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MOLO GRAPHITE MINE

EXPANSION

NI 43-101 TECHNICAL FEASIBILITY STUDY REPORT 2023

MOLO GRAPHITE EXPANSION PROJECT

National Instrument 43-101 Feasibility Study Report

On the Molo Graphite Project located near the village of Fotadrevo, in the Province of Toliara, Madagascar

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Date: September 1, 2023

CAUTIONARY NOTE WITH RESPECT TO FORWARD LOOKING INFORMATION

Certain information and statements contained in this report are “forward looking” in nature. All information and statements in this report, other than statements of historical fact, that address events, results, outcomes, or developments that NextSource Materials Inc. and/or the Qualified Persons who authored this report expect to occur are “forward-looking statements”. Forward-looking statements are statements that are not historical facts and are generally, but not always, identified by the use of forward-looking terminology such as “plans”, “expects”, “is expected”, “budget”, “scheduled”, “estimates”, “forecasts”, “intends”, “anticipates”, “Projects”, “potential”, “believes” or variations of such words and phrases or statements that certain actions, events or results “may”, “could”, “would”, “should”, “might”, or “will be taken”, “occur”, or “be achieved”, or variations, including the negative connotation, of such terms.

Forward-looking statements include, but are not limited to, statements with respect to anticipated production rates, grades, Projected metallurgical recovery rates, infrastructure, capital, operating and sustaining costs; the Projected Life of Mine; the pit design phase development and potential impact on cash flow; estimates of Mineral Reserves and Mineral Resources and realization of such Mineral Reserves and Mineral Reserves; intentions and expectations relating to the Molo Graphite Mine and the Molo Graphite Expansion Project, the future price of metals; government regulations; the maintenance, or renewal of any permits, or mineral tenures, estimates of reclamation obligations that may be assumed, requirements for additional capital, environmental risks, and general business and economic conditions.

All forward-looking statements in this report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled, or predicted. Material assumptions regarding forward-looking statements are discussed in the Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in the Report, the forward-looking statements in this report are subject to the following assumptions:

1. There being no signification disruptions affecting the development, or operation of the mine.
2. The availability of certain consumables and services and other key supplies being approximately consistent with current levels.
3. Labour and materials costs increasing on a basis consistent with current expectations.
4. That all environmental approvals, required permits, licenses and authorizations will continue to be held on the same or similar terms and obtained, as necessary, from the relevant governments and other relevant stakeholders within the expected timelines.
5. Certain tax rates.
6. Assumptions made in Mineral Resource and Mineral Reserve estimates, including geological interpretation grade, recovery rates, metal price assumption, and operational cost, general business and economic conditions.

Forward-looking statements involves known and unknown risks, uncertainties and other factors which may cause the actual results, performance, or achievements to be materially different from any of the future results, performance, or achievements expressed, or implied by forward-looking statements.

These risks, uncertainties and other factors include, but are not limited to, decrease of future metal prices; cost of labour, supplies, fuel and equipment rising; adverse changes in anticipated production, including discrepancies between actual and estimated production, Mineral Reserves, Mineral Resources and recoveries, exchange rate fluctuations, title risks, regulatory risks, and political, or economic developments in 2023; changes to tax rates; risks and uncertainties with respect to obtaining any necessary permits, land use rights and other tenure related risks, risks associated with maintaining and renewing permits and complying with permitting requirements, and other risks involved in the mining industry, as well as those risk factors discussed elsewhere in this Report. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All forward-looking statements herein are qualified by this cautionary statement.

Accordingly, readers should not place undue reliance on forward-looking statements. NextSource Materials Inc. and the Qualified Persons who authored this Report undertake no obligation to update publicly, or otherwise revise any forward-looking statements whether as a result of new information or future events or otherwise, except as may be required by law.

NON-GAAP MEASURES

This Report contains certain non-GAAP measures. The non-GAAP measures do not have any standardized meaning within IFRS and therefore may not be comparable to similar measures presented by other companies. These measures provide information that is customary in the mining industry and that is useful in evaluating the Molo Graphite Mine and Molo Graphite Expansion Project. This data should not be considered as a substitute for measures of performance prepared in accordance with IFRS.

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Abbreviations Symbols Acronyms and Units of Measure	
Aluminium	Al
Ammonium Nitrate / Fuel Oil	ANFO
Ariary Madagascan Currency	MGA
Articulated Dump Truck	ADT
Battery Energy Storage Systems	BESS
Bureau de Recherches Géologiques et Minières (France)	BRGM
Bureau du Cadastre Minier de Madagascar	BCMM
Canadian Institute of Mining	CIM
Capital Expenditure	CAPEX
Caracle Creek International Consulting	CCIC
Carbon	C
Centimetres	cm
Closed Circuit Television	CCTV
Compound Annual Growth Rate	CAGR
Cubic metres	m^3
Cubic metres per hour	m^3/h
Cubic metres per minute	m^3/min
Degrees	°
Degrees Celsius	°C
Discounted Cash Flow	DCF
Dry metric tonnes	dmt
Dry metric tonnes per hour	dmt/h
Dry solids	Ds
Earnings before interest, taxes and depreciation and amortization	EBITDA
Electric Vehicles	EV
Engineer, Procure and Construct	EPC
Engineering, Procurement and Construction Management	EPCM
Ex-works	EXW
Factor of Safety	FoS
Feasibility Study	FS

Abbreviations Symbols Acronyms and Units of Measure	
Feet	‘ or ft
Foreign Exchange	Forex
Free on board	FOB
Front End Loader	FEL
Global Industry Standard for Tailings Management	GISTM
Global Positioning System	GPS
Grams	g
Grams per tonne	g/t
Greater than	>
Greater than or equal to	≥
Hectares	ha
High Density Polyethylene	HDPE
Hours	h
Independent Power Producer	IPP
Inductively Coupled Plasma (Assay)	ICP
Internal Diameter	ID
Iron	Fe
Joint Venture Agreement	JVA
Kilobar	kbar
Kilograms	kg
Kilograms per tonne	kg/t
Kilometres	km
Kilovolt-amps	kVA
Less than	<
Less than or equal to	≤
Life of Mine	LoM
Light Delivery Vehicle	LDV
Light Emitting Diode	LED
Madagascar Ariary (currency)	MGA
Madagascar Ministry of Environment’s Office National pour l'Environnement (the National Office for the Environment)	ONE

Abbreviations Symbols Acronyms and Units of Measure	
Manganese	Mn
Mean Annual Evaporation	MAE
Mean Annual Precipitation	MAP
Mesh size (of screen or sieve)	Mesh
Methyl Isobutyl Carbinol	MIBC
Metres	m
Metres above Mean Sea Level	MAMSL
Micrometres	µm
Millimetres	mm
Million Cubic Metres	Mm^3
Million tonnes	Mt
Million tonnes per annum	Mtpa
Million Watts	MW
Mine Rehabilitation and Closure Programme	MRCP
Mineral Reserve Estimate	MRE
National Instrument 43-101	NI-43-101
Net Smelter Royalty	NSR
Operating Expenditure	OPEX
Original Equipment Manufacturer	OEM
Particle size distribution where 80% of particles in a stream are larger than the size indicated	P_{80}
Parts per million	PPM
Percent	%
Percent graphitic carbon	% C(g)
Percent total carbon	% C(t)
Pollution Control Dam	PCD
Preliminary Economic Assessment	PEA
Qualified Persons	QPs
Quality Assurance / Quality Control	QA/QC
Return Water Dam	RWD
Run of Mine	ROM

Abbreviations Symbols Acronyms and Units of Measure	
South African Rand (currency)	ZAR
Square metres	m^2
Storm Water Management Plan	SWMP
Tailings Storage Facility	TSF
Techno-economic model	TEM
Terra-Watt	TW
Thousand Pascals	kPa
Thousand tonnes per annum	ktpa
Thousand tonnes per month	ktpm
Thousand Watts	kW
Three dimensional	3D
Tonnes (1,000 kg) (metric ton)	t
Tonnes per annum	tpa
Tonnes per cubic metre	t/m^3
Toronto Stock Exchange	TSX
United States Dollars (currency)	USD, US$
Universal Transverse Mercator	UTM
Variable Speed Drive	VSD
X-Ray Fluorescence	XRF

1 SUMMARY

1.1 Introduction

NextSource Materials Inc. ("NEXT" or the "Company") is a mineral exploration and development company based in Toronto, Canada. The Company is currently focused on the development of its 100% owned Molo Graphite Project using a phased approach to de-risk its development plans, NEXT has constructed the first phase during 2022 and 2023 and is ramping up production to nameplate capacity of 17,000 tpa. This Technical Report sets out the development plans to increase capacity to 150,000 tpa.

The Molo deposit is situated 160 km south-east of the city of Toliara, in the Tulear region of south-western Madagascar. The Molo deposit occurs in a sparsely populated, dry savannah grassland region, which has easy access via a network of seasonal secondary roads radiating outward from the village of Fotadrevo. Fotadrevo has an all-weather airstrip and access to a road system that leads to the regional capital, (and port city) of Toliara and the Port of d'Ehoala at Fort Dauphin.

Geologically, the Molo deposit is situated in the Bekily block, (Tolagnaro-Ampanihy high grade metamorphic province) of southern Madagascar. The Molo deposit is underlain predominantly by moderately to highly metamorphosed and sheared graphitic (biotite, chlorite and garnet-rich) quartzo-feldspathic schists and gneisses, which are variably mineralised. Near surface rocks are oxidised, and saprolitic to a depth, usually of less than 5m.

Molo was one of several surficial graphite trends discovered by the Company in early January 2012, previously known as Energizer Resources. The Molo deposit was originally drill tested in 2012, with an initial seven holes being completed. Resource delineation, drilling and trenching on Molo took place between May and November of 2012 and allowed for a maiden Indicated and Inferred Resource to be stated in December of the same year. This maiden Mineral Resource Estimate ("MRE") formed the basis for a PEA, which was undertaken by DRA Projects in 2013 (the "Molo 2013 PEA").

The positive results of the Molo 2013 PEA led the Company to undertake another phase of exploratory drilling and sampling in 2014, which was done under the supervision of Caracle Creek International Consulting (CCIC). This phase of exploration was aimed at improving the geological confidence of the deposit and it's contained mineral resources and included an additional 32 diamond drill holes, (totalling 2,063m) and 9 trenches (totalling 1,876m).

CCIC were subsequently engaged to update the geological model and resource estimate. The database on which the updated model and resource estimate was based contained 80 drill holes, (totalling 11,660m, and 35 trenches (totalling 8,492m). This new resource formed the basis of the first Molo Feasibility Study, which was based on ore processing capacity of 860 ktpa (the "Molo 2015 FS").

In 2017, the Company released the results of an updated Molo Feasibility Study, which was based on an ore processing capacity of 240 ktpa (the "Molo 2017 FS").

On September 27 2019, the Company reported the results of an updated Feasibility Study (the “2019 FS Technical Report”) consisting of two phases: Phase 1 as a fully operational and sustainable graphite mine with a permanent processing plant capable of processing 240,000 tpa of ore, producing approximately 17,000 tpa of graphite concentrate over a 30 year Life of Mine (LoM), and Phase 2 as a modular expansion to a production capacity of 45,000 tpa of graphite concentrate in Year 3.

On March 29, 2021, the Company announced the initiation of the construction process for Phase 1 of the Molo Graphite Mine with a processing plant capable of processing 240,000 tpa of ore, producing approximately 17,000 tpa of graphite concentrate.

The demand for graphite is predominantly driven by the development of the lithium-ion battery market to support the growth of the electric vehicle market. Market research completed by various industry analysts has resulted in a consensus view that a significant increase in demand is expected over the next decade. The principal uncertainty, however, is the timing of the increase in demand.

The Company has announced graphite concentrate off-takes with a Japanese Trader and with ThyssenKrupp. The Company is in the process of formalizing additional sales agreements. To ensure that the Company remains ahead of the competition and to appropriately align its production capacity with future market demand, the Company has opted for a flexible modular development approach yielding optimal cashflow and return metrics while preserving suitable flexibility to enable the Company to rapidly respond to market demands.

As such, the Company requested the completion of a Feasibility level study for an enhanced expansion of the Molo Graphite Mine.

This Feasibility Study Report, (hereinafter referred to as the “FS”), considers the construction of an additional standalone processing plant that increases the steady-state production rate from 17,000 tpa to 150,000 tpa of SuperFlake® graphite concentrate over a 25 year LoM. This FS utilises the knowledge base of the 2019 FS Technical Report and to a larger extent utilises the recently completed Phase 1 of the Molo Graphite Mine. Where applicable and relevant, amounts from the 2019 FS Technical Report were updated for current market conditions, and experience gained during the construction of Phase 1. The expansion costs are, therefore, deemed accurate to a Feasibility Study level.

The Company has every intention of completing the expansion of the Molo Mine in close succession to the completion of Phase 1 and has the mineral resources to support further increases of the mining and beneficiation capacity in order to satisfy further increases in market demand.

1.2 Project Location

The Molo deposit is located some 160 km south-east of Madagascar’s administrative capital (and port city) of Toliara, in the Tulear region and about 220 km north-west of Fort Dauphin and is approximately 13 km north-east of the local village of Fotadrevo.

1.3 Project Description

Phase 1 of the Molo Graphite Mine consists of the construction of a Greenfield open pit mine, a processing plant capable of processing 240,000 tpa of ore, producing 17,000 tpa of graphite concentrate, and all supporting infrastructure including water, fuel, power, co-disposal tailings, buildings and permanent accommodation.

This FS considers the addition of a stand-alone processing plant capable of processing 2,640,000 tpa of ore, increasing production to 150,000 tpa of graphite concentrate over a 25 year LoM, and all supporting infrastructure including water, fuel, power, co-disposal tailings facility, buildings and permanent accommodation.

1.4 Property Description and Ownership

1.4.1 Property Description

The property includes 790 claims and an area totalling 308.6 km^2.

The Molo deposit is centred on UTM co-ordinates 495,289 easting 7,345,473 northing (UTM 38S, WGS 84 datum), and is located 11.5 km east-north-east of the town of Fotadrevo.

The property is within Exploitation / Mining Permit PE #39807 which covers an area of 175 km^2 or 17,500 hectares (“ha”), and Exploration Permits PR #39806 and PR #39810 which cover areas of 96.1 km^2 (9,609 ha) and 37.5 km^2 (3,750 ha), respectively.

1.4.2 Ownership

On December 14, 2011, the Company entered into a Definitive JVA with Malagasy Minerals Limited, (hereinafter referred to as "Malagasy"), a public company on the Australian Stock Exchange, to acquire a 75% interest to explore and develop a group of industrial minerals, including graphite, vanadium and approximately 25 other minerals. On October 24, 2013, the Company signed a memorandum of understanding with Malagasy to acquire the remaining 25% interest in the land position.

On April 16, 2014, the Company signed a Sale and Purchase Agreement and a Mineral Rights Agreement with Malagasy to acquire the remaining 25% interest. Malagasy retains a 1.5% Net Smelter Return Royalty ("NSR").

CCIC reviewed a copy of the Contrat d’Amodiation pertaining to this right and are satisfied that the rights to explore this permit have been ceded to the Company, or one of its Madagascar subsidiaries.

The Project was located within Exploration Permit PR #3432 as issued by the Bureau de Cadastre Minier de Madagascar (“BCMM”) pursuant to the Mining Code 1999 (as amended), and its implementing decrees. On January 18, 2019, Permit PR #3432 was transformed into two Exploration Permits (PR #39806 and PR #39810) and an Exploitation Permit (PE #39807) by the Ministry of Mines, with the official permit being granted to the Company by the BCMM on February 14, 2019.

Mineral Resources and Reserves delineated in Sections 14 and 15 of this Report are entirely within the bounds of Exploitation Permit PE #39807. The Company holds the exclusive right to exploit / mine and explore for graphite within this license area for a period of 40 years and can renew the license several times for a further period of 20 years upon each renewal.

The Company holds the exclusive right to explore for a defined group of industrial minerals within Exploration Permits PR #39806 and PR #39810. These industrial minerals include the following: Vanadium, Lithium, Aggregates, Alunite, Barite, Bentonite, Vermiculite, Carbonatites, Corundum, Dimensional stone, (excluding labradorite), Feldspar (excluding labradorite), Fluorspar, Granite, Graphite, Gypsum, Kaolin, Kyanite, Limestone / DoLoMite, Marble, Mica, Olivine, Perlite, Phosphate, Potash–Potassium minerals, Pumice Quartz, Staurolite, Zeolites.

Companies in Madagascar first apply for an exploration mining permit with the BCMM, a government agency falling under the authority of the Minister of Mines. Permits under usual circumstances are generally issued within a month. The number of squares varies widely by claim number.

The updated Decret requires the payment of annual administration fees of Permits Research of 15,000 Ariary (MGA) for exploitation permits in years 1 and 2. Annual fees increase by multiplying by a factor equivalent to the number of years (plus 1) that the company has held the permit. Exploration permits have an updated duration of 5 years, with the possibility of 2 renewals of an additional 3 years each.

Payments of the administration fees are due each year on March 31, along with the submission of an activity report. Each year the Company is required to pay a similar, although increasing amount to maintain the claims in good standing.

Reporting requirements of exploration activities carried out by the title holder on an Exploration Permit are minimal. A title holder must maintain a diary of events and record the names and dates present of persons active on the Project. In addition, a site plan with a scale between 1/100 and 1/10,000 showing “a map of the work completed” must be presented. CCIC is of the opinion that the Company is compliant in terms of its commitments under these reporting requirements.

The Project has not been legally surveyed; however, since all claim boundaries conform to the pre-determined rectilinear LaBorde Projection grid, these can be readily located on the ground by use of Global Positioning System (“GPS”) instruments. Most current GPS units and software packages do not, however, offer LaBorde among their available options and therefore defined shifts must be employed to display LaBorde data in the WGS 84 system. For convenience, all the Company’s positional data is collected in WGS 84, and if necessary, converted back to LaBorde.

1.4.3 The Company’s Royalties

Prior to the expansion 5% of revenue and 3.5% of the FOB value of sales is payable to the Government of Madagascar. After the expansion, and successful application of the LGIM, the royalty decreases to 2.5% of revenue and 3.5% of FOB value of sales.

Vision Blue Resources Limited (“Vision Blue”) retains a 3% gross revenue royalty, and Malagasy retains a 1.5% net smelter return royalty on the Project.

1.5 Exploitation and Exploration Permits

Exploitation Permit PE #39807 (175 km^2) and Exploration Permits PR #39806 and PR #39810 are held under the name of a subsidiary of the Company called ERG (Madagascar) Ltd. S.A.R.L.U. and was granted to the Company by BCMM on February 14, 2019.

1.6 Geologic Setting and Mineralization

The Molo deposit occurs within the regional Ampanihy Shear Zone. The most conspicuous feature of rocks found within this shear zone is their well-developed north-south foliation and vertical to sub-vertical nature. Martelat et al. (2000) states that this observed bulk strain pattern is clearly related to a transpressional regime during bulk horizontal shortening of heated crust, which resulted in the exhumation of lower crustal material.

The Project area is underlain by supracrustal and plutonic rocks of late Neoproterozoic age that were metamorphosed under upper amphibolite facies and deformed with upright north-northeast-trending structures. The supracrustal rocks involve migmatitic (± biotite, garnet) quartzