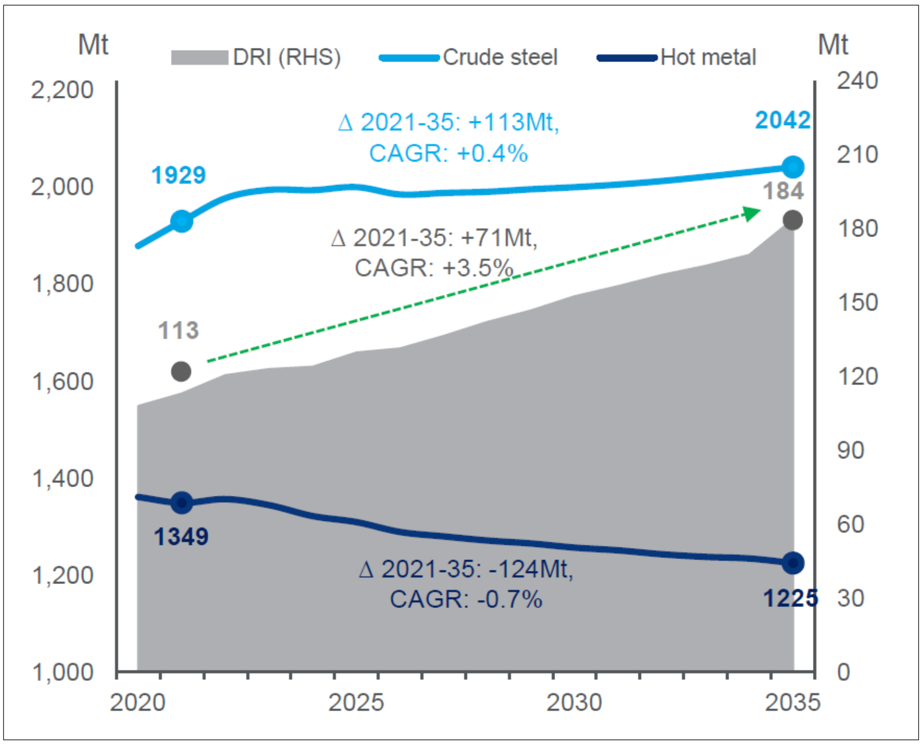


Figure 19-5: Global Steel and Iron Forecast to 2035

Source: Wood Mackenzie; WSA

Whilst steel output from China is set to decline moderately to 2050, the fall in Chinese output is forecast to be compensated by all other major economies. An increase in incremental supply from India and other parts of south-east Asia is expected to underpin a robust demand outlook.

Global hot metal production is forecast to fall for the period through to 2035. (See Figure 19-6 below).

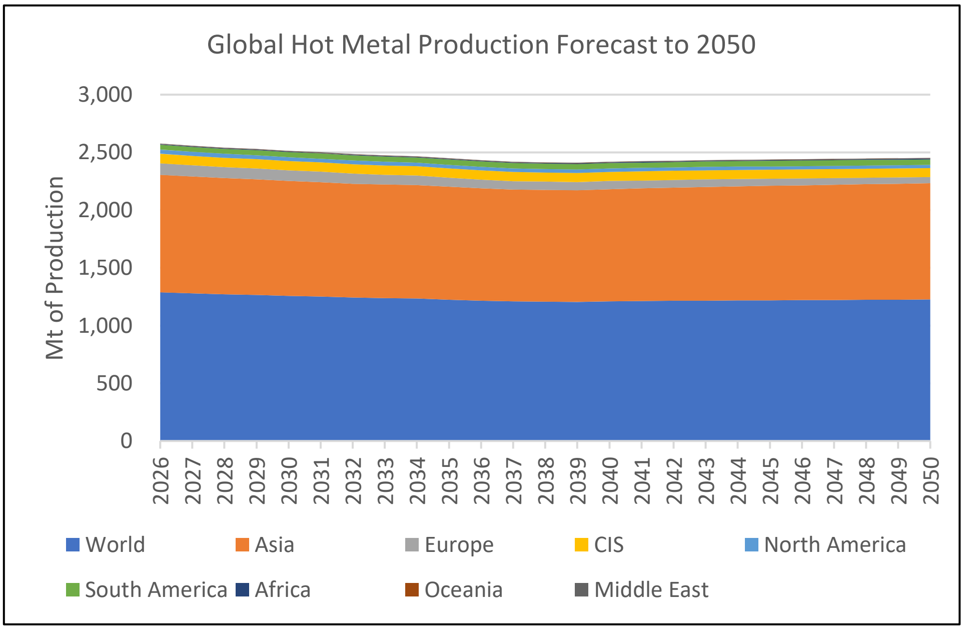


Figure 19-6: Global Hot Metal Production Forecast to 2050

Data source: Wood Mackenzie; worldsteel.org

The production of crude steel is forecast to largely stagnate, driven by decarbonisation that will drive a swing in favour of scrap steel (at the expense of pig iron) and a continued transition to the use of EAF. (See Figure 19-7 below).

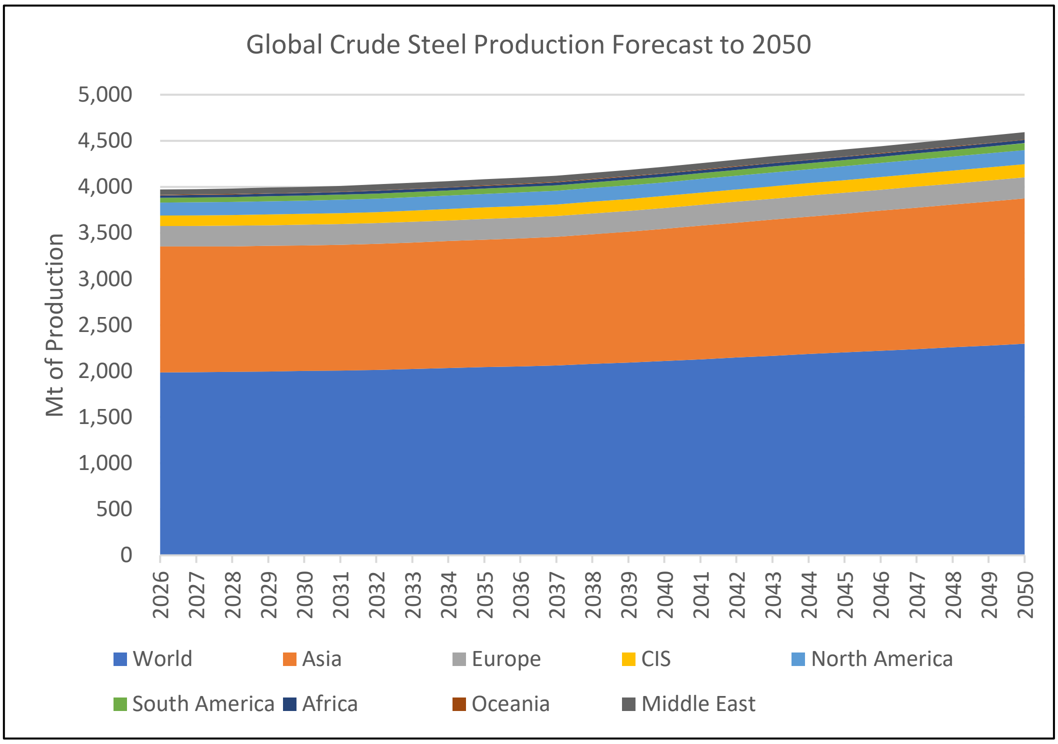


Figure 19-7: Global Hot Metal Production Forecast to 2050

Data source: Wood Mackenzie; worldsteel.org

At the same time however, direct reduction iron (DRI) production will partially offset the reduction in hot metal production, and DRI production generally will experience a significant upward revision, with the largest increase in DRI production set to occur in Europe where Wood Mackenzie are forecasting DRI production to reach 26 Mtpa by 2035 (versus 1 Mtpa in 2021), much of which may be fuelled using hydrogen as the reductant. (See Figure 19-8 below).

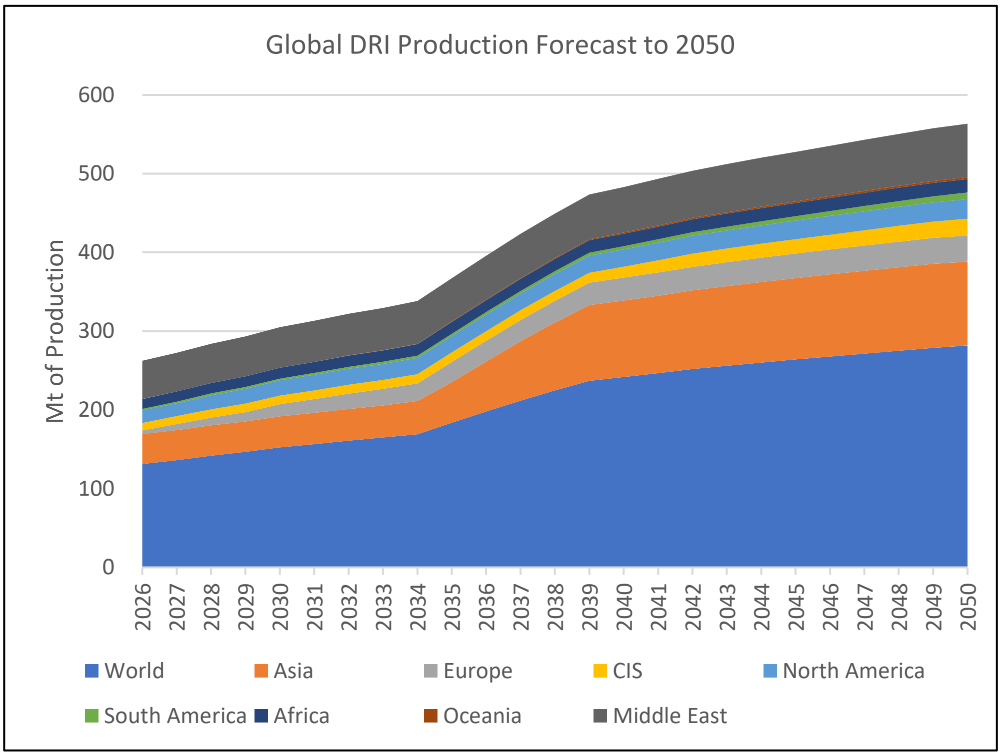


Figure 19-8: Global Hot Metal Production Forecast to 2050

Data source: Wood Mackenzie; worldsteel.org

19.5.2 Iron ore demand

Global iron ore demand will be driven by the steel production and demand projections referred to at 19.5.1 above. The projected rates shown in Figure 19-9 below reflect the forecast pattern for steel growth. Generally, global growth rates for iron ore are expected to moderate, particularly in China, but with a rebound in other major economies.

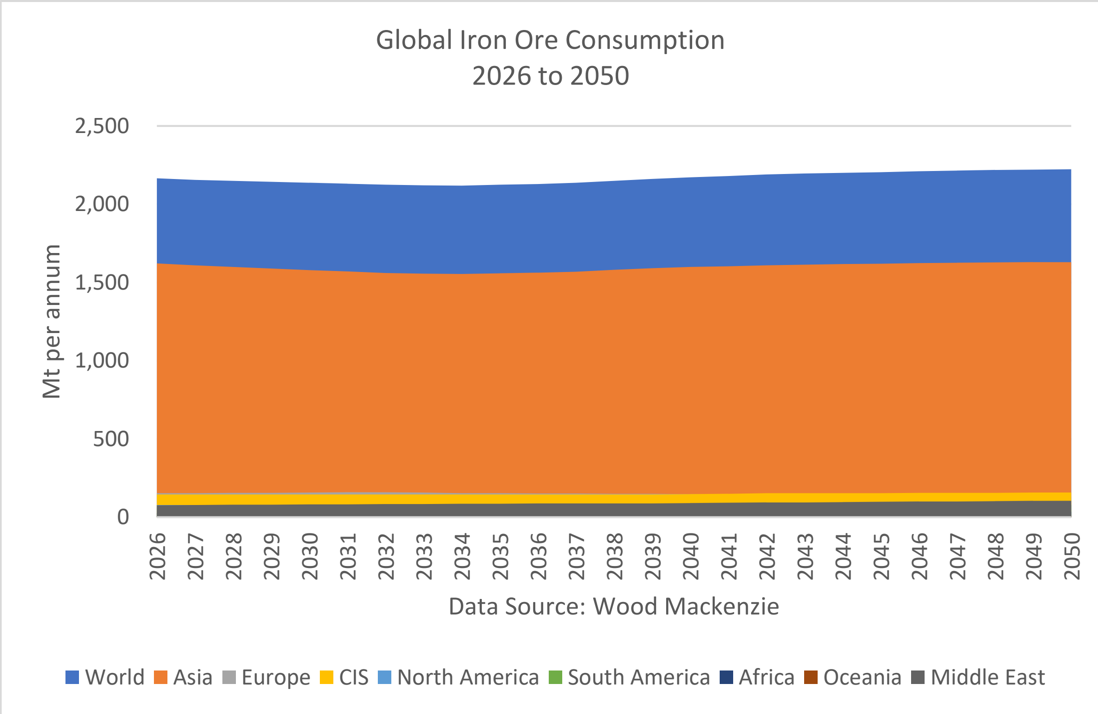


Figure 19-9: Global Iron Ore Consumption 2026 to 2050

Data source: Wood Mackenzie

Global iron ore imports are projected to weaken between 2022 and 2050, reflecting iron ore consumption declines (see Table 19-1 below). Whilst the use of blast furnaces to produce steel is expected to make gains in some emerging economies, across the board, the consumption decline for iron ore is expected to result from an anticipated acceleration in the use of EAF, with the share of EAF in steel production forecast to rise by 41% to 2035. The transition to EAF is expected to result in a global iron ore consumption decline to 2035, as a greater transition of steel production towards EAF continues, mainly as a consequence of the increase in use of scrap steel referred to previously.

Table 19-1: Global Iron Ore Imports (Projections based on assumed iron ore supply, consumption, and trade flows)

Region	Iron Ore Imports (Mt) in Year					
	2026	2030	2035	2040	2045	2050
Asia	1,277	1,206	1,158	1,144	1,132	1,123
Europe	142	146	143	135	126	117
CIS	14	13	13	14	20	23
North America	27	25	31	30	29	28
South America	5	5	5	6	6	7
Africa	17	19	20	21	21	21
Oceania	0	0	0	0	0	0
Middle East	38	38	38	40	45	49
Global Iron Ore Imports	1,520	1,451	1,407	1,390	1,380	1,368

Data source: Wood Mackenzie

Over the longer term, there is forecast to be an increasing focus on productivity and emissions control in the steel making process which will favour demand for high-grade and low impurity iron in EAF, such as magnetite.

Higher grades of iron ore such as magnetite typically require less metallurgical coal to be used in the steelmaking process. The combination of reduced coking coal inputs and lower levels of impurities in higher-grade iron ore products leads to reduced emissions levels (for a given level of output).

The preference for high-grade iron ore with lower impurities is a trend that was exacerbated in 2021 by high steel margins and high coking coal costs, both of which incentivised mills to maximise usage of superior quality feedstock. It was this combination of emissions related curbs and elevated coal prices that resulted in the premium for higher grades of iron ore (65% Fe content and above) against the 62% Fe benchmark price reaching multi-year highs during the first half of 2021 (see Figure 19-10 below).

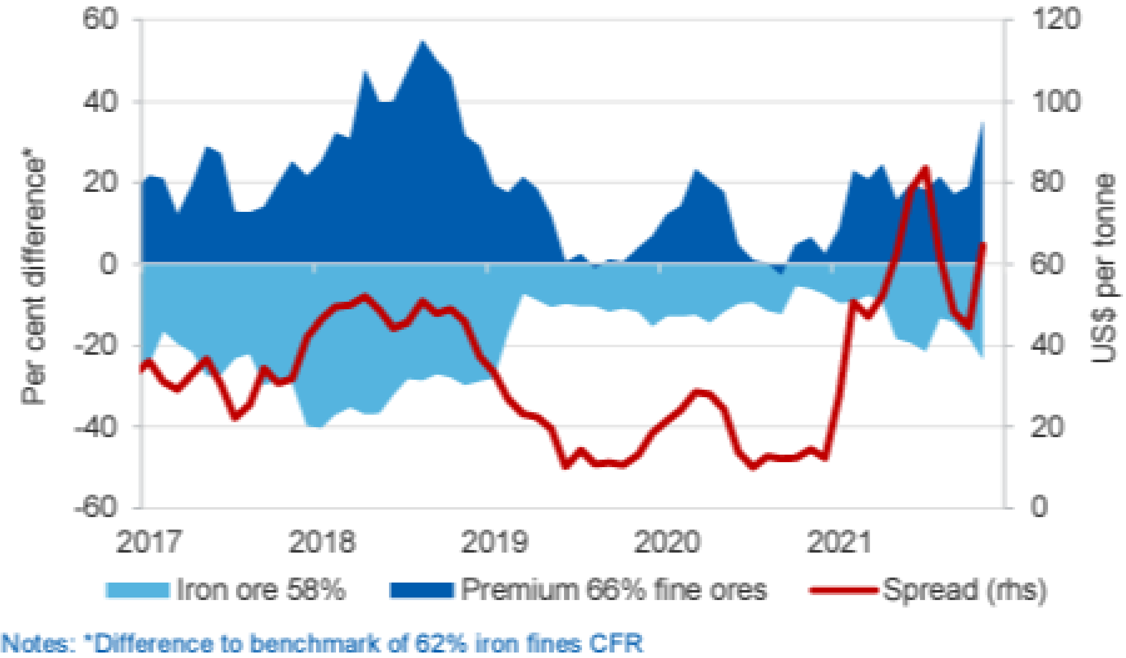


Figure 19-10: Iron ore prices and spread between grades

Source: Bloomberg (2021); China import prices

The forecast for a long-term transition over coming decades towards the production of 'green steel' suggests that high-grade iron ore products, especially high-grade and low impurity magnetite products like Macarthur's should enjoy sustainable demand into the future.

This demand narrative is further supported by recent advances in the development of 'green steel' manufacturing processes. Notably, in August 2020 a Swedish joint venture between SSAB, LKAB and Vattenfall announced the development of a new 'Hybrit' technology aimed at producing the world's first fossil free steel and in 2021 Hybrit delivered the world's first 'carbon free' steel to truck manufacturer AB Volvo in Sweden.

Consumer preferences are set to drive product demand and will inform the production choices that manufacturers make. For manufacturers that rely upon steel products, this will inevitably flow through to the products that bulk iron ore producers must mine and process. There is already a concerted move by big industrials to mandate 'green' product procurement supply chains, with audit trails being pushed back as far as possible.

Notably, large European car manufacturers such as Volvo and Mercedes Benz have confirmed that their procurement strategies are going to include green steel and they have signalled a move to carbon-free value chains across their businesses. Accordingly, procurement strategies aimed at reducing and preventing CO2 emissions instead of moving down the compensation path are expected to become more prevalent in the current policy and regulatory environment and this is expected to have an impact across the entire steel production value chain.

19.5.3 Iron ore supply

Figure 19-11 and Figure 19-12 below highlight projected global mined iron ore production to 2050 on both an aggregated and distributed basis for fines, lump and pellet feed.

The projections indicate that by 2050, global mined iron ore production for lumps, fines and pellet feed is expected to reach almost 2.6 billion tonnes.

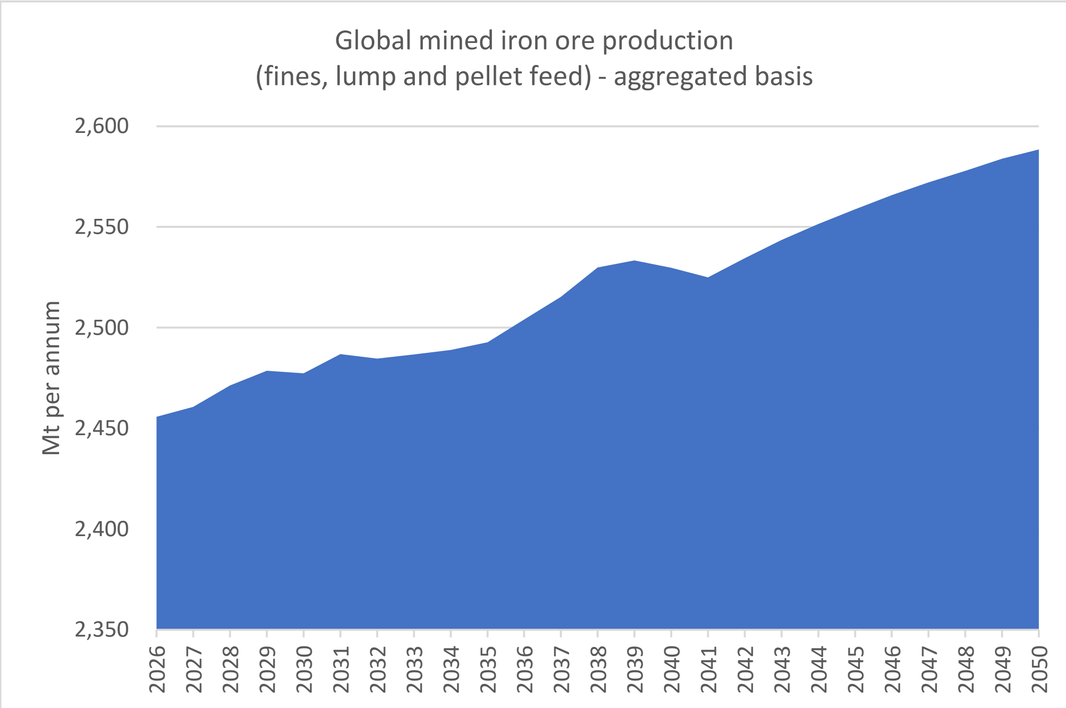


Figure 19-11: Global mined iron ore production (fines, lump and pellet feed)

Source: Wood Mackenzie data

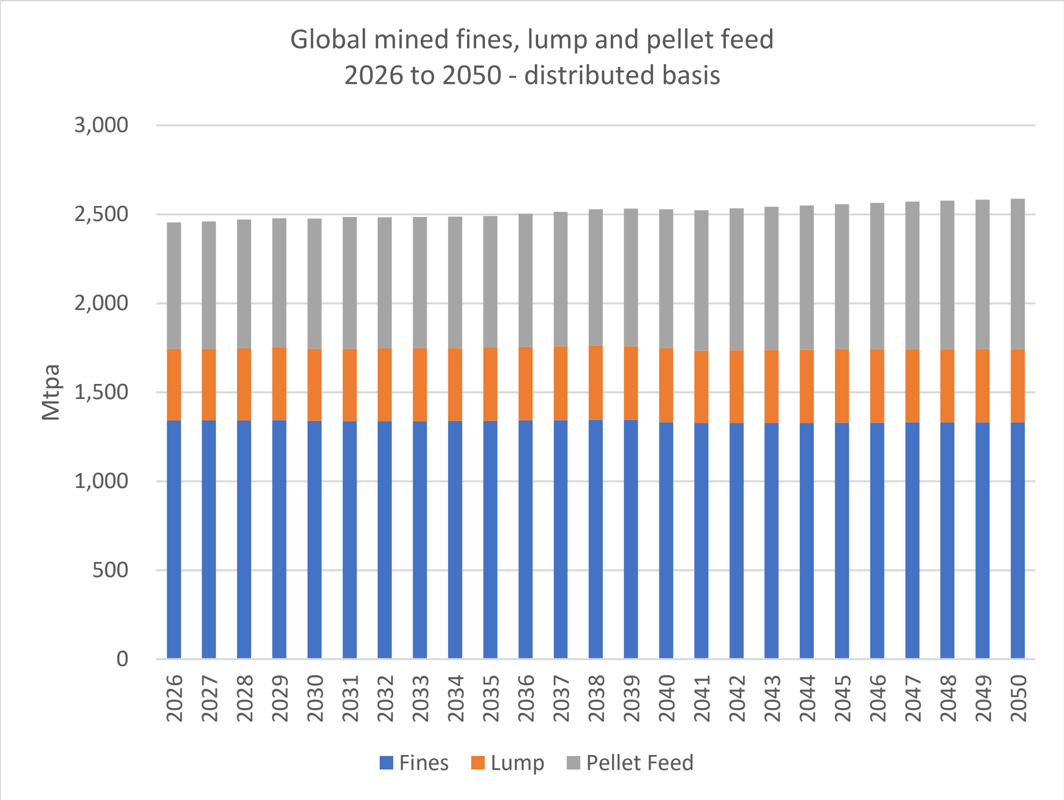


***Figure 19-12:* Global mined fines, lump and pellet feed 2026 to 2050**

Source: Wood Mackenzie data

Global iron ore exports are expected to moderate over the period to 2050. (See Table 19-2 below). However, a key factor will include whether future supplies may be affected by new mines being brought online by other countries.

In May 2021, the Chinese Government announced its objective to diversify that country's current iron ore supply. Australia currently accounts for more than 60% of the nation's iron ore imports. The Chinese Government's new plan included a target of 45% self-sufficiency in steelmaking raw materials by 2025; increased domestic exploration and output of iron ore; and securing more overseas reserves. As part of this, China is investigating a number of possible iron ore mines in Africa, including large deposits in Gabon and Madagascar.

The most notable prospect in Africa is the proposed Simandou iron ore mine, located in Guinea. The project has been increasingly emphasised as a key element in China's future supply chains, although production remains a number of years away. With potential full production capacity of 200 million tonnes per year, this is equal to around 15-20% of output currently produced in the Pilbara region of Western Australia. However, there are significant risks for this project to be brought into development. The project requires long term and significant investment in mining-related and transport infrastructure to get minerals to market including development of a new port and 650 kilometres of new railway. Additionally, during September 2021, a coup against President Alpha Condé demonstrated how exposed the project is to potential political instability in the country.

Table 19-2: Global Iron Ore Exports (Projections based on assumed iron ore supply, consumption, and trade flows)

Region	Iron Ore Exports (Mt) in Year					
	2026	2030	2035	2040	2045	2050
Asia	38	38	31	31	31	31
Europe	30	30	30	30	30	30
CIS	67	67	67	69	69	69
North America	79	78	73	71	71	72
South America	503	509	517	517	518	518
Africa	84	84	79	55	45	45
Oceania	943	924	922	919	918	916
Middle East	19	19	19	19	19	19
Global Iron Ore Exports	1,762	1,748	1,738	1,712	1,701	1,700

Data source: Wood Mackenzie

Notwithstanding these alternatives supply sources, the market structure is not expected to alter significantly, with Australia's market share expected to be maintained. A recovery in Brazilian supply is likely in the short-term, but a number of high-cost mines in Brazil and China are also expected to face closure or depletion over the next 10 years.

19.6 Lake Giles Market Positioning

19.6.1 Macarthur Concentrate Specification – March 2022

The Lake Giles Iron Project magnetite concentrate (Lake Giles Concentrate) will be a high-grade concentrate (66.1% Fe) that is expected to be attractive to blend at low levels into sinter feed, improving the sinter quality, or potentially for blast furnace pellet production.

The Lake Giles Concentrate is expected to have correspondingly low levels of silica and alumina (see Table 19-3 below). As a headline grade, the Lake Giles Concentrate product chemistry is expected to be generally consistent with Anglo's Minas Rio BF product (which grades between 66-67% Fe) and Champion Iron's Bloom Lake product (66.5% Fe), both of which have achieved substantial sales at prices that are considerably higher than the major fines brands.

Table 19-3: Lake Giles Concentrate Typical Specifications

Size	Fe	SiO_2	Al_2O_3	P	S	MnO	CaO	MgO	Na_2O	K_2O	TiO_2	LOI
P80 38 um	66.1	4.9	0.1	0.05	0.2 to 0.6	0.07	0.3	0.5	0.01	0.01	0.01	-2.7

A comparison of the Lake Giles Concentrate product chemistry against other seaborne traded iron ores is set out in Table 19-4 below.

Table 19-4: Comparison of Lake Giles Concentrate vs Seaborne Traded Ores

Ore Brand	Type	Mean Value (%)											
		FE	SIO2	AL203	P	S	Cu	Zn	Pb	Mn	K20	Na2O	TI02
LGIP - Magnetite	**PF**	**66.1**	**4.9**	**0.1**	**0.05**	**0.2 to 0.6**	**0.003**	**0.002**		**0.05**	**0.01**	**0.01**	**0.01**
Pfcj (carajas)	PF	65.02	2.72	1.52	0.065	-							
Atamaca	PF	66.02	4.65	1.52	0.006	-							
Lebedinski Gok	PF	68.07	4.75	0.14	0.009	-							
Pfft (tubarao)	PF	68.01	1.82	0.23	0.017	-							
Minas Rio	PF	67.30	2.56	0.45	0.026	-							
Peruvian Hg	PF	69.77	1.47	0.28	0.008	0.145	<0.03				<0.4		
Lkab	PF	70.50	1.50	0.40	0.020	0.005							
Robe river	LUMP	57.50	4.63	2.42	0.038	0.015							
Ssft (tubarao)	FINE	62.93	7.48	0.68	0.040	-							
Mac	FINE	60.54	4.93	2.27	0.083	0.026	0.0005	0.003	0.0006	0.25	0.01	0.01	0.01
Iocj (Carajas)	FINE	65.37	1.45	1.33	0.081	0.003							
Roy Hill	FINE	61.14	4.37	2.28	0.056	0.031							
Pilbara	FINEFINE	61.43	3.78	2.17	0.093	0.018							
Csn (Itaguai)	FINE	62.38	6.42	1.35	0.066	-							
Brfb (Dalian)	FINE	63.01	4.92	1.35	0.079	-							
Bhp Yandi	FINE	56.93	6.36	1.68	0.042	0.010	0.0005	0.002	0.0004	0.03	0.01	0.01	0.06
Hamersley Yandi	FINE	58.47	4.59	1.43	0.051	0.007	0.0005						
Robe River	FINE	56.45	5.50	2.96	0.037	0.017							
Mt. Newman	FINE	62.13	4.15	2.37	0.096	0.013	0.0010	0.006	0.0005	0.08	0.02	0.01	0.08
Marcona sinter feed	FINE	67.00	<4	<0.8	0.040	<1.5	<0.03				<0.4		
Champion Iron	CONC	66.46	4.42	0.21	0.011	-							
Arcelormittal	CONC	66.00	5.05	0.29	0.012	-							
Metinvest	CONC	67.25	5.88	0.16	0.008	-							

Data source: LFJ Consulting, BHP Billiton (Our Quality Story)

The sulphur content in the Lake Giles Concentrate product specification is currently set within a range of 0.2 to 0.6. At 0.6, the upper-level sulphur content of the Lake Giles magnetite concentrate is high when compared to other magnetite concentrate products, but it will have a market.

In determining this, the Macarthur surveyed several steel mills throughout the Asia region. Steel mills with good desulfurization facilities indicated an ability to use the iron ore with elevated sulphur levels.

However, as the cost of such desulfurization is relatively expensive some mills may be reluctant to accept ore with sulphur levels above 0.2 without further blending.

The Company has undertaken a QEMSCAN analysis to better understand the mineralogy and liberation sizes of sulphide and iron minerals. Preliminary reverse flotation test work to resolve the elevated sulphur levels in the process flow sheet was also undertaken. A further test work programme will be undertaken following the completion of this Feasibility Study to reduce sulphur levels.

Flotation of sulphide minerals is well understood in mineral processing and has been successfully demonstrated in other magnetite projects such as Grange Resources Southdown project.

19.6.2 Marketing of Lake Giles Concentrate

Currently, all future iron ore product produced by Macarthur from the Lake Giles Iron Project is expected to be traded by Glencore Plc under an existing binding Offtake Agreement.

Glencore markets iron ore products using in-house iron ore marketing expertise. Whilst Glencore does not currently export Macarthur iron ore products, Glencore has reviewed the product specification sheet developed for the Feasibility Study and the current relative value for Macarthur's magnetite concentrate product is understood.

The likely markets are anticipated to be Asian customers. Demand in this market is driven by internal consumption.

19.7 Iron Ore Prices

This section forms the basis for the iron ore pricing employed in financial modelling. The pricing scenario has been developed using assumptions of iron ore supply and demand, and assumptions about future steel demand which have been outlined in previous sections of this chapter, as well as assumptions about the Lake Giles product in the global iron ore market.

19.7.1 Price Drivers

Steel production is a key driver of iron ore pricing. China dominates the global steel production space, and its industrialisation programs over the last several decades have seen it emerge as the largest consumer of iron ore products globally. China produced over 1000Mt of steel in 2020 (Commonwealth Department of Industry, Innovation and Science, ‘Resources and Energy Quarterly’, March 2021). However future iron ore consumption growth is expected to be driven from countries such as India and other developing south-east Asian nations.

Prices for iron ore are driven by supply and demand factors in the global market, but it can also be influenced by factors affecting specific market segments and regions. Some examples of this include (but are not limited to):

- changes to government policies which can have a material impact, for example changes to policies on emissions standards generally, and adopted targets for achieving net zero carbon emissions;
- changes to cost structures within the industry in combination with market demand; and
- the prices for key inputs into the steel making process, such as the prices of coke and metallurgical coal.

All of these drivers can have an impact upon the value of different iron ore grades and the types of iron ore products.

19.7.2 62% Fe Fines Iron Ore Reference Price

The key reference price for internationally traded iron ore is the price of 62% Fe fines delivered to China on a CFR basis. CFR refers to ‘cost and freight’ and is a trade term that requires the seller to transport goods by sea to a required port. (Cost, insurance, and freight (CIF) is what a seller pays to cover the cost of shipping, as well as the insurance to protect against the potential damage of loss to a buyer's order).

The price forecast is based on the Company’s assessment of published consensus pricing, forecasts derived directly from steel mills, the various analyst reports referred to in this Chapter, and a comparison of historical analyst forecasts against actual pricing over time. The Company has then adopted an adjustment for grade using historical and projected premiums to arrive at a long-term price for the Lake Giles Concentrate.

19.7.3 Pricing period

The current project execution schedule for the Lake Giles Iron Project assumes a 2-year final project approvals period. A 24-to-28-month construction period will be required, with a 12 month ramp up to full production. The Company has adopted pricing assumptions from the commencement of commissioning over a 25-year life of mine (discussed at 19-9 below).

19.8 Iron Ore Pricing Evolution

Pricing mechanisms for iron ore have evolved over the last several decades. The iron ore industry has transitioned from an annually negotiated benchmark system to index linked pricing. Iron ore is priced based on the cost of iron units with an adjustment for the ore’s specific properties, impurities, and other steelmaking related characteristics.

The value of iron ore is therefore dependent upon the steel products that are being produced by individual steel mills, the specifications required by each steel mill based around how the iron ore product will perform in the steel mill (in terms of the impact on productivity, energy consumption and emissions) and the final quality of the steel product that is being manufactured.

Iron ore is a non-fungible commodity, and its quality varies. To assist the facilitation of price adjustments for differences between expected and delivered product specifications, "value-in-use" indices are therefore used to adjust the value of iron ore products for key price-affecting chemical components. These adjustments then translate through to either a discount or a premium being applied to the product received. Adjustments are also often made for the physical form of the iron ore products (eg., lumps, fines and pellets).

19.8.1 Benchmark Pricing

Benchmark pricing was adopted for iron ore under long-term volume contracts from the 1970's right through to 2010, with annual price renegotiations determining prices between iron ore suppliers and steel mills. However, that system began to break down in response to the increase in steel output from China during the early 2000's. China's share of global demand for iron ore surged and supplies were unable to be maintained. The consequence of this was the gradual evolution of a spot market outside of the benchmark system that had operated until that point. The large iron ore producers allowed long-term volume contracts to expire and new contracts with more frequent adjustments to the new published index soon became the norm.

19.8.2 "Benchmark" or "Reference" Iron Ore Price

The "benchmark" or "reference" price for internationally traded iron ore is for 62% Fe fines CFR China. A number of different price reporting organisations now track spot prices for iron ore and compile them into indices for a range of iron ore products.

The most commonly referenced index for 62% Fe fines is the Platts' IODEX 62% Fe CFR China index. To determine the reference price, Platts runs an assessment of iron ore trading transactions for fines grading between 60% Fe and 63.5% Fe and normalises that price information to establish a base standard specification for 62% Fe fines products. The quality specifications for the Platts IODEX 62% Fe CFR China index are set out below in Table 19-5.

Table 19-5: Quality specifications – Platts IODEX 62% Fe CFR China

Content	Fe	Silica	Alumina	Phos.	Sulphur	Moisture
%	62	4.0	2.25	0.095	0.02	8

Source: Platts

19.8.3 Demand and Pricing for High Grade Iron Ore (Fines and Concentrates)

Other indices exist in addition to the "benchmark" 62% Fe fines index. These include (but are not limited to) both the 58% Fe and 65% Fe indices. The quality specifications for the Platts 65% Fe CFR China index are set out below in Table 19-6.

Table 19-6: Quality specifications – Platts 65% Fe CFR China

Content	Fe	Silica	Alumina	Phos.	Sulphur	Moisture
%	65	3.5	1.0	0.075	-	8.5

Source: Platts

19.9 Price Premiums & Concentrate Price Forecast

19.9.1 Lake Giles Iron Project Price Estimate – CFR China Basis (MIO/MMS)

The pricing mechanism for the study is based on the 62% Fe fines index. Realised pricing has been adjusted for a nominal iron grade of 66.1% on a dmt basis. FOB pricing has been adjusted for sea freight from the Port of Esperance, Western Australia to Qingdao, China.

The 15-year (2007 to 2022) average for 62% Fe CFR China is US$110/dmt. However, for the purposes of this Study, The Company has assumed that the 62% Fe index will trend close to the 10-year average to December 2021 over the long term. The 10-year average iron ore price (January 2011 to December 2021) was US$103.86 CFR for 62% Fe fines. (See Figure 19-13 below). This is also consistent with the average pricing forecasts produced by Wood Mackenzie for the period of planned mining operations.

Figure 19-13: 10-year average price for 62% Fe Fines CFR China

Source: Trading Economics

It has been assumed that the long-term transition towards lower emissions and decarbonised steel will result in the average price spread between for 62% Fe CFR China and 65% Fe CFR fines products widening beyond 2022. A premium of 25% to the 62% Fe reference price has therefore been adopted for 65% Fe fines for the purposes of this Study, against which a further grade and magnetite premium has been applied to the Lake Giles Concentrate.

The Feasibility Study financial model uses a base price assumption of US$99/dmt, CFR China 62% Fe Fines. An overall price premium to the 62% Fe price of US$32.40/dmt is used for the Lake Giles concentrate. This equates to a nominal US$8.10/dmt, which is inclusive of both a grade premium and a magnetite premium.

The actual pricing for Lake Giles Concentrate will depend upon a range of factors in addition to Fe content. The final target concentrate will be a fine grind (at P80 38 micron), but this is not unusually fine when compared to other magnetite fines products currently in the market. Phosphorous levels will also be low, which may be an advantage due to phosphorous levels in the Pilbara increasing on trend.

Based upon the above assumptions and analysis, a long-term sales price of US$131.40/dmt for Macarthur's 66.1% Fe concentrate product specification has been adopted. This is expected to result a realised free on board (FOB) sales price of USD$120.30/dmt after shipping and marketing costs.

20 Environmental Studies, Permitting and Social or Community Impact

Environmental approval of the project will be required from various Decision-Making Authorities (DMAs) of the Western Australian and Australian governments under various pieces of environmental legislation before the project can be implemented. To achieve these approvals, the Company is required to conduct an Environmental Impact Assessment (EIA) of the project area.

20.1. Current Approval Status

The Company holds programme of works (PoW) approvals from DMIRS that provide consent for exploration drilling across the Moonshine and Moonshine North deposits.

Mining approval and construction of supporting infrastructure requires a number of approvals that the Company will need to obtain prior to development of the Project. This section outlines the regulatory framework and key activities to secure environmental approval.

20.2. Environmental Decision-Making Authorities

The Department of Mines, Industry Regulation and Safety (DMIRS) is the WA Government body that administers the Mining Act 1978 and is the lead agency for the regulation of mining activities in WA. Approvals and advice provided by the DMP include:

- tenure for exploration and mining projects
- environmental approvals
- petroleum pipeline licences
- facilitation of native title agreements
- occupational safety and health; and
- dangerous goods.

Other departmental roles include:

- **Environmental Protection Authority (EPA)** – assess and provide public advice on proposals likely to have a significant effect on the environment and develop statutory policy and advice to protect the environment.
- **Department of Water and Environment Regulation (DWER)** – regulate pollution and clearing of native vegetation; manage and regulate CALM Act, lands and waters and provide advice on biodiversity, wetlands, contamination, pollution and waste, and environmental harm; water licensing.
- **Department of Aboriginal Affairs (DAA)** – assessment and advice on proposals likely to have an impact on Aboriginal heritage; assessment and advice on access to and use of lands held by the Aboriginal Lands Trust and develop administrative policy and advice to protect Aboriginal heritage and manage lands held by the Aboriginal Lands Trust.
- **Department of Health (DoH)** – provide advice and guidelines on acceptable use and background levels of hazardous substances, provide permits to use some substances and regulation of Health Act, 1911.
- **Local Government** – Building Approvals.
- **Commonwealth Department of the Agriculture, Water and the Environment (DAWE)** – Controlled actions under the EPBC Act ; and

- **Department of Transport** – integrated transport planning that arises from the aims of land use planning; ensure all aspects of intermodal transport are taken into consideration; evaluating the transport economics of different transport solutions.

20.3. Primary approvals

20.3.1. Environment Protection and Biodiversity Conservation Act 1999

The Federal Environment Protection and Biodiversity Conservation Act 1999 (EPBC Act) provides a legal framework to protect and manage nationally and internationally important flora, fauna, ecological communities, and heritage places, as defined within the EPBC Act as Matters of National Environmental Significance (MNES). Any project that has, will have, or is likely to have a significant impact on MNES requires approval from the Australian Government Environment Minister through submission of a referral to the Department of Agriculture, Water, and the Environment (DAWE).

The MNES that may be relevant to the Project are:

- World and national heritage properties
- Nationally threated species and ecological communities; and
- Migratory species.

A proponent shall undertake a self-assessment to conclude whether the project is likely to have a significant impact on MNES, in accordance with their guidance document 'Matters of National Environmental Significance: Significant Impact Guidelines 1.1' (DOE 2013). Within 20 business days, the environment minister will make one of the following decisions:

1. Controlled action – the project is subject to the assessment and approval process under the EPBC Act
2. Not a controlled action 'Particular Manner' – the project does not require approval if it is undertaken in accordance with the manner specified; and
3. Not a controlled action – the project does not require approval if undertaken in accordance with the referral.

Where projects are given a 'controlled action' decision, the bilateral agreement with the Western Australian EPA allows the EPA to assess the project on behalf of DAWE, preventing the duplication of assessment processes. Following the review of the EPAs assessment report, DAWE will make their own approval decision issued under the EPBC Act.

Timeframes of this approval process is therefore limited to the timeframes for the EPA assessment processes under the Western Australian Environmental Protection Act 1986 (EP Act).

20.4. Environmental Protection Act 1986 – Part IV

The EP Act is the primary legalisation for the management of environmental values in Western Australia. Specifically, Part IV of the EP Act establishes the provisions for the EPA to carry out environmental impact assessment for Projects developed in the State.

As required under the 'Environmental Impact Assessment Procedures Manual' (EPA 2020a), a project is required to be referred to the EPA for assessment if it is considered either a significant proposal or a strategic proposal, that may include plans for a future staged development. A proposal may be deemed significant based on consideration of the:

1. Values, sensitivity, and quality of the environmental which is likely to be impacted
2. Extent (intensity, duration, magnitude, and geographic footprint) of the likely impacts
3. Consequence of the likely impacts (or change)
4. Resilience of the environment to cope with the impacts (or change)
5. Cumulative impacts with other existing or reasonably foreseeable activities, developments, and land uses

6. Connections and interactions between parts of the environment to inform a holistic view of impacts to the whole environment
7. Level of confidence in the prediction of impacts and the success of proposed mitigation; and
8. Public interest about the likely effects of the proposal or scheme, if implemented, on the environment, and public information that informs the EPAs assessment.

Should the proponent consider a project to be significant or strategic, it should be referred to the EPA for assessment. Third parties (e.g., Decision-Making Authority, stakeholder, or member of the community) can also refer a project to the EPA, or it can be called-in by the EPA itself.

20.5. Referral of a Project

Once a project has been referred, the EPA uses environmental principles, factors, and associated objectives as the basis for assessing whether the project's impact on the environment is acceptable. The environmental factors and objectives that unpin the environmental impact assessment process are summarised in Table 20-1, as defined in the 'Statement of Environmental Principles, Factors and Objectives' (EPA 2020b).

Based on the referral information provided by the proponent and following a seven-day public comment period, the EPA is afforded a 28-business day statutory timeframe to determine whether the project requires assessment under the EP Act (subject to the provision of adequate information by the proponent.

Table 20-1: Relevant Environmental Factors and Objectives for Environmental Impact Assessment in WA

Theme	Factor	Objective
Land	Flora and Vegetation	To protect flora and vegetation so that biological diversity and ecological integrity are maintained.
	Landforms	To maintain the variety and integrity of distinctive physical landforms so that environmental values are protected.
	Subterranean Fauna	To protect subterranean fauna so that biological diversity and ecological integrity are maintained.
	Terrestrial Environmental Quality	To maintain the quality of land and soils so that environmental values are protected.
	Terrestrial Fauna	To protect terrestrial fauna so that biological diversity and ecological integrity are maintained.
Water	Inland Waters	To maintain the hydrological regimes and quality of groundwater and surface water so that environmental values are protected.
Air	Air Quality	To maintain air quality and minimise emissions so that environmental values are protected.
	Greenhouse Gas Emissions	To reduce net greenhouse gas emissions to minimise the risk of environmental harm associated with climate change.
People	Social Surroundings	To protect social surroundings from significant harm.
	Human Health	To protect human health from significant harm.

20.6. Decision to Assess

Based on referral information, where the EPA decides to assess a project, it may do so under the following pathways:

1. Referral information – where the EPA determines that it has enough information to assess the proposal from the referral information
2. Environmental review (no public review) – where the EPA determine that an environmental review is required, but the Environmental Review Document (ERD) is not made public
3. Public environmental review – where the EPA determines that an environmental review is required, and the ERD is to be made available for public review; and
4. Technical report and peer review – where the EPA determines that issues relating to one preliminary key environmental factor requires a technical report and an independent peer review of that report for its assessment, rather that the proponent undertaking a full environmental review. The EPA may also determine that this technical report and peer review should be made available for public review.

Environmental reviews will also typically require the EPA or the proponent to prepare an Environmental Scoping Document (ESD), including a two-week public review period, that outlines the content, timing, and procedures for the environmental review for that specific project.

In general, and where assessed by the EPA, significant mining operations in WA may require environmental review under either pathway ii) ERD with no public review, or iii) public environmental review.

20.7. Assessment Decision

Following the preparation and review of the environmental review process, the EPA will prepare an assessment report in which the EPA considers all relevant information received for the project, including:

- key environmental factors
- environmental management plans, existing and/or required plans
- project offsets
- advice from the Commonwealth on MNES; and
- advice from any other decision-making authorities and/or government agencies.

The EPA uses this information to decide whether they support the implementation of the proposal, and if so, recommends specific conditions which must be met by the proponent. This assessment report is provided to the Environment Minister for consideration and the final decision on the implementation of a proposal is issued by the Minister in the form of a Ministerial Statement. This ‘approval’ is granted after the Minister consults with other relevant Ministerial portfolios.

20.8. Mining Act

The Mining Act 1978 (Mining Act) is the principal mining legislation in WA and is administered by the Department of Mines, Industry Regulation and Safety (DMIRS). Proponents must submit Mining Proposals (MPs) and Mine Closure Plans (MCPs) to DMIRS for assessment against environmental and social related impacts relevant to the Project.

These plans are typically assessed within 60 business days and can be submitted in parallel with other approvals. However, DMIRS may reserve their final decision until relevant tenure is granted or until assessments under Part IV of the EP Act and the AH Act have been completed.

Under the 2020 Guidelines for MPs (DMIRS 2020), these applications must be submitted with a commensurate MCP (which also need to align with the DMIRS Guidelines).

20.9. Secondary Approvals

Mining activity in WA is subject to several legislative requirements outside of Section 38c of the EP Act and the EPBC Act. The following sections describe the other major regulatory approvals that may be required for mining projects in WA.

These approvals will typically be submitted in parallel or following decisions of the primary approvals.

20.10. EP Act – Part V

Part V of the EP Act and its associated Regulations (GWA 2020) list the Prescribed Premises which have the potential to pollute air, land or water through discharges and emissions and managed by the Department of Water and Environmental Regulation (DWER). Works Approvals are typically required to manage the construction of Prescribed Premises and Licences are required to manage their operation.

The Project is likely to involve several Prescribed Premises that will require Works Approvals and Licences:

- Sewage facility (≥ 100 cubic metres (m3) per year
- Category II or III putrescible landfill site (≥ 20 tonnes per year)
- Electric power generation (≥10 megawatts per year, using diesel)
- Processing or beneficiation of metallic or non-metallic ore (≥50,000 tonnes per year); and
- Mine dewatering (≥50,000 tonnes per year).

These approvals have a target timeframe of 60 working days and must be received prior to the commencement of construction and/or commissioning.

20.11. Water

The DWER also administers the Rights in Water and Irrigation 1914 Act (RIWI Act) which was developed to manage the State's water resources. Proponents require licences and permits for the extraction of groundwater resources to supply water for mining related activities, including both construction, operation, and processing.

There are two main licences a proponent must obtain through DWER:

- 26D Licence to construct an artesian well; and
- 5C Licence to take water from an underground source.

20.12. Heritage

The Aboriginal Heritage Act 1972 (AH Act) was established to protect and manage places of significance to Aboriginal people of Australia. Where an Aboriginal site is identified, it is required to be reported for assessment and determination by the Minister for Aboriginal Affairs as to whether it is recognised as a site that should be recorded and preserved and added to the Register of Places and Objects. This decision is based on the advice of the Aboriginal Cultural Material Committee (ACMC) managed by the Department of Planning, Lands and Heritage (DPLH).

Where an Aboriginal site is identified within a project area and the proponent concludes that impact to a site in unavoidable, approval must be sought under Section 18 of the AH Act through the ACMC to remove or disturb the site.

One Aboriginal site is currently known to exist within the Project area and further work may be required to determine its archaeological or ethnographic significance. Further heritage surveys may also be required over areas not previously investigated. Some tenure is yet to approved by DMIRS and may require consultation with relevant heritage groups.

20.13. Native Title

The Commonwealth Native Title Act 1993 (NT Act) aims to provide recognition and protection of native title and establishes a mechanism for determining claims of native title and ways in which future dealings affecting native title may proceed. The National Native Title Tribunal (NNTT) is an independent body created to assist claimants of native title applications manage their claims made to the Federal Court, as well as maintaining a national Native Title Register.

Registered Native Title claimants and determined Native Title holders have certain rights under the provisions of the NT Act, when governments intend to conduct business considered future acts, such as the granting of mineral tenure. Any holder or claimant for Native Title has the right to be notified of any mining tenure application and mining lease applications can be made through DMIRS in parallel with Native Title processes, where appropriate. In most cases, mining lease applications cannot be approved until NH Act processes are satisfied.

The granted mining leases of the Project were granted prior to registration of the current native title claim and are therefore not subject to heritage agreements. The Company requires additional tenure for infrastructure areas which will require consultation with the native title claimants.

20.14. Environmental Impact Assessment

20.14.1. Environmental Factors

Based on the EPA approval process, several environmental factors are required to be assessed by the Proponent to determine if the Project poses significant risks to each relevant factor. The assessment of these factors will provide the basis for the overall environmental impact assessment process.

The environmental factors that are considered relevant to the Project are:

- Land Themes:
  - Flora and vegetation
  - Landforms
  - Subterranean Fauna
  - Terrestrial Environmental Quality (land and soils); and
  - Terrestrial Fauna.
- Water Theme:
  - Inland Waters
- Air Themes:
  - Air Quality; and
  - Greenhouse Gas Emissions.
- People Themes:
  - Social Surroundings; and
  - Human Health.

20.15. Impact Assessment and Baseline Surveys

To adequately assess any potential impacts the Project may present to each environmental factor, a formal study is typically required to be undertaken by industry consultants. The type and/or level of assessment required for each environmental factor can depend on several elements and these are prescribed by the EPA in their Technical Guidance documents.

For the purpose of defining specific EIA surveys the Project has been split into four project areas based on differing land uses or disturbances:

- Mine Site Area (MSA; including mine pits, tailings, and waste dumps, mine operations camp, air strip)
- Rail Siding Area (RSA; including stockpiles, rail loading infrastructure and supporting infrastructure)
- Haul Road Corridor (HRC; extending between the MSA and the Rail Siding Area; and
- Groundwater Bore field & Pipelines Corridor (GBPC, including bore field areas, access tracks, pipelines and supporting infrastructure).

Table 20-2 summarises the environmental factors identified for the Project and the proposed studies required to assess the significance of the Project on the existing environmental values.

Table 20-2: Proposed Environmental Factors and Impact Assessment Studies

EPA Factor	EPA Objective	Relevant Project Areas	Potential studies required to support assessment of factors	Risk Level	Survey Recommendations
Land Themes					
Flora and Vegetation	To protect flora and vegetation so that biological diversity and ecological integrity maintained	MSA	Detailed survey of a 'Development envelope'. Potential for targeted surveys for conservation significant species. Surveys across two seasons: primary – Spring secondary – Winter	**Significant**	Detailed survey recommended due to known EPA/DCBA interest in BIF, proximity to Nature Reserves and other conservation estate and the potential for conservation significant flora and vegetation communities to be present. Targeted surveys may also be required in the event that conservation significant species are identified in Project "impact" areas and cannot be avoided. EPA Guidance for the bioregion requires a two-season survey to adequately identify all flowering species. As there is potential for DRF in the area, it will also reduce uncertainty in flora identification. Avoidance management measures is also less practicable for this disturbance type, where certain footprints cannot be relocated (i.e., mine pits).
		RSA	Reconnaissance survey and targeted search: • primary – Spring • secondary – Winter	**Low**	It is recommended that a reconnaissance survey be undertaken to assess and characterise the existing condition and perform a targeted search of known, local conservation significant flora and vegetation. The Project area is not located on areas of BIF and the proposed disturbance at the RSA is unlikely to have significant impacts that cannot be managed. Due to the size of the disturbance area and nature of disturbance, there may be a case to perform a single survey to adequately assess this area, however it is recommended for planning purposes to factor a dual season at this stage. Management measures to avoid and/or minimise conservation significant species is more likely to be possible.

EPA Factor	EPA Objective	Relevant Project Areas	Potential studies required to support assessment of factors	Risk Level	Survey Recommendations
Flora and Vegetation (cont.)	To protect flora and vegetation so that biological diversity and ecological Integrity is maintained	HRC GBPC	Reconnaissance survey and targeted search. Surveys across two seasons: • primary – Spring secondary – Winter	**Moderate**	It is recommended that a reconnaissance survey be undertaken to assess and characterise the existing vegetation and condition and perform a targeted search of conservation significant flora and vegetation. This can inform the final design of infrastructure, so that it can avoid and/or minimise impacts to conservation significant species and communities. Theses Project areas are not located on areas of BIF and due to the nature of infrastructure at these Project locations, it is unlikely that the Project will pose significant impacts that cannot be managed. Due to the nature of infrastructure at these Project locations (i.e., linear infrastructure), avoidance of conservation significant species may be a practical management approach, particularly for borrow pits. Based on the location of the Project areas (bioregion) and given that much of these areas have little or no existing survey data, two seasons may be required to minimise uncertainty in identification of conservation significant species.
Landforms	To maintain the variety and integrity of distinctive physical landforms so that environmental	MSA RSA	Landform Impact Assessment	**Significant**	Landform impact assessment is recommended to provide a comparative analysis of the Moonshine landform against other BIF landforms in the region and to consider the visual amenity of the region due to proximity to a Nature Reserve and other conservation and recreational lands.
			Mine Waste Landform Design	**Moderate**	Waste rock landforms will be required to be designed with the aim to maintain the pre- disturbance visual amenity values of the MSA and surrounding region.

EPA Factor	EPA Objective	Relevant Project Areas	Potential studies required to support assessment of factors	Risk Level	Survey Recommendations
	values are protected	HRC GBPC	Landform Impact Assessment	**Low**	Due the nature of disturbance in these Project areas (linear, low-lying infrastructure) the impact to visual amenity of these areas is likely to be low. However, given the proximity to a Nature Reserve, conservation and recreational lands, the impacts should be considered.
Subterranean Fauna	To protect subterranean fauna so that biological diversity and ecological integrity are maintained	MSA GBPC	Troglofauna Level 2 Survey	**Moderate**	BIF regions are considered highly likely to support troglofaunal communities. Proposed activities in these Project areas may impact on troglofauna habitat and as such, as Level 2 survey is recommended to adequately assess the presence of troglofaunal and the significance of species and/or habitats.
		MSA GBPC	Stygofauna – Desktop and Level 1 survey, if required	**Low**	BIF regions may provide suitable habitat for stygofauna communities however, the Project groundwater is likely to be saline to hypersaline and therefore unlikely to sustain populations. It is recommended that a desktop study be completed initially once the location of the bore field has been determined and pit design is completed. The outcomes of the desktop survey may conclude whether sampling events are required.
		HRC RSA	No subterranean surveys required.	**None**	No survey requirements for subterranean fauna at these Project areas as disturbance activities is predominately above ground activities (except for shallow trenching or borrow pits) and therefore will not significantly impact on local troglofauna or stygofauna communities.
Terrestrial Environmental Quality	To maintain the quality of land and soils so that environmental values are	MSA	Soils characterisation	**Moderate**	Soils will be required to be characterised to identify potential contaminants based on their geochemical and physical characteristics, as well as to facilitate design and stability of mine pits, waste rock and tailings landforms, surface hydrology and mine rehabilitation. This work will also inform mine closure aspects.

EPA Factor	EPA Objective	Relevant Project Areas	Potential studies required to support assessment of factors	Risk Level	Survey Recommendations
	protected	RSA HRC GBPC	Soils characterisation	**Low**	Soils will be required to be characterised to identify potential contaminants based on their geochemical and physical characteristics, as well as to facilitate design elements for construction activities (i.e., use for fill material or borrow pits, erosion control, permeability, use for rehabilitation).
		MSA	Mine Waste characterisation	**Moderate**	Waste rock and tailings will be required to be characterised to identify potential contaminants based on their geochemical and physical characteristics. Soils and waste also need to be assessed to ensure they are suitable for stockpiling and that final landforms can be constructed to be safe, stable, and non-polluting. This work will also inform mine closure aspects.
		MSA	Mine Closure and Rehabilitation	**Moderate**	Outcomes of soil and waste characterisation studies will be utilised to inform mine closure and rehabilitation aspects, including the design and management of final post-mining landform designs and site-specific rehabilitation criteria.
Terrestrial Fauna	To protect terrestrial fauna so that biological diversity and ecological integrity are maintained	MSA	Detailed survey targeting vertebrate fauna and short-range endemics (SREs) Surveys across two seasons: • primary – Spring • secondary – Winter	**Significant**	Level 2 surveys recommended due to known EPA/DCBA interest on BIF, proximity to Nature Reserves and other conservation estate and the potential for conservation significant fauna to occupy the area, such as the EPBC Listed Threatened Malleefowl. Avoidance management measures is also less practicable for this disturbance type, such as location of mine pits. EPA Guidance for the bioregion requires a two-season survey to adequately identify all species in their seasons of maximum activity. As there is potential for Threatened fauna in the area, undertaking two season survey will also reduce uncertainty of missing the timeframes of maximum activity.

EPA Factor	EPA Objective	Relevant Project Areas	Potential studies required to support assessment of factors	Risk Level	Survey Recommendations
		RSA HRC GBPC	Basic & targeted survey, targeting vertebrate fauna and short-range endemics (SREs) Single season (may be dependent on desktop analysis of potential species)	**Moderate**	A basic survey (lower-intensity) may be possible at these Project areas as it is not located on areas of BIF and the lower-impact nature of the disturbance activities. The basic survey can gather information on broad fauna and habitat types and inform further targeted surveys to identify evidence or likely habitat of conservation significant species. Further consultation with DCBA and/or EPA may be required to confirm if this is a viable option. Due to the nature of infrastructure at these Project locations (i.e., lower-impact, linear infrastructure), avoidance of conservation significant species or habitats is more likely to be possible.
Water Themes					
Inland Water	To maintain the hydrological regimes and quality of groundwater and surface water so that environmental values are protected	MSA RSA HRC GBPC	Surface water hydrology	**Moderate**	A surface water assessment is recommended to identify existing hydrological regimes of the Project area and inform surface water design for the Project.
		MSA	Pit lake modelling	**Moderate**	Pit lake modelling study is recommended to determine the mine closure aspects of the final mine pit voids.
		MSA GBPC	Groundwater	**Moderate**	A groundwater assessment is recommended in the MSA to identify existing quality and quantity of groundwater beneath the Project area and inform groundwater abstraction requirements, dewatering requirements, potential hydrogeological impacts both onsite and to surrounding users or dependent ecosystems and mine closure aspects for the Project.
Air Themes					

EPA Factor	EPA Objective	Relevant Project Areas	Potential studies required to support assessment of factors	Risk Level	Survey Recommendations
Air Quality	To maintain air quality and minimise emissions so that environmental values are protected	MSA RSA HRC GBPC	Air quality assessment	**Low**	It is recommended that an air quality assessment be undertaken to assess the background air quality at the Project areas and the likely Project sources that may reduce the air quality and amenity of the Project and surrounding areas, such as the Nature Reserve and other conservation lands and recreational users of the region. This assessment would focus on the chemical, physical and biological characteristics of the air. This assessment would also inform on social surrounding factors (air quality) and people factors (human health) factors.
Greenhouse Gas Emissions	To reduce net Greenhouse gas emissions to minimise the risk of environmental harm associated with climate change	MSA	Carbon assessment	**Moderate**	MMS has advised that it is likely activities in the MSA will exceed the EPAs annual carbon emissions trigger level (100,000 tonnes carbon dioxide (CO2)) to consider greenhouse gas emissions. Therefore, it is recommended that a comprehensive carbon assessment is undertaken for the Project to estimate potential emissions over the life of the Project. The carbon assessment will also be used to inform a greenhouse gas management plan for the Project.
People Themes					
Social Surroundings	To protect social surroundings from significant harm	MSA RSA HRC GBPC	Air quality assessment – dust	**Low**	It is recommended that an air quality assessment be undertaken to assess the background air quality at the Project areas and the likely Project sources that may reduce the air quality and amenity of the Project and surrounding areas, such as the Nature Reserve and other conservation lands and recreational users of the region. This aspect of the assessment would focus on the existing aesthetic quality and the potential impacts to visual amenity relating to dust generated by the Project.

EPA Factor	EPA Objective	Relevant Project Areas	Potential studies required to support assessment of factors	Risk Level	Survey Recommendations
					This assessment also informs on air factors (environmental air quality) and people factors (human health) factors.
			Light, noise & vibration assessment	Low	It is recommended that an assessment of light, noise and vibration is undertaken to assess the background levels of these factors and identify the likely Project sources that may reduce their amenity and impact on existing social values (i.e., recreation and tourism). This assessment may also require further consideration of conservation significant fauna that are sensitive to changes to noise, light, or vibration, as informed by the Terrestrial Fauna studies.
			Aboriginal Heritage surveys – ethnographic and archaeological factors	Moderate	Aboriginal heritage surveys, focusing on the presence of ethnographic and archaeological sites will be required to enable the Proponent to avoid disturbance to these areas, or if required, seek appropriate approval to disturb sites.
			Native Title assessment		Native title should also be considered to ensure that all relevant Traditional Owners are consulted on the Project.
			Social impact assessment	Low	A social impact assessment may be required to assess the potential impacts the Project poses to existing social factors such as visual

EPA Factor	EPA Objective	Relevant Project Areas	Potential studies required to support assessment of factors	Risk Level	Survey Recommendations
					amenity, cultural, social, and economic values.
Human Health	To protect human health from significant harm	MSA RSA HRC GBPC	Air quality dust, asbestos	**Low**	It is recommended that an air quality assessment be undertaken to assess the background air quality at the Project areas and the likely Project sources that may reduce the air quality and amenity of the Project and surrounding areas, such as the Nature Reserve and other conservation lands and recreational users of the region. This aspect of the assessment would focus on the potential impacts to air quality that may result in significant harm to human health. This assessment also informs on air factors (environmental air quality) and social factors (amenity) factors.
			Assessment of noise & vibration	**Low**	It is recommended that an assessment of noise and vibration is undertaken to assess the background levels of these factors and identify the likely Project sources that may reduce their amenity and impact on human health (i.e., Project workforce or recreational / tourists in surrounding areas). This assessment may also require further consideration of conservation significant fauna that are sensitive to changes to noise, light, or vibration, as informed by the Terrestrial Fauna studies.

20.16. Approvals Pathway

The following sections outline the overall approvals pathway recommended for the Project. A visual approvals schedule outlining key milestones and forecasted assessment timeframes, is provided as Figure 20.1.

20.16.1. Federal

It is likely that MNES will occur in the Project area, based on the review of other key projects in the region, and therefore it is recommended that MMS refer this project to the DAWE under the EPBC Act 1999 to determine whether it is a controlled action.

It is expected that the DWAE will determine the Project a 'controlled action' and as such, the WA bilateral agreement may be enacted to allow the EPA to assess the project on behalf of DWAE regarding MNES.

Due to the longer assessment timeframes of the EPA process, it is recommended that the Project is referred early, and prior to the referral to the EPA, to ensure that the assessment can be captured in the EPA process, if required.

20.16.2. State

The Project's proposed activities will likely trigger several the EPA's 'significance' tests to determine whether proposals should be referred. It is recommended that the Project be referred to the EPA.

Prior to referral, the Company will need to assess all environmental factors relevant to the Project through the completion of environmental baseline surveys. The overall timeframe for the submission of referral will depend on the scheduling and outcomes of environmental surveys.

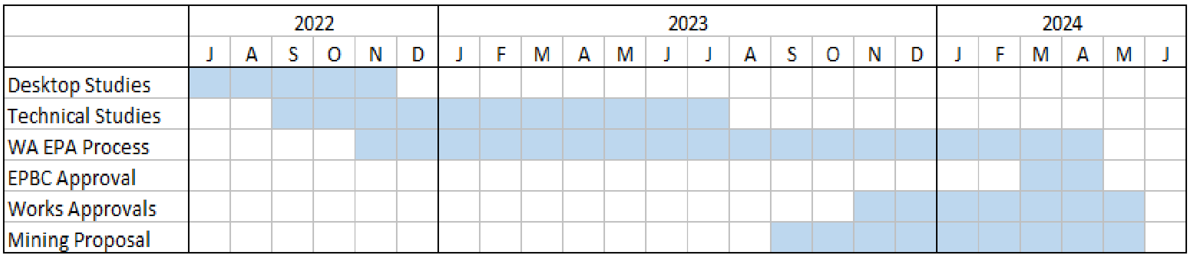
Based on the review of other key projects in the area and the EPA's conservative position on mining activities in BIF environments, it is considered likely that the Project will be assessed under the 'public environmental review' pathway. The implications of this process relate predominately to the extended timeframes required throughout the process to allow for mandatory public review periods.

20.17. Project Approval Timeframe

The forecasted approvals timeframe for the Project has been summarised in Table 20-3 below. This diagram outlines the major project milestones and forecasts assessment timeframes for each of the primary environmental approvals, as well as the Part V requirements under the EP Act, required for the Project.

The forecast has been developed assuming a start date in July 2022. This allows the Company to be able to capture the spring survey season this year, as required for a number of its technical baseline studies.

Table 20-3: Environmental Approvals Schedule



20.18. Rehabilitation and Mine Closure

As outlined in section 20.4.2 a mine closure plan is required to be submitted to DMIRS in concert with a Mining Proposal. The plan is approved prior to mining commencement and requires continual update as new information is obtained, areas are rehabilitated, or mining plans are varied. The MCP is to detail:

- Post mining land use

- Closure outcomes and completion criteria
- Closure monitoring and maintenance
- Closure risk assessment; and
- Financial provisioning for closure.

An assessment of proposed mine closure costs has been developed for the Project using the rehabilitation liability estimate (RLE) outlined in the Mining Rehabilitation Fund Regulations 2013. This method only provides a guide to the potential rehabilitation costs but considered appropriate for this level of study. The closure costs developed using this method relate to project infrastructure areas and exclude the large disturbance areas of the mine pits, waste dumps and mine haul roads. These costs have been incorporated into the mining cost as rehabilitation will occur progressively for these areas with project infrastructure dealt with at the end of the operation. Closure costs for project infrastructure areas are presented in Table 20-4.

Table 20-4: Projected mine closure costs for infrastructure areas

Area	Disturbance (Ha)	RLE Rate ($/Ha)	Closure Cost (AUD)
Airstrip	72.2	18,000	1,300,140
Tailings storage facility	819.8	30,000	24,594,000
Evaporation pond	188.5	50,000	9,425,000
Solar farm	31.0	18,000	558,360
ROM	10.6	18,000	190,980
NPI	23.7	18,000	427,320
Plant area	24.8	30,000	744,120
Raw water pond	1.8	30,000	52,680
ANFO	4.0	18,000	72,198
Magazine	3.0	18,000	53,910
Access road – ANFO	3.6	18,000	64,497
Camp	6.6	18,000	119,052
Access road – Camp	29.7	18,000	534,600
Access road – Bore field	22.4	18,000	403,200
Access road – solar, evap, TSF	1.8	18,000	32,400
Haul road	223.2	18,000	4,017,600
Rail loop	5.4	18,000	97,200
Access road – siding & gate house	12.0	18,000	216,000
Rail facilities – workshops, laydown	46.4	30,000	1,392,900
Bore field	1.3	18,000	23,040
Mine waste dumps roads & abandonment bund	812	17,100	13,876,981
Total			**58,196,178**

21 Capital and Operating Costs

The capital cost estimate was compiled by Engenium utilising information provided by Macarthur, Orelogy and the Engenium Project team.

21.1 Estimate Scope

Capital and operating costs have been prepared based on the Study Scope, from this a Work Breakdown Structure (WBS) was developed and used as the framework for the estimates.

Estimated costs have been broken down into the main areas required to support the Project from extraction to ship loading: mining, processing, logistics and port operations. It encompasses development capital costs to be expended from the commencement of the Project execution phase through to completion of the facilities commissioning and commencement of operations.

21.2 Capital Estimate Basis

The estimate is a Feasibility Study level estimate with an expected accuracy range of between +/-10% to +/-15% (AUSIMM Class 3), based on engineering to 25% definition.

All costs are estimated based on the pricing for labour and materials existing in Q4 2021. Escalation of costs beyond this date is not included in the capital cost estimate and has therefore not been considered within the financial model.

Estimated job hours and costs were developed based on recent historical norms for similar types of work, both firm and budgetary prices, telephone quotes and in-house data and quotations from known Vendors including the Client's preferred Vendor list where applicable. In the event where quantities, engineering data and/or equipment pricing were not available, best engineering practice and assumptions was utilised as well as industry benchmarking.

21.3 Capital Cost Estimate Summary

The capital cost estimates for the options considered are presented at a summary level in Table 21-1. The costs shown are broken down by WBS area. Table 21-1 is a summary of initial development capital and excludes deferred & sustaining capital.

Total project capital inclusive of sustaining capital and deferred capital costs are shown in Table 21-2. To minimise initial capital cost, the TSF requires three expansions in years 4, 9 and 16 totalling $39.8m. Rehabilitation of mining areas has been included in the mining costs as on a continual basis. Closure costs and rehabilitation of all other areas has been included as deferred to the end of the mine life with capital totalling $58.2m. Overburden removal for each mining stage has been capitalised in the financial model with total non-operational waste mining shown in Table 21-2.

Table 21-1: Summary of direct & indirect capital costs

Area	USD M	AUD M
DIRECTS		
Facilities process plant	11.6	16.4
Process plant	227.6	320.5
Product transport logistics	36.5	51.4
Port storage & ship loading	24.2	34.0
Infrastructure & headworks	72.0	101.3
G&A	1.3	1.8
Total direct costs	**373.1**	**525.5**
INDIRECTS		
Construction Indirects	83.6	117.8
EPCM	52.2	73.5
Spares & Commissioning	4.8	6.8
Freight	11.2	15.7
Contingency	43.9	61.9
Total indirect costs	**195.7**	**275.7**
Total project	**568.8**	**801.2**

Table 21-2: Summary of initial and deferred capital costs

Initial Capital Expenditure	USD M	AUD M
Construction Capex	568.8	801.1
Capitalised Pre-Production Operational Costs	43.8	61.6
Total Initial Capital Expenditure	**612.5**	**862.7**
Future Capital Expenditure		
Sustaining Capex	143.8	202.5
Tailing Storage Facility Lifts	28.2	39.8
Capitalised Non-Operational Waste Mining Costs	252.5	355.7
Total Future Capital Expenditure	424.6	598.0
Closure & Rehabilitation Costs		
Closure and Rehabilitation Cash Expenses	41.32	58.20

The Capital Costs include:

- **Mine Capital** – Mine capital investment includes costs for:
  - site developing, clearing and grubbing
  - laydown areas and internal roads,
  - technical services software and equipment

  - overburden removal to first ore, and
  - initial grade control.
- **Crushing Capital** – Crushing capital investment includes costs for:
  - site developing, clearing and grubbing
  - ROM pad construction
  - complete primary and secondary crushing and screening facility
  - HPGR and wet screening
  - stockpile and reclaim facility including conveyors, and
  - all associated earthworks.
- **Process Capital** – Process capital investment includes costs for:
  - site developing, clearing and grubbing
  - complete concentrate process facility
  - concentrate storage facility, and
  - all associated earthworks.
- **Tailings Capital** – Tailings capital investment includes costs for:
  - site developing, clearing and grubbing
  - tailings storage facility
  - return water system, and
  - all associated earthworks.
- **Infrastructure Capital** – Infrastructure capital investment includes costs for:
  - 440-man permanent camp including amenities
  - administration offices
  - Mine Operation Centre
  - laboratory
  - workshops
  - magazine
  - ANFO storage facility
  - dangerous goods storage facility
  - refuelling facility
  - mobile plant and equipment
  - 40 MW power plant
  - HV reticulation to entire site
  - bore water supply
  - associated bores and pumping stations

  - water treatment facilities and potable water storage
  - water reticulation
  - complete data & voice communications system, and
  - all associated earthworks.
- **Logistics Capital** – Logistics capital investment includes costs for:
  - site developing, clearing and grubbing
  - access road
  - haulage roads to rail siding
  - rail siding
  - rail siding infrastructure
  - all associated earthworks, and
  - rail upgrades are excluded and have been priced into the rail access tariff from Arc (see Section 21.9.2).
- **Port Storage Capital** – Port storage and handling capital investment includes costs for:
  - site developing, clearing and grubbing
  - storage shed; and
  - material handling to the ship loader infrastructure.
- **Capital Indirects** – Capital indirect costs include:
  - construction indirects at 15% of direct costs, cost to cover temporary construction facilities and utilities
  - owners costs at 3% of direct costs, cost to cover owners facilities and utilities
  - EPCM costs at 10% of direct costs, to cover engineering, procurement and contract management throughout construction
  - commissioning including start up and ramp up
  - freight at 3%, and
  - contingency of 10% to bring the capital estimate into line with the accuracy required for a Feasibility Study.

21.4 Direct Costs

21.4.1 Civil Earthworks

Bulk earthworks quantities were developed by Engenium as a material take-off (MTO) from the general arrangement and layout drawings. Costs were calculated and estimated using cost values from recent projects and Tier 1 Contractors.

Where limited details were available good engineering practice assumptions were utilised or factored using percentage-based statistics from Engenium's cost database of past projects.

21.4.2 Concrete

Concrete quantities were obtained from past projects with similar type and size of equipment, developed by Engenium as an MTO from the general arrangement and layout drawings. The cost was calculated and entered in the estimate as a cubic metre rate for finished concrete. The unit costs were developed from similar projects conducted at a construction level.

21.4.3 Construction and Onsite

Structural steelwork quantities were obtained from past projects with similar type and size of equipment, developed by Engenium as an MTO from the layouts. Costs were calculated and estimated using cost values from recent projects and Tier 1 Contractors.

21.4.4 Platework

Platework ex-works quantities were developed by Engenium as an MTO from the general arrangements and layout drawings. Costs were calculated and estimated using cost values from recent projects and Tier 1 Contractors.

21.4.5 Major Equipment

Major equipment ex-works costs were entered in the estimate as a series of rates based on budget quotations provided by Engenium and priced by reputable vendors from previous projects and catalogued in Engenium's cost database.

21.4.6 Pipework

Piping and valve quantities were developed by Engenium as an MTO/list. The cost was calculated and estimated using cost values from recent projects.

21.4.7 Electrical and Instrumentation

Major equipment packages costs were entered in the estimate as a series of rates based on budget quotations and priced by reputable vendors while minor items of plant were based on recent previous projects and catalogued in the Company’s cost database.

Rates for the calculation of electrical costs for the project were derived through the development of Project-specific MTOs. Where limited detail was available, good engineering practice assumptions were utilised or factored using percentage-based statistics from Engenium’s cost database of past projects.

21.4.8 Accommodation Village

The accommodation facilities capital cost is based on budgets for construction costs from companies who specialise in large mining accommodation facilities allowing for the weather, climatic conditions, and standards of accommodation and messing considered expected for the industry and the operating parameters. A design and construct package is intended for the main part of the accommodation village, with a lease component for additional rooms required for construction.

21.4.9 Direct Labour Installation Man-hours

Installation labour hours are based on in-house norms and published data for the type of work and locality.

21.4.10 Mobilisation and Demobilisation of Contractors

Contractor mobilisation and demobilisation costs (Fixed Preliminary Cost) are calculated based on a high-level assessment of the construction plant and infrastructure required for the specific portion of the works and included as a percentage of the direct cost.

21.4.11 Currency Basis

All estimates are based in Australian dollars. Where costs have been quoted in currencies other than AUD these costs were converted to the currency of the estimate based on exchange rates applicable at the base date of the estimate. The project is to utilise the foreign exchange rates as posted on the RBA website as at the base date of the estimate.

21.4.12 Benchmarking

A benchmarking comparison was made to compare project costs to production volumes with other similar projects.

Engenium has a database of several magnetite project’s costs at Pre-feasibility, Bankable Feasibility study and construction level stages.

The researched data is supported by information that can be found in the public arena and does not encroach on any confidential information.

The benchmark compared:

- Capital cost per annual tonne of production
- Total capital costs for processing
- Transport and logistics cost per ton kilometre, and
- Value per annual tonne for construction comparisons.

21.5 Indirect Costs

21.5.1 Construction and Onsite

Onsite costs include all construction management and support staff, all mobile and portable plant and equipment required to complete the work. It also includes mobilisation and demobilisation costs inclusive of all personnel, material and equipment mobilisation and demobilisation expenses associated with the contractor’s construction requirements.

Fuel, consumable materials, offices, office set-up, crib rooms, maintenance, onsite computer system, phone and communication systems, medical costs, safety equipment, mobilisation air fares, accommodation for management, inductions, and site establishment.

The Contractors Distributable component, which comprises of all other costs associated with the installation work, has been developed from first principles. The estimate includes provisions for costs normally provided by the contractor's management during construction, such as:

- Construction equipment includes the operation of equipment for offloading, materials handling & warehousing. Also included are the construction management's site vehicles.
- Operation and maintenance of buses to transport personnel from camp to their work location
- Equipment Mobilisation/Demobilisation
- Amenities and operating costs
- Construction equipment
- Indirect labour (crane drivers etc.)
- Manual support labour
- Site based overhead labour (supervision and office support)
- Accommodation and travel
- Small tools and consumables including Safety Equipment
- Fuel and lube facilities
- Off-site support; and
- Overheads and margins.

The following costs are typically excluded from the Contractors' Distributable as they are normally provided by the Owner:

- Drug and alcohol testing

- Additional security services required
- Additional medical services required
- IT Support & Infrastructure; and
- Costs of inductions, project specific HSE requirements.

In-line with the parameters of the FS estimate, costs for Construction Support are based on a factor per direct man hour.

21.5.2 Commissioning and Start Up

Commissioning allows for wet load commissioning including start-up engineering and operation crew with a squad of trades people to handle modifications and start-up issues. In line with the parameters of a FS estimate, Commissioning/Start-up costs are based on a factor of the total direct costs or based upon preliminary estimates. These costs typically include:

- First fill and commissioning consumables
- Commissioning including vendor representatives, and ramp-up (allowance of 3 months) costs to full production; and
- Capital and Commissioning Spares.

21.5.3 Owner's and Pre-production Costs

In line with the parameters of the FS estimate Owner's costs were based on a factor of the total direct costs or based upon preliminary estimates. Costs included in this factor are:

- Owner's team costs except for those included in the EPCM Project Management Team. These include offices, furniture, communications, vehicles, accommodation, salaries, and wages
- Pre-production operating costs; and
- Insurance including:
    - Contractors' All Risk
    - Construction Liability
    - Workers' Compensation
    - Marine Transit, Contractors; and
    - Automobile, Public Liability.

21.5.4 EPCM

These are the Project Manager's costs for head office and site office services. They include engineering design, consultant charges, construction supervision, commissioning, vendor reps costs, purchasing, inspection, cost control, and other administrative costs incurred as part of the Project Manager's activities. As for direct costs, an allowance for undefined items is also included.

In-line with the parameters of the FS estimate, cost for the EPCM is based on a factor of the total direct costs. Costs included in this factor are:

- Project Management Team and expenses
- Engineering Design Team and expenses
- Site Construction Management Team and expenses
- Temporary services and construction facilities (not expressly included as part of distributable contractor costs)
- Commissioning Team and expenses
- Demobilisation & Close-out, and

- Engineering Fees.

Macarthur plans to complete the project under an Integrated Owners Team (IOT) which consists of Engineering, Procurement, and Construction Management (EPCM). The construction will be contracted out to a Construction Contractor.

21.5.5 Freight

Freight costs are calculated as 3% of the ex-works material cost and or tonnage. Freight costs are shown separately from equipment and material prices in the estimate.

Fabrication and delivery lead times were sourced during the pricing enquires and clearly identified.

21.6 Capital Cost Contingency

Contingency is the provision of funds for known undefined costs, which cannot be clearly attributed at this stage of the project, but which experience tells us will be expended during the execution stage of the project. To assess the level of contingency to be applied, an assessment was made of the estimate to determine the value of contingency to be added to the estimate based on the level of confidence in the level of design complete, the accuracy of the data provided and the availability of the source of the estimated costs.

A contingency of A$61.8m was developed for the range of risks assessed during the preparation of the estimate. A Monte Carlo risk analysis was used to assess the accuracy of the estimate and to calculate the contingency necessary to provide the required probability of achieving the estimated costs.

21.6.1 Capital Cost Qualifications

The following items are specifically excluded from the capital cost estimate as they are included in the operations estimate:

- Contractor mobilisation costs have been included in the capital estimate. Demobilisation costs are included in the Financial Model.
- The following items are specifically excluded from the capital cost estimate and would require further definition and study:
  - purchase or lease of land, payment to landowners
  - capital contributions to local, State or Federal governments for infrastructure
  - sales tax on permanent equipment and materials
  - goods and services tax (GST)
  - development approvals (accommodation camp approvals are included)
  - right of way costs (approval of lease boundaries), and
  - environmental approvals & permitting.
- Costs that are excluded from the capital cost estimate and are typically considered within the financial modelling include:
  - mining contractor demobilisation (accounted for in operations costs)
  - demobilisation and rehabilitation of the site areas after the conclusion of mining operations
  - Australian Fringe Benefits Tax (FBT)
  - foreign currency exposure
  - financing costs
  - sunk costs
  - escalation costs

- project funding establishment cost
- project finance costs and associated bank charges
- deferred, working and/or sustaining capital
- marketing costs
- exploration/investigation/feasibility study costs
- government licences, royalties fees and taxes, and
- native title compensation.

21.7 Operating Cost Estimate

The operating cost estimate was compiled by Engenium utilising information provided by Macarthur, Orelogy and the Engenium Project team.

The cost estimate is a Feasibility Study level estimate with an expected accuracy range of between +/-10% to +/-15% (AUSIMM Class 3), based on engineering to 25% definition.

All costs are estimated based on the pricing for labour and materials existing in Q4 2021. Escalation of costs beyond this date is not included.

Scope of Operations Estimate

The operating cost estimate covers the cost from the mining of the ore to delivery of concentrate FOB at the port. It includes the production of water through bore field supply, the disposal of tailings, and the administration of onsite operations and corporate activities assumed to be in Perth.

The scope of the operating cost estimate covers the maintenance and operations of the following work areas over the life of mine and is organised into the following main areas.

- Mining Operations
- Primary Crushing, Concentrating and Filtration
- Logistics – road and rail transport, port operations; and
- General and Administration.

Operating Costs Estimate Organisation

Operating costs can be separated into three categories:

- Operating consumables – costs incurred as a direct result of operating
- Maintenance costs – ensuring the equipment is capable of operating at design; and
- Labour costs.

This approach can be developed for each stage of the process – mining, concentration, transport etc, as far as can be determined through the provision of detailed operating data, site layout, conditions, flow sheets, and detailed design data provided by the Owner.

A spreadsheet approach has been used for this project which allows the commodity costs and the unit consumptions to be separately altered which ensures a change in price applies across all sections using that commodity. An example is power, which is one of the main commodities used in each section.

The Operating Cost Estimate (OPEX) has been built up on an Excel based spreadsheet model, which is intended to provide the basic inputs into the financial model.

21.8 Operating Cost Estimate Summary

Table 21-3 summarises the operating costs for the Project.

Table 21-3: Summary of operating cost ($/t concentrate)

Area	USD/t	AUD/t
Mining	26.08	36.73
Crushing & Processing	22.41	31.56
Logistics	21.25	29.93
General & Administration	2.00	2.82
Subtotal	**71.74**	**101.05**
Royalties	6.05	8.51
Total operating costs ($/t concentrate)	**77.79**	**109.56**

The above costs are an annualised average cost. They do not include one-off costs such as demobilisation or reflect production ramp-up/ramp-down.

21.9 Crushing and Processing

21.9.1 Source of Estimate Quantities

The operating cost at the mine has been developed for the project by obtaining prices for a series of commodities and then applying the usage rate based on measured values or experience from other projects.

The usage rates for commodities are partly calculated and partly from typical values from other projects and experience. The major cost item is the usage of electricity, the usage of which is derived from the power draw developed in the equipment list.

The usage of comminution consumables is based on the mill power draw for grinding media using generic rules of thumb, and the likely wear life of liners at the crushers and mills is based on the experience of similar materials at similar plants.

- Power – based on the developed equipment list and installed power,
- Water – based on the mass balance
- Crusher liners – based on a new liner every 5 Mt
- Ball Mill and Vertimill mill liners – based on one per two years per mill
- Screens – based on 6-month life
- Conveyor belting – based on a 5-year life in the crushing area, and
- Grinding balls – based on 0.5 kg/t of concentrate.

Labour costs are based on the labour rate and the plant manning levels. A manning chart has been developed for the crusher and concentrator based on typical manning structures. Eleven levels of skill are allowed for and a variety of shift rosters, such as Monday to Friday, 12-hour shift rosters, continuous day shift each of which have their own adjustment in terms of add-on costs.

A ten percent contingency was applied to the developed costs.

A maintenance cost was estimated as a percentage of the mechanical capital costs.

The operating costs are reported in Australian dollars per tonne of concentrate using the applicable project exchange rate when necessary for converting the price of commodities to AUD.

21.9.2 Source of Cost Estimate Rates

The operating cost estimate rates were sourced from Engenium's in-house database, vendor information, and through discussion with the engineering team.

The cost of power per kWh was provided by a third-party power vendor at a nominated cost per kWh.

The following tables show the build-up of costs for crushing, concentrating and filtration (Table 21-3 and Table 21-5). Power and labour typically represent 60-70% of costs followed by general maintenance and consumables.

Table 21-4: Crushing operating cost ($/t ore feed)

	Annual	$/t ore	%
Power	1,178,159	0.12	19%
Labour	2,395,767	0.25	40%
Maintenance	1,250,000	0.13	21%
Consumables	1,235,060	0.13	20%
Total	6,058,986	0.63	

Table 21-5: Concentrator operating cost

	Annual $	$/t conc.	%
Power	29,180,050	9.73	36%
Process water	311,496	0.10	0.4%
Labour	25,514,862	8.50	31%
Maintenance	6,250,000	2.08	8%
Consumables	16,262,105	5.42	20%
Tailings disposal	3,614,516	1.20	4%
Total	81,133,030	27.04	

Table 21-6: Filtration operating cost

	Annual $	$/t conc.	%
Power	2,838,483	0.95	38%
Labour	2,568,928	0.86	34%
Maintenance	1,250,000	0.42	17%
Consumables	863,077	0.29	11%
Total	7,520,488	2.51	

21.10 Mining

The mining operating costs were developed by the consultant mining engineer, Orelogy.

Mining rates are based on contract mining operations with Macarthur managing the mining technical services. The mining contractor scope of services covers site establishment, drill and blast, load and haul and includes provision of all mining fleet and support equipment, workshops and maintenance facilities. Macarthur is responsible for travel to site and accommodation costs which have been built into the overall mining cost.

Contract mining rates were sourced through a request for pricing proposal and supplemented with internal cost estimates for technical services.

LOM mining costs are presented in

Table ***21-7***.

Table 21-7: Life of Mine Operating Costs

Mining OPEX	$M LOM	$/t mined	$/t Ore	% of total
VARIABLE MINING COSTS				
Direct Mining costs				
Clearing & Topsoil removal	5.0	0.01	0.02	0.2%
Road Con-struction	3.0	0.00	0.01	0.1%
L&H Overburden	349.2	0.41	1.48	11.3%
L&H Waste	493.4	0.57	2.09	16.0%
L&H Ore	346.5	0.40	1.46	11.2%
D&B Ore	242.6	0.28	1.03	7.8%
D&B Waste	332.1	0.39	1.40	10.7%
D&B Presplits	47.8	0.06	0.20	1.5%
Grade control Drilling	42.5	0.05	0.18	1.4%
Dayworks	36.8	0.04	0.16	1.2%
Ore Handling Costs				
ROM Rehandle	218.7	0.25	0.92	7.1%
Stockpile reclaim	26.2	0.03	0.11	0.8%
LIMS Dry reject	20.9	0.02	0.09	0.7%
Fuel Costs				
Bulk fuel supply	319.6	0.37	1.35	10.3%
FIXED MINING COSTS				
Owners Overhead	173.2	0.20	0.73	5.6%
General Overheads	20.6	0.02	0.09	0.7%
Contract Fees	297.5	0.35	1.26	9.6%
Flights & Accommodation	100.9	0.12	0.43	3.3%
TOTAL	**3,076.3**	**3.57**	**13.00**	

21.11 General and Administration

Administration on-site covers management, safety, training, emergency services, security and warehousing. Environmental costs have been factored to cover environmental monitoring and licence fees. Site laboratory costs are based on a build-own-operate facility inclusive of lab assays for mining grade control, processing plant and product assays.

An allowance for corporate costs has been made for off-site personnel to manage accounting, employment, contractual, and purchasing services.

The base labour rates used are pre-loading for shifts, overtime, and site allowances. To this are added the costs of the actual shift roster, a daily accommodation cost, and a FIFO cost/person/roster. For senior management the relevant level of costs for travel, accommodation and facilities are included.

Total general and administration costs are detailed in Table 21-8.

Table 21-8: General and administration operating costs

	Annual $	$/t conc
Corporate	750,000	0.25
Management & Site	4,120,331	1.37
Environmental	950,000	0.32
Laboratory	2,137,370	0.71
Total	7,957,701	2.65

21.12 Logistics

Transport logistics cover road haulage from mine to a rail head, rail load and haul to Esperance port, rail unloading, product storage, reclaim and ship loading. Haulage activities are performed by contract operators utilising their own equipment. Port operations are undertaken by Southern Ports Authority or under their management.

21.12.1 Road Haulage

Several haulage contractors haul various ore types in the region and a budget quotation was used to apply the rate for haulage of the final concentrate to the rail head. The services include all vehicles, plant, equipment and offices necessary for the provision of the services. The contractor will also have responsibility for stockpile management and train loading at the rail siding. Road haulage rates are inclusive of FIFO flights and accommodation provided by Macarthur. The contract service is based on free issue of fuel by Macarthur and fuel pricing using a rate of $0.80/L has been applied that reflects the current delivered diesel price net of the diesel excise rebate.

21.12.2 Rail Haulage

The rates for rail haulage and rail access were provided by Macarthur based on a request for proposal with several rail haulage contractors and the rail asset owner. Haulage rates are inclusive of rolling stock by the operator.

21.12.3 Port Handling and Ship Loading

The Esperance Port Authority currently manage and reclaim materials at the port for various operators. The base case assumes use of the existing iron ore unloading and ship loading circuit. Material would be reclaimed from the storage shed and loaded to a vessel by SPA as currently managed for other operators.

Ship loading rates were provided by Macarthur based on past SPA tariffs and escalated to current date. Materials handling through the RCD were factored from previous engineering studies of similar infrastructure.

Table 21-9: Product logistics operating cost

Area	$wmt/conc.
Road haulage	9.46
Rail operations	13.16
Port	4.84
Total	27.46

21.13 Operations Estimate Qualifications

The following items are specifically excluded from the operations cost estimate:

- escalation
- marketing
- vessel demurrage at the port
- sea freight of final product
- corporate overheads
- Shire rates
- mining lease costs
- exploration costs
- amortization, depreciation, financing, and accounting effects
- legal costs
- insurances
- equipment replacement costs
- operating costs for facilities outside the battery limits
- environmental approval requirements
- foreign exchange rate variations
- taxation, and
- public road usage charge.

22 Economic Analysis

22.1 Economic Model Assumptions

FTI Consulting (FTI) was engaged to develop an Excel based discounted cash flow (DCF) model based on the mine production schedule and employing capital and operating cost estimates for mining, processing, logistics and general administration costs.

The model uses constant (real, non-inflated) 2021 AU dollars for operating and capital costs with shipping and iron ore sales in US dollars with cash flows modelled in monthly periods.

The economic analysis assumes a two -year construction period and a 12-month ramp-up period that was incorporated into the mining schedule. The plant ramp-up period assumes concentrate output of approximately 1.6 Mtpa of concentrate in the first 12 months with nameplate capacity achieved in month 12.

The economic model considers cash flows from the beginning of construction onwards. Expenditure for pre-development studies, environmental permitting, mine optimisation, detailed engineering design and other pre-construction activities were not modelled. Furthermore, the model does not place the project within an estimated calendar timeline and is intended as an indication of economic potential of the project to assist in investment decisions. Between the date of this report and the commencement of construction, a period for the aforementioned pre-construction activities must be allowed.

The key assumptions used in the economic model are shown in Table 22-1. All costs are presented in AU dollars unless otherwise stated.

Table 22-1: Economic Model Assumptions

ROM ore	9.7 Mtpa
Concentrate	3 Mtpa
Recovery	31%
Concentrate grade	66.1% Fe
Mine Life	25
Exchange rate	0.71
Total concentrate	74 Mt
Long-term iron ore price (P62)	US 99/t
WA State Royalties (FOB sales)	5%
Tax rate	30%
Discount rate	6%
Salvage value at end on mine life	0
Ramp-up period	12 months
Construction period	24 months
Working capital (creditors)	30 days
Initial tax losses	91m

The following considerations were also applied to the economic analysis:

- Base iron ore price adjusted for grade and magnetite content premium
- Sales price adjusted for sea freight and marketing fees

- Sustaining capital of totalling $203 M over the life of the project
- Mine closure and rehabilitation costs of $58 M included as deferred capital at end of the mine
- Tailings storage facility expansion costs of $9.4 M, 12.8 M and $17.6 M in years 4, 9 and 16, respectively, included as deferred capital
- Overburden mining costs capitalised for each stage of pit development; and
- Operating costs are as outlined in Section 21.21 and Capital costs are as outlined in Section 21.3.

22.2 Taxes and Royalties

The economic analysis has been undertaken on both a pre-tax and post-tax basis. A corporate tax rate of 30% has been applied on taxable income. Macarthur currently holds $91 M of tax losses from operations and capital through its development activities. The Company's tax losses can be carried forward to offset its future income and the future income of members of its tax consolidated group, subject to the satisfaction of the Continuity of Ownership Test or the Same or Similar Business Test. At this stage Macarthur has received independent advice that these tax losses are likely to apply and have therefore been incorporated into the economic analysis.

A depreciation schedule was calculated by FTI with capital expenditure generally depreciated on units of production method with overburden mining capitalised costs depreciated on a straight-line basis over the effective pit life.

State royalties for processed minerals was applied at a rate of 5% of sales price minus shipping costs.

Payroll taxes and similar employee costs have been incorporated into the labour component of the operating costs.

22.3 Financing

No consideration of financing was applied to the economic analysis. The model considers the cash flow at an asset level and assumes 100% equity ownership.

22.4 Inflation

Modelling was primarily undertaken in real 2021 Australian dollars for operating and capital costs with shipping and sales in US dollars. No inflation was applied to either commodity prices or costs.

22.5 Discounted Cash Flow Analysis

A discounted cash flow model at a discount rate of 6% was used to derive pre- and post-tax NPV for the Project using the assumptions in Table 22-1.

At a 6% discount rate, the model shows a pre-tax NPV of $816 M with an IRR of 13.0%. After tax the NPV is $443 M with an IRR of 10.1%.

Total operating cash flows equal $2,979 M with an after-tax cash flow of $2,106 M.

The project generates a total of $1,475 M payable to government comprising $873 M in Federal taxes and $631 M in royalties for the Western Australian Government.

The outcomes of the base case financial valuation at 6% discount rate are shown in Table 22-2 and cash flow analysis shown in Table 22-7.

Table 22-2: Summary of Project Economics

Production	
Ore mined	236.6 Mt
Waste mined	624.9 Mt
Total mined	861.5 Mt
Strip ratio	2.64
Concentrate produced	74 Mt
Concentrate iron grade	66.1
Plant recovery	31%
Financials	**AUD M**
Sales revenue	12,614
Operating Expenses	8,116
Initial Capital Expenditure	
Construction capex	801.1
Mining overburden pre-strip	61.6
Total initial capital	862.7
Future Capital Expenditure	
Sustaining capital	203
Deferred capital - Tailings	39.8
Capitalised non-operational waste mining	355.7
Total future capital	598.0
Closure Expenditure	
Closure and rehabilitation	58.2
Total Capital Expenditure	1,460.7
Total Operating Cash Flows	3,625
Taxes & Royalties	
Tax paid	873
Royalties	631
Valuation	**AUD M**
NPV (6%) Pre-tax	816
NPV (6%) Pre-tax	443
IRR Pre-tax	13.0%
IRR Post-tax	10.1%

22.6 Sensitivity Analysis

Sensitivity analysis was undertaken on key economic inputs including:

- Iron ore price
- Capital costs
- Operating costs – individually and cumulative
- Discount rate

The Project NPV is most sensitive to iron ore pricing, followed by the exchange rate and then operating costs. When viewing the operating costs by main cost areas, NPV is most sensitive to mining costs followed equally by processing and logistics. Project NPV is least sensitive to capital cost.

The NPV sensitivities are shown in Table 22-3 to Table 22-6 and Figure 22-1 to Figure 22-2 for scenarios +/- 10% and 20% variations in the above key factors. Further sensitivity analysis was performed for the base case scenario using a discount rate between 6% and 10% (Table 22-7).

Table 22-3: Pre-tax NPV sensitivity analysis of key economic factors

Metric	-20%	-10%	Base Case	10%	20%
Iron ore price FOB	-370	223	816	1,409	2002
Capex	968	892	816	740	665
Opex	1568	1192	816	440	64
FX	2180	1422	816	320	-93

Table 22-4: Post-tax NPV sensitivity analysis of key economic factors

Metric	-20%	-10%	Base Case	10%	20%
Iron ore price FOB	-398	25	443	861	1277
Capex	574	508	443	378	313
Opex	979	711	443	175	-94
FX	1403	870	443	94	-199

Table 22-5: Pre-tax NPV sensitivity analysis of operating cost areas

Metric	-20%	-10%	Base Case	10%	20%
Mining	1136	976	816	656	497
Processing	1028	922	816	710	604
Logistics	1018	917	816	715	615

Table 22-6: Post-tax NPV sensitivity analysis of operating cost areas

Metric	-20%	-10%	Base Case	10%	20%
Mining	675	559	443	328	212
Processing	593	518	443	369	294
Logistics	585	514	443	372	302

Table 22-7: NPV sensitivity analysis of key economic factors

Discount rate	Pre-Tax	Post-Tax
6%	816	443
7%	635	305
8%	482	189
9%	353	91
10%	243	8

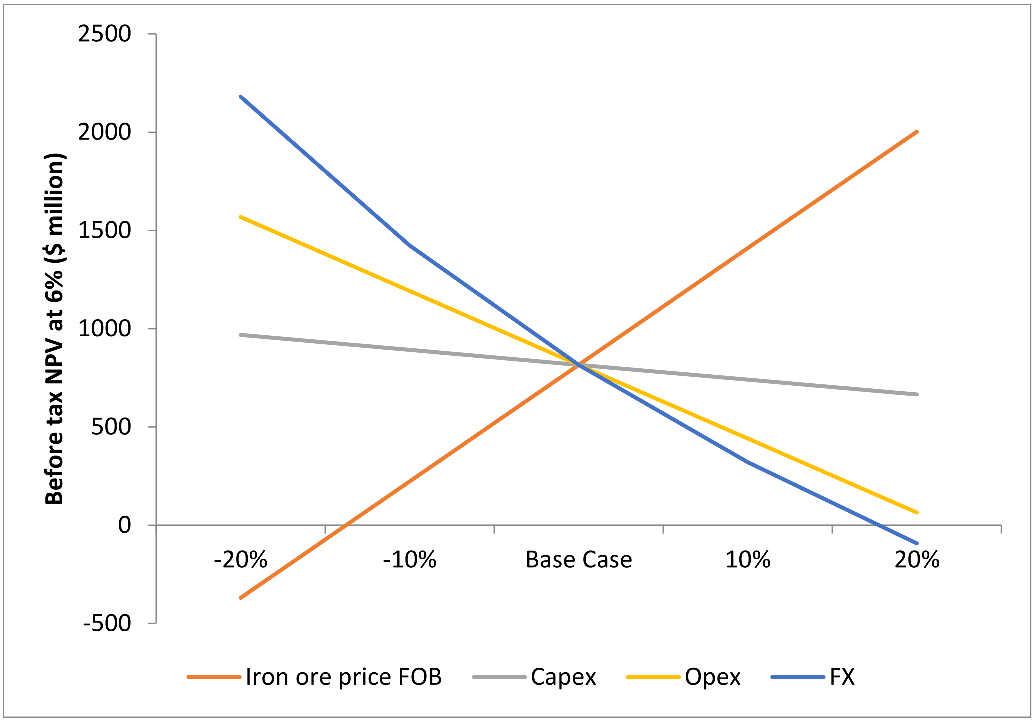


Figure 22-1: Pre-tax NPV sensitivity

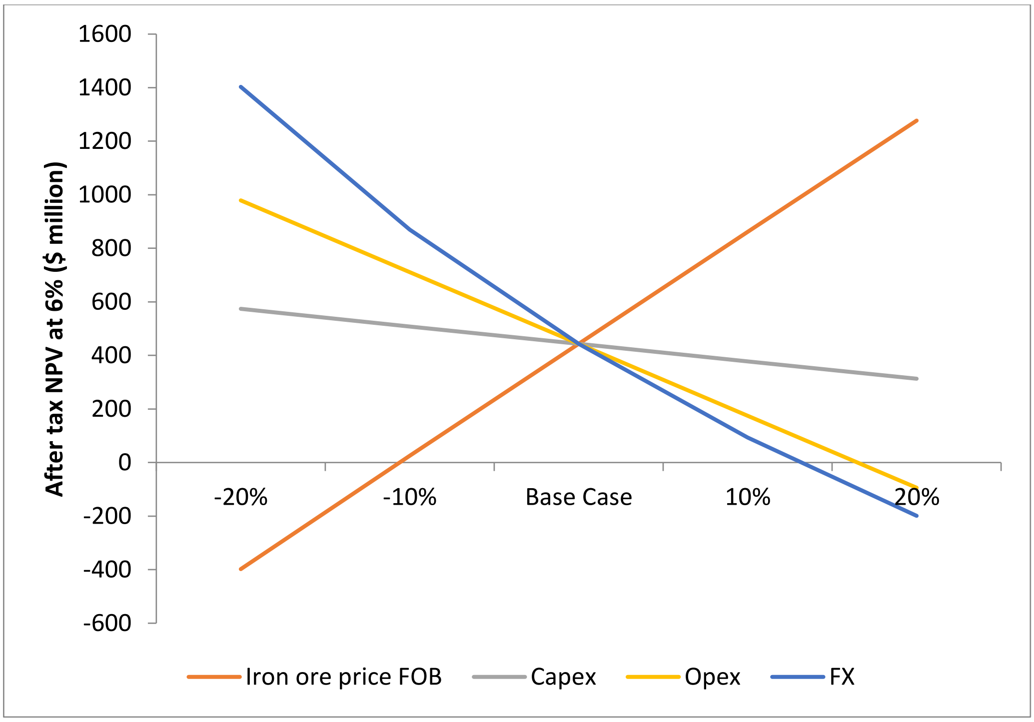


Figure 22-2: Post-tax NPV sensitivity

Table 22-8: Lake Giles Project Financial Outcomes

			-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12
Mining																	
Tonnes																	
Direct (High-grade) ore mined	dmt	200,445,586	-	-	67,764	5,480,223	8,939,435	7,360,953	7,686,136	7,633,928	8,113,702	8,140,595	8,885,196	8,383,108	8,724,187	9,455,608	9,449,649
Low Grade - HG Silica ore mined	dmt	14,702,373	-	-	172,121	286,399	57,314	406,457	150,136	82,836	325,897	1,284,466	-	89,953	2,582,151	396,216	-
Low Grade - LG Silica ore mined	dmt	21,438,234	-	-	4,240	1,486,797	1,732,779	156,224	117,806	740,857	842,995	861,569	1,400,219	1,538,041	842,116	1,814,123	2,662,101
Total Ore Mined	dmt	236,586,192	-	-	244,124	7,253,419	10,729,528	7,923,635	7,954,079	8,457,621	9,282,593	10,286,631	10,285,414	10,011,102	12,148,454	11,665,947	12,111,750
Waste Mined	dmt	624,129,631	-	-	31,598,412	36,424,600	32,947,393	36,410,340	34,603,220	33,393,014	35,203,031	32,224,936	33,778,381	32,804,164	29,329,566	32,311,760	28,765,473
Strip Ratio	W:O	2.64x	-	-	129.44x	5.02x	3.07x	4.60x	4.35x	3.95x	3.79x	3.13x	3.28x	3.28x	2.41x	2.77x	2.38x
Total Material Moved	dmt	902,536,424	-	-	32,478,849	44,360,205	44,420,687	46,741,268	44,065,280	42,605,924	44,723,217	43,482,622	44,690,120	45,043,735	42,820,979	44,120,273	41,278,004
Processing																	
Ore Processed	dmt	236,586,192	-	-	67,764	5,844,765	9,518,263	9,534,319	8,871,731	8,317,937	8,215,666	9,013,507	9,381,460	9,536,003	9,589,677	9,455,608	9,721,946
Production																	
Magnetite Concentrate	dmt	74,078,307	-	-	20,234	1,900,561	3,007,099	2,983,075	3,006,131	3,005,350	3,009,277	3,001,164	3,002,583	3,005,352	3,009,235	3,000,362	3,000,093
Iron Product Grade	%	66.19%	-	-	67.18%	67.42%	66.94%	66.07%	64.73%	64.94%	65.19%	65.82%	65.78%	66.12%	66.10%	66.11%	66.20%
Logistics																	
Product hauled	kwmt	80,745,355	-	-	22,055	2,071,494	3,277,856	3,251,552	3,274,676	3,277,838	3,280,112	3,271,268	3,272,815	3,271,813	3,283,152	3,271,329	3,270,101
Product railed	kwmt	80,745,355	-	-	22,055	2,068,067	3,276,737	3,247,576	3,272,917	3,278,434	3,281,662	3,271,535	3,271,785	3,271,678	3,281,662	3,281,662	3,270,294
Product shipped	kwmt	80,745,355	-	-	-	2,040,000	3,230,000	3,230,000	3,230,000	3,400,000	3,230,000	3,230,000	3,400,000	3,230,000	3,230,000	3,230,000	3,400,000
Project Operational Cashflows																	
Total Revenue	A$ M	**12,614**	-	-	-	305	505	507	505	526	507	507	526	507	505	507	526
Operating Expenses																	
Mining Costs	A$ M	(2,721)	-	-	(11)	(107)	(129)	(129)	(145)	(120)	(121)	(112)	(109)	(127)	(156)	(126)	(122)
Processing Costs	A$ M	(2,338)	-	-	(0)	(53)	(95)	(94)	(94)	(94)	(94)	(94)	(95)	(95)	(95)	(95)	(95)
Logistics Costs	A$ M	(2,217)	-	-	(0)	(50)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(91)	(90)
G&A Costs	A$ M	(209)	-	-	(1)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)
Royalties	A$ M	(631)	-	-	-	(13)	(25)	(27)	(25)	(25)	(25)	(27)	(25)	(25)	(25)	(27)	(25)
Total Operating Expenses	A$ M	**(8,116)**	-	-	(12)	(231)	(347)	(348)	(363)	(337)	(338)	(331)	(326)	(345)	(374)	(346)	(340)
Initial Capital Expenditure																	
Construction Capex	A$ M	(801)	(29)	(512)	(260)	-	-	-	-	-	-	-	-	-	-	-	-
Capitalised Pre-Production Operational Costs	A$ M	(62)	-	-	(62)	-	-	-	-	-	-	-	-	-	-	-	-
Total Initial Capital Expenditure	A$ M	**(863)**	(29)	(512)	(322)	-	-	-	-	-	-	-	-	-	-	-	-
Future Capital Expenditure																	
Sustaining Capex	A$ M	(203)	-	-	(1)	(5)	(5)	(5)	(5)	(5)	(6)	(10)	(10)	(10)	(10)	(10)	(10)
Tailing Storage Facility Lifts	A$ M	(40)	-	-	-	-	-	(5)	(5)	-	-	-	(6)	(6)	-	-	-
Capitalised Non-Operational Waste Mining Costs	A$ M	(356)	-	-	(12)	(28)	(18)	(22)	(8)	(22)	(27)	(28)	(31)	(16)	(3)	(22)	(21)
Total Future Capital Expenditure	A$ M	**(598)**	-	-	(13)	(33)	(23)	(31)	(18)	(27)	(33)	(38)	(48)	(32)	(13)	(32)	(31)
Total Capex	A$ M	**(1,461)**	(29)	(512)	(335)	(33)	(23)	(31)	(18)	(27)	(33)	(38)	(48)	(32)	(13)	(32)	(31)
Closure/Rehab Costs																	
Closure and Rehabilitation Cash Expenses	A$ M	(58)	-	-	-	-	-	-	-	-	-	-	-	(5)	(2)	-	-
Total Operating Cashflows	A$ M	**2,979**	**(29)**	**(512)**	**(347)**	**42**	**135**	**128**	**124**	**162**	**136**	**138**	**152**	**125**	**116**	**130**	**155**
Financial and Corporate Cashflows																	
Taxation																	
Tax Paid	A$ M	(873)	-	-	-	-	(3)	(36)	(33)	(37)	(41)	(37)	(47)	(34)	(26)	(36)	(33)
Equity Cashflows																	
Equity Drawdowns	A$ M	105	29	76	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Corporate Cashflows	A$ M	**2,212**	**-**	**(435)**	**(347)**	**42**	**132**	**92**	**91**	**125**	**96**	**101**	**104**	**92**	**90**	**94**	**122**
Key Performance Measures																	
EBITDA	A$ M	4,498	-	-	(22)	71	154	158	145	193	164	179	200	158	130	164	189
EBT	A$ M	3,023	-	-	(24)	43	110	114	102	154	124	135	153	108	79	113	135
NPAT	A$ M	2,106	-	-	(24)	42	99	77	72	108	86	93	107	75	54	77	95
CFADS	A$ M	2,106	(29)	(512)	(347)	42	132	92	91	125	96	101	104	92	90	94	122

			13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
Mining																	
Tonnes																	
Direct (High-grade) ore mined	dmt	200,445,586	7,784,158	9,279,908	9,368,719	9,321,513	5,976,479	6,308,844	7,201,451	5,374,294	9,013,220	5,887,273	6,953,287	9,286,090	10,167,817	202,049	-
Low Grade - HG Silica ore mined	dmt	14,702,373	-	-	-	128,772	3,428,258	2,430,778	2,880,616	-	-	-	-	-	-	-	-
Low Grade - LG Silica ore mined	dmt	21,438,234	1,210,984	1,706,061	1,635,883	1,535,845	-	-	-	-	-	-	1,149,594	-	-	-	-
Total Ore Mined	dmt	236,586,192	8,995,143	10,985,969	11,004,602	10,986,130	9,404,737	8,739,622	10,082,067	5,374,294	9,013,220	5,887,273	8,102,881	9,286,090	10,167,817	202,049	-
Waste Mined	dmt	624,129,631	35,990,064	17,881,422	15,400,700	15,956,380	18,898,490	16,201,823	17,809,109	23,960,508	6,689,061	11,332,333	5,658,081	5,628,879	2,911,594	16,896	-
Strip Ratio	W:O	2.64x	4.00x	1.63x	1.40x	1.45x	2.01x	1.85x	1.77x	4.46x	0.74x	1.92x	0.70x	0.61x	0.29x	0.08x	-
Total Material Moved	dmt	902,536,424	47,109,461	29,072,572	26,577,339	26,942,510	32,345,989	28,655,982	30,209,233	34,150,573	16,834,324	22,440,668	16,672,499	15,816,116	13,079,411	941,774	856,811
Processing																	
Ore Processed	dmt	236,586,192	9,746,308	9,404,840	9,368,719	9,321,513	10,019,241	10,023,381	9,519,508	10,190,065	10,145,263	11,108,334	9,864,824	10,187,237	10,167,817	450,496	-
Production																	
Magnetite Concentrate	dmt	74,078,307	3,001,280	3,000,983	3,000,027	3,000,078	3,001,266	3,001,466	3,000,684	3,000,236	3,001,322	3,000,821	3,000,092	3,000,784	3,001,211	117,542	-
Iron Product Grade	%	66.19%	66.07%	66.63%	66.55%	66.62%	66.11%	65.95%	66.21%	66.03%	65.87%	65.86%	66.25%	67.54%	67.93%	67.37%	-
Logistics																	
Product hauled	kwmt	80,745,355	3,271,395	3,271,071	3,270,029	3,270,085	3,271,380	3,271,598	3,270,745	3,270,257	3,271,441	3,270,895	3,270,100	3,270,854	3,271,320	128,121	-
Product railed	kwmt	80,745,355	3,271,395	3,271,071	3,270,029	3,270,085	3,271,380	3,271,598	3,270,745	3,270,257	3,271,441	3,270,895	3,270,100	3,270,854	3,271,320	128,121	-
Product shipped	kwmt	80,745,355	3,230,000	3,230,000	3,230,000	3,400,000	3,230,000	3,230,000	3,230,000	3,400,000	3,230,000	3,230,000	3,230,000	3,230,000	3,400,000	165,355	-
Project Operational Cashflows																	
Total Revenue	A$ M	**12,614**	507	505	507	526	507	505	507	526	507	505	507	505	528	34	5
Operating Expenses																	
Mining Costs	A$ M	(2,721)	(104)	(116)	(113)	(102)	(100)	(96)	(85)	(79)	(82)	(80)	(75)	(75)	(71)	(25)	(4)
Processing Costs	A$ M	(2,338)	(95)	(95)	(95)	(95)	(95)	(95)	(95)	(95)	(95)	(96)	(95)	(95)	(95)	(11)	(0)
Logistics Costs	A$ M	(2,217)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(90)	(10)	(1)
G&A Costs	A$ M	(209)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(1)
Royalties	A$ M	(631)	(25)	(25)	(27)	(25)	(25)	(25)	(27)	(25)	(25)	(25)	(27)	(25)	(25)	(3)	(1)
Total Operating Expenses	A$ M	**(8,116)**	(322)	(333)	(333)	(319)	(318)	(314)	(305)	(297)	(300)	(298)	(294)	(293)	(289)	(57)	(7)
Initial Capital Expenditure																	
Construction Capex	A$ M	(801)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capitalised Pre-Production Operational Costs	A$ M	(62)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Initial Capital Expenditure	A$ M	**(863)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Future Capital Expenditure																	
Sustaining Capex	A$ M	(203)	(10)	(10)	(10)	(10)	(10)	(10)	(10)	(10)	(10)	(10)	(10)	-	-	-	-
Tailing Storage Facility Lifts	A$ M	(40)	-	-	(9)	(9)	-	-	-	-	-	-	-	-	-	-	-
Capitalised Non-Operational Waste Mining Costs	A$ M	(356)	(38)	(2)	(0)	(7)	(9)	(6)	(18)	(20)	-	-	-	-	-	-	-
Total Future Capital Expenditure	A$ M	**(598)**	(48)	(12)	(19)	(25)	(19)	(16)	(28)	(30)	(10)	(10)	(10)	-	-	-	-
Total Capex	A$ M	**(1,461)**	(48)	(12)	(19)	(25)	(19)	(16)	(28)	(30)	(10)	(10)	(10)	-	-	-	-
Closure/Rehab Costs																	
Closure and Rehabilitation Cash Expenses	A$ M	(58)	-	-	-	-	-	-	-	-	-	-	-	-	-	(3)	(48)
Total Operating Cashflows	A$ M	**2,979**	**137**	**159**	**156**	**181**	**170**	**174**	**175**	**200**	**198**	**197**	**203**	**211**	**239**	**(26)**	**(50)**
Financial and Corporate Cashflows																	
Taxation																	
Tax Paid	A$ M	(873)	(37)	(36)	(40)	(37)	(36)	(38)	(39)	(46)	(37)	(36)	(37)	(43)	(37)	(11)	-
Equity Cashflows																	
Equity Drawdowns	A$ M	105	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Corporate Cashflows	A$ M	**2,212**	**100**	**123**	**115**	**144**	**134**	**137**	**136**	**154**	**160**	**160**	**166**	**169**	**202**	**(37)**	**(50)**
Key Performance Measures																	
EBITDA	A$ M	4,498	184	170	174	211	186	191	203	233	205	207	213	211	240	(12)	(2)
EBT	A$ M	3,023	127	113	116	152	121	124	136	156	125	117	130	123	152	(16)	(2)
NPAT	A$ M	2,106	88	78	79	106	84	85	93	109	86	80	88	85	106	(18)	(4)
CFADS	A$ M	2,106	100	123	115	144	134	137	136	154	160	160	166	169	202	(37)	(50)

23 Adjacent Properties

Macarthur is also managing the Ularring Hematite Project, located in the same regional area as the Moonshine Magnetite Project. Hematite is hosted in the same suite of BIF ridges as the magnetite but does not exist together with hematite in the same geological location. Hematite requires a separate metallurgical process and infrastructure making it distinctly different to the Lake Giles Iron Project discussed in this report.

A number of other companies hold almost all of the ground favourable for iron mineralisation exploration within approximately 100 km of the Lake Giles Iron Project. These include Mineral Resources Ltd, Mindax Ltd, Jupiter Mines Limited and Cashmere Iron Ltd.

Iron ore (DSO) mining operations are presently being undertaken by Mineral Resources Ltd at the Deception and Koolyanobbing deposits. Iron mineralisation has also been recently mined at the Windarling, Mount Jackson and Carina, owned by Mineral Resources Ltd. Figure 23-1 shows the Mineral Resources Ltd tenement holdings for its various iron projects in close proximity to Macarthur's Lake Giles Iron Project.

The QP has been unable to verify this information and the information is not necessarily indicative of the mineralisation on the property that is the subject of this Technical Report.

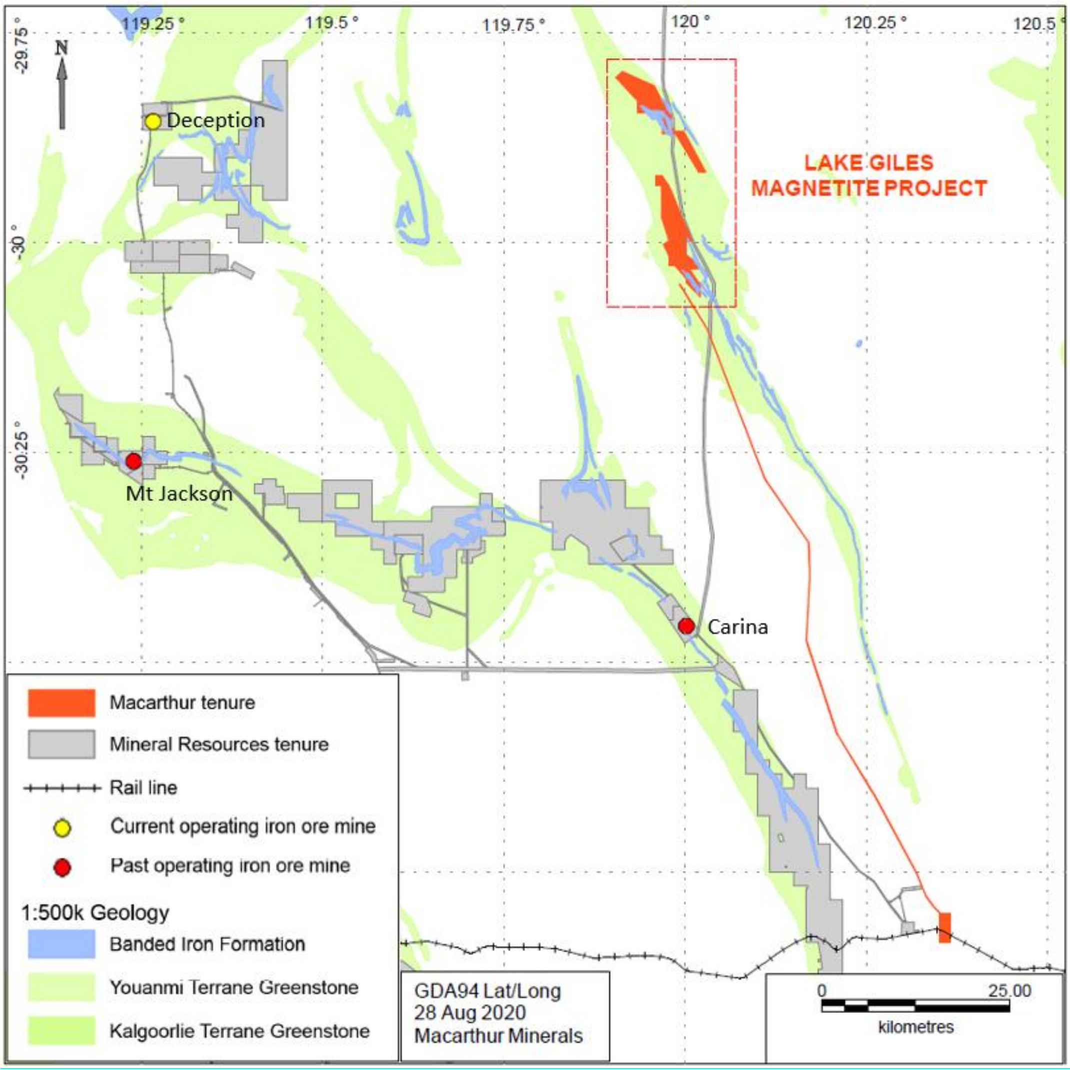


Figure 23-1: Iron Ore Exploration and Mining tenements adjacent to the Project

Source: Macarthur (2020)

24 Other Relevant Data and Information

It is the authors' opinion that there is no other data or information that is relevant to this assessment of the Lake Giles Iron Project that has not been disclosed elsewhere in the document.

25. Interpretation and Conclusions

25.1. General Conclusions

This study confirms an economically viable project producing 3 million tonnes per annum (dry basis) of high-grade magnetite concentrate over a 25-year mine life. The project will leverage off access to existing regional rail and port infrastructure and deliver a premium concentrate (66% Fe) product with low impurities. The Feasibility Study underpins a maiden Ore Reserve of 237 million tonnes.

A discounted cash flow model at a discount rate of 6% was used to derive pre- and post-tax NPV for the Project. All figures are presented in AUD unless otherwise specified.

At a 6% discount rate, the model shows a pre-tax NPV of $775 M with an IRR of 12.7%. After tax the NPV is $416 M with an IRR of 9.9%.

Total operating cash flows equal $2,881 M with an after-tax cash flow of $2,143 M.

The project generates a total of $1,475 M payable to government comprising $844 M in Federal taxes and $631 M in royalties for the Western Australian Government.

The Project NPV is most sensitive to iron ore pricing, followed by the exchange rate and then operating costs. When viewing the operating costs by main cost areas, NPV is most sensitive to mining costs followed equally by processing and logistics. Project NPV is least sensitive to capital cost.

The work undertaken for this study has shown that the Project is very dependent on:

1. Liberation size of the magnetite mineralisation
2. Water and power supplies to the Project; and
3. Port access and infrastructure.

A number of key risks have been identified during the Preliminary Assessment, which include:

1. Current crushing and grinding test work is limited in its representation; but is being addressed in the upcoming drilling programme
2. It is important to remove the siliceous gangue minerals at as coarse a grind size as possible so as to reduce the comminution energy required at each stage if the product specification is found to be unsuitable, due to the high sulphur content, further work will be needed to address the issue
3. Approvals and licensing process should commence as soon as feasible to ensure security in obtaining the resource for the Project; and
4. Given port capacity constraints and port development timelines, negotiation with SPA should commence as soon as possible to address issues and reduce options.

25.2. Mineral Resources

A Mineral Resource estimate has been prepared for the Lake Giles Iron Project, based upon a total of 352 RC drillholes and 21 diamond holes. Results from these drillholes, and from geological field mapping and observations, provided the basis for the geological interpretations. The Mineral Resource estimate was classified as a combination of Measured, Indicated and Inferred in accordance with 2014 CIM Definition Standards. The classification level is based upon an assessment of the geological understanding of the deposit, QAQC of the samples, mass recovery results, density data and drillhole spacing.

The outcropping iron mineralisation in the Project area is comprised of a combination of unaltered silica-rich BIFs and altered, enriched haematite/goethite BIFs. Weathering has resulted in the leaching of majority of the silica from the BIFs, thus producing a rock with elevated iron and decreased silica grades, near surface. These enriched bands vary from 10 m to 150 m in true thickness and are steeply dipping at 70–90°. The outcrop of weathered iron mineralisation is indicative of the fresh (non-weathered) magnetite mineralisation located down dip which is favourable for hosting a Mineral Resource.

The main zones of magnetite mineralisation are interpreted as a series of thick tabular units, with moderate to minimal structural deformation. More intense deformation is modelled at the south edge of the Moonshine prospect with several synclinal structures and possible shearing related to recumbent folds, which increase the apparent thickness of the zones of mineralisation.

Depth and consistency of magnetite mineralisation has been confirmed to in excess of 250 m below surface as demonstrated by results from several drillholes, confirming a consistent easterly dip of the hanging wall for the majority of the Moonshine and Moonshine North prospects.

The Lake Giles Magnetite deposits were drilled with either RC or diamond core drilling. The RC holes are drilled with a 140 mm diameter hammer, often on track mounted rigs due to the rugged terrain of the deposit. Diamond holes were drilled with HQ diameter core, or larger PQ diameter core if metallurgical samples were required. Not all holes penetrated mineralisation.

Macarthur provided geological and mineralisation interpretations to CSA as 3D wireframe solids and surfaces. The drillhole samples were flagged within the mineralisation domains, and geostatistical studies carried out for the head and concentrate assay data, including variography to ascertain the spatial variation of the various grade variables.

A block model was constructed for the Moonshine and Moonshine North deposits using Datamine software, with parent block sizes 25 m (along strike) x 25 m (across strike) x 10 m (vertical). A larger block size of 50 m (along strike) x 50 m (across strike) x 10 m (vertical) was used for the magnetite deposits to the north of Moonshine (Sandalwood, Clark Hill North, Clark Hill South, and Snark). Head and concentrate grades, and mass recovery, were estimated into the block model using ordinary kriging.

For Moonshine and Moonshine North, a minimum of eight and maximum of 18 samples were used in any one block estimate, with a maximum of four samples per drillhole. Search ellipsoid radii varied between the deposits. Typically, a primary search ellipse of 240 m along strike and down plunge x 120 m down dip x 40 m across strike was used.

For Sandalwood, Clark Hill North, Clark Hill South and Snark, a minimum of 12 and maximum of 30 samples were used in any one block estimate, with a maximum of six samples per drillhole. Search ellipsoid radii of 300 m along strike and down plunge x 100 m down dip x 100 m across strike was used.

Block grades were validated by visually comparing block and adjacent drill sample grades, by the use of swath plots, and by comparing mean sample and block grades by mineralisation domain.

A total of 624 drill samples with bulk density measurements were captured within the Moonshine and Moonshine North mineralisation domains and statistically assessed to determine the mean and ranges, and to see if any excessively low or high bulk density values were present. Three mineralisation domains contain bulk density data. A further 400 samples were taken from the BIF oxide zones, or the footwall and hanging wall waste zones. Core samples, both from the fresh and oxidised zones, were highly competent, without any fractures or voids, and were not required to be wax sealed prior to immersion in water. A conventional Archimedes wet/dry weighing was used to measure density.

Algorithms were developed to calculate the density to apply to the Moonshine and Moonshine North block models based upon correlations between the head iron grade from assays, and the corresponding bulk density value of the sample. The density algorithms as applied to the Mineral Resources, are given here:

- Moonshine: DENSITY = (0.0241*FE) + 2.624
- Moonshine North: DENSITY = (0.0295*FE) + 2.468; and
- Moonshine (East): DENSITY = (0.0293*FE) + 2.492.

For the Sandalwood, Clark Hill North, Clark Hill South, and Snark deposits, density measurements were taken from drill sample data located at Clark Hill. A total of 122 diamond core billets were taken from four diamond holes, with 63 of the samples located within the BIF host rock.

The Sandalwood, Clark Hill North, Clark Hill South, and Snark Mineral Resources were all applied a density value of 3.3 t/m3, which is a typical density value for the style of mineralisation and is similar to the average bulk density at Moonshine and Moonshine North.

The Measured Mineral Resources were based upon a confirmed understanding of the geological and grade continuity. Drill spacing is typically 25 m along the northerly strike, with often two to three holes per section. The Measured volumes also contain samples subject to DTR test work, with associated assays from the recovered concentrates. Bulk density measurements were also available.

The Indicated Mineral Resources were based upon an assumed understanding of the geological and grade continuity. Drill spacing is typically 25–50/100 m along the northerly strike, with at least one hole per section. The Indicated volumes also contain samples subject to DTR test work, with associated assays from the recovered concentrates. Bulk density measurements may also be available.

The Inferred Mineral Resources were based upon an implied understanding of the geological and grade continuity. Some mineralisation domains are only cut by one drillhole, and the geological models are strongly guided by surface mapping of the BIF outcrops. Drill spacing is typically ≥100 m along the northerly strike. DTR and bulk density results are generally absent from within the Inferred volumes, although the Sandalwood, Clark Hill North, Clark Hill South, and Snark Mineral Resources are supported by sufficient DTR test work results to support the reporting of concentrate grade estimates.

The Mineral Resources are based upon data collected over the history of the Project, all of which exhibit margins of error, whether natural or human induced. Examples are provided below:

25.2.1. Drilling:

1. Downhole surveys provide estimates for the spatial location of drill samples. At downhole depths of >100 m, margins of error tend to increase. This is mitigated in the Mineral Resource via the Mineral Resource classification categories, with the deeper volumes, which are impacted more by potential errors in down hole survey locations of sample data, classified as Inferred, being the highest risk category.
2. Samples exhibit both natural and human induced errors. Macarthur's sampling procedures are designed to minimise or eliminate the human errors as much as possible, and the QPs are of the opinion that sampling error is minimised overall.

25.2.2. Geological interpretations:

1. The geological logging of drill samples is a subjective exercise, and the results are used to guide the geological interpretations underpinning the Mineral Resource. Macarthur's geologists are experienced in geological logging of iron mineralisation and used sample analyses to confirm their logs. Macarthur's geological procedures are designed to minimise or eliminate the human errors as much as possible, and the QPs are of the opinion that any errors in the geological logs and geological interpretation are minimised overall

25.2.3. Bulk density, sample assays and DTR:

1. Bulk density test work was carried out in accordance with Macarthur's procedures and results reflect those of other magnetite Mineral Resources reported from other properties, with a minimal margin of error. The classification categories for the Mineral Resource reflect the quantity of bulk density test work.
2. Sample assays and DTR test work was carried out by accredited analytical laboratories, in accordance with their own procedures, and their quality control protocols were followed. The classification categories for the Mineral Resource reflect the quantity of DTR test work from local samples.

25.2.4. Mineral Resource:

1. The Mineral Resource estimate combines all the above data, with their margins of error. The Mineral Resource is not a calculation and is referred to as an estimate due to the margins of error inherent in the input data. The Mineral Resource classification categories appropriately convey the risks for the various volumes within the magnetite mineralisation domains:
    - The Mineral Resource estimate is classified according to levels of risk (Measured, Indicated and Inferred), which are defined in Section 0.
    - The highest levels of risk are in the Inferred Mineral Resources, and the risks can be reduced by additional drilling and associated geological and metallurgical studies, after the inclusion of their results in any future Mineral Resource estimates.
    - The interpretations and conclusions reached in this report are based on current geological understanding and the best evidence available to the authors 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 based on interpretations or conclusions contained in this report will therefore carry an element of risk.

- All available data was assessed and the QP's relative confidence in the data was used to assist in the classification of the Mineral Resource. The current classification appropriately reflects the QP's view of the deposit.

25.3. Mining

The Mineral Reserve estimate has been prepared for the Moonshine and Moonshine North pits in accordance with 2014 CIM Definition and Standards with more than 20% based on material classified as Measured and the remainder as Indicated.

The geometry of the wide sub-vertical orebodies is amenable to bulk mining methods with low dilution and ore loss. The continuity of the orebodies is also favourable for blasting along strike to minimise dilution on the edges of the ore zones.

Pit optimisation using the latest physical, technical, and economic parameters was used to determine the ideal geometry of the ultimate pit outline. The optimised pit shells bottom out on the Measured and Indicated resources. Inclusion of Inferred material produced a larger pit shell by more than 80%, however, the discounted cashflow of the larger pit was 35% less than the shell selected for design. This indicates that deeper drilling is not warranted at the current iron ore price due to the high strip ratios to access the ore at depth.

Silica reporting through to concentrate was identified as a primary driver of ore feed to the plant. Based on information by Engenium, analysis of the grade distribution indicated that 7% SiO_2 in the DTR concentrate would be viable feed for the plant to produce saleable concentrate. The Moonshine North pit contains ore with higher mass recovery based on the DTR grade, but this is associated with substantially high SiO_2 in concentrate compared to the Moonshine pit. The blending strategy was developed to manage the DTR grade and the silica reporting to the concentrate.

The pit designs were guided by the nested pit optimisation shells. Moonshine North is a smaller pit and can be mined in two stages. Due to the size of the Moonshine pit, a total of five stages were developed to facilitate ore extraction at consistent blend with material from the Moonshine North pit whilst balancing waste movement. The blending strategy is sensitive to the extraction sequence and operation of the mine will require tight controls to ensure that the short-term schedules are kept in line with the life of mine plan.

The plant was relocated from the previous site to a central point to the southwest between the two pits. This provided more even terrain for the process plant and situated the ROM pad centrally to pits reducing the overall haulage distances. The waste rock dumps were developed using haulage simulation to minimize the haulage distances over the life of the mine.

The operating strategy using experienced mining contractors with the Owner maintaining orebody definition, quality control, supervision and management reduces the operational risk at start-up and provides opportunity for value improvement by transitioning to Owner mining once the operation becomes steady state.

Orelogy has relied upon foundation data supplied by other experts in the preparation of the mine plan for the Lake Giles Project. The QP assessed the information provided and is confident that the data is of a standard for reporting the Mineral Reserve at Feasibility level.

25.4. Metallurgy and Processing

A discussion of the metallurgical test work results alongside the resource model, led to the project product being defined as below. A process flowsheet was developed to achieve 3 Mtpa of product, with conventional crushing and screening, followed by HPGR and wet screening, two stage fine grinding and magnetic separation, reverse flotation and a final wash before filtration for storage and loadout.

Table 25-1: Project Product Specification (%)

Fe	Al_2O_3	SiO_2	P	LOI	S
66.1	0.10	4.9	0.02	-2.7	0.6

25.5. Infrastructure, Logistics and Port

The study has identified the preferred logistics option of hauling the product by private road to a rail loop on the existing Eastern Goldfields Railway (EGR), transport by rail to Esperance and finally loading onto cape class vessels for export.

Apart from the existing EGR and ship loader at the port, all infrastructure has been designed and costed by Engenium. Where applicable, the Company has elected to develop several facilities under a build-own-operate (BOO) model funded and managed by interested third parties. Such facilities include the laboratory and power station with pricing treated as an operating cost over the term of the proposed contract.

To facilitate export from the Port of Esperance, new infrastructure is required to unload the concentrate from rail wagons, handle and store it locally, and finally deliver it onto existing bulk product ship loading facilities at the Port.

The proposed development excluding the concentrate storage shed is to be funded by a third-party infrastructure asset group. Under this scenario, the Company would be charged a tariff for material handled through the circuit, operated by SPA or the asset owner.

25.6. Marketing

On the 21 March 2019, Macarthur Minerals Limited announced the entering into binding Offtake and Marketing agreement with Glencore. Transaction Highlights:

1. Glencore secures offtake for the Project with commercial terms for approximately 4 million tonnes per annum average for the first 10 years, with the option to extend for a following 10 years for all tonnes of future Lake Giles iron ore production
2. Glencore agrees to release up to 70% of their off-take volume where Macarthur secures project financing from a Strategic Industry Investor, subject to their securing off-take of the product produced
3. Glencore will take possession of the iron ore once it is being loaded onto a vessel for export
4. Glencore is responsible for the marketing, shipping, delivery and associated freight insurances
5. This Agreement with Glencore positions Macarthur to go forward to complete their project financing; and
6. Terms and conditions have been competitively negotiated reflecting strong forward demand.

Iron pricing for this study is based on a consensus view of several broker reports described above and a comparison of historical broker forecasts against actual pricing over time. Iron ore pricing and assumptions used in the economic analysis are shown in Table 25-1.

A long-term pricing scenario of US$99/t has been employed in the base case scenario, adjusted for grade. This is considered a conservative forecast in comparison to historical pricing dating back to 2010.

26. Recommendations

The following recommendations have been identified during compilation of this study. A detailed Scope of Work considering all recommendation to progress the Project is required to identify suitable work programmes and cost estimates for the work.

26.1. Mineral Resource Recommendations

26.1.1. Exploration Strategy and Budget

CSA recommend the following actions are completed to support the ongoing exploration and evaluation effort at the Lake Giles Iron Project:

1. Continue to develop a deposit scale geological model incorporating lithology, mineralisation, weathering and structural features that locally control the occurrence and location of BIF host rock.
2. Consider domaining a zone exhibiting higher magnetite concentration, and lower silica levels, for future Mineral Resource estimates. The domain would need to exhibit sufficient strike and down dip extent to be justified for future use.
3. Maintain field geological procedures with respect to drill rig inspections and sampling procedures, vetting the maintenance and cleanliness of sample splitters and sample recovery.
4. Monitor the performance of CRMs and field duplicates immediately upon receipt of assays.
5. Macarthur geologists to compile a QAQC report prior to future Mineral Resource estimates.
6. Merge the drillhole databases containing the pre-2019, and 2019 drill data.
7. Complete additional drilling in Indicated, Inferred and un-classified Mineral Resource areas to increase geological confidence of individual mineralised units. This will require budgeting of money and resources and will require a time frame of at least 10 months from initial drillhole planning and budgetary approval to final receipt of sample assays. A proposed budget is provided in Table 26-1 (excludes fixed costs).

Table 26-1: Proposed Exploration Budget (A$)

Project	Work Program	Cost
Moonshine / Moonshine North	Extensional RC Drilling - 2500m	$550,000
	Assay samples	$220,000
	Site Prep/Rehab	$22,000
	Consumables	$11,000
Sandalwood, Clark Hill, Snark	No further work proposed	
Total		**$803,000**

26.2. Mining and Mineral Reserves Recommendations

Orelogy recommends the following actions before moving to implementation through to early operations:

- Variability test work program: Blending of the silica reporting through to concentrate was a primary driver of the mine schedule and resulted in periods where stockpiles were drawn down to very low levels in early years and built up to larger tonnages in the later years. The mine schedule is based on the DTR samples and estimated concentrate grades interpolated within the resource model whilst the process data is based on a single bulk sample from each of the pits. It is recommended that variability test work is undertaken to correlate the DTR grades with the test work.
- Alternative rosters: The Mining Contractors proposed rosters based on 14 days on and 7 days off, however, the industry in Western Australia is moving towards rosters weighted towards an improved work-life balance with rosters like 8 days on 6 days off. Further analysis is

recommended to examine the cost benefit of adopting alternative rosters to improve recruitment and retention of the workforce.

- Automation: The mining industry in Western Australia is rapidly moving towards autonomation of mining equipment with cost benefits both operationally and in reduction in capital for construction of accommodation at site. The size and duration of the Lake Giles Project is favourable for automation and given that mobilisation to site is more than 3 years away, it is recommended that this option is explored in more detail through early engagement with Mining contractors.
- Blasting trials: The fresh rock at Lake Giles is very hard and massive and will require significant explosive energy to deliver suitable fragmentation for both excavation and processing of the ore. The blasting analysis undertaken for this study will require further analysis and fine tuning as the mine is opened and performance data is gathered.
- Detailed designs: The pit designs presented in this study can be improved upon to reduce waste and improve on operability. It is recommended that further design work is conducted on the final pit and internal stages to improve operability before engaging in the Tender process for the mining services.

26.3. Mineral Processing

The following recommendations arise from the completed metallurgical test work and analyses.

- Further drilling should be performed, in order to produce representative composites based on the ore types in the deposit, in sufficient quantities to allow the performance of a comminution and pilot plant programme. The number of drill holes should be determined by addressing any ore types made evident in the geological modelling so that sufficient sample of each ore type to make feed for a significant flotation programme as well as for a Pilot Plant programme using a master composite. This would be a number of tonnes of sample.
- For the comminution programme CMD recommend that, assuming a payback period of 7 years, at 10 Mtpa would require at least 70 samples, each sample representing 1 Mt of ore.
- The plant will need to be designed to treat a highly abrasive ore.
- The removal of material during dry LIMS processing is small compared to industry benchmarks, so an assessment of the benefit of the dry LIMS processing should be included during a value engineering stage.
- The final size for the grinding circuit will be 80% passing 38 µm.
- Further bench-scale reverse flotation work will be required to optimise reagent selection, dosing and recovery profile. A scale-up factor will be needed in sizing the flotation cells, expected to be in the range of 2 – 2.5 times the laboratory retention times. This should be vendor advised.
- Further assessment of the sulphide mineralisation, in order to determine a mechanism to address desulphurisation and provide a path going forward.
- Further recommendations from the CMD report include:
- Algorithms that correlate ore properties with geological data such as RQD and fracture frequency could be an economical way of defining the ore over time.
- Forecast modelling is recommended to better manage the operating conditions of the circuit if and when the ore blends change.
- MO will need to provide process guarantees for the Vertimills and show methods for the design and scale-up procedures.

26.4. Logistics and Project Infrastructure

Macarthur will need to undertake further investigation and discussions with potential 3rd party providers for power supply, rail infrastructure and access, port infrastructure and access. Continuing engagement with these providers will ensure that the Project meets the proposed development timeframes.

Further works should also be undertaken to see if alternative fuel / power supply facilities can be utilised for the proposed infrastructure to simplify the Project and further reduction the Project’s carbon footprint.

26.5. Environmental

Macarthur needs to commence desktop surveys and baseline environmental surveys as identified in Section 20 to facilitate environmental approval of the project. In accordance with the EPA Guidance notes for flora and fauna surveys, baseline studies need to be undertaken in appropriate seasons with some studies requiring multiple seasons. To ensure the Project meets the development timeframes proposed, Macarthur needs to ensure it is sufficiently resourced to commence field studies this year.

To avoid delays in final grant of approvals, tenure for outstanding project areas needs to be progressed and the development envelope clearly defined.

26.6. Water

Macarthur will need to undertake further investigation of water sources for the Project. To validate the potential water supply sources, field drilling and water testing will be required. All holes are to be geologically and hydro-geologically logged with water strike and flow rate data recorded during drilling.

Sustainability tests will need to be undertaken along with water quality analysis to determine each of the selected areas ability to supply water at the volumes and quality required for continuous mining operations for the Project.

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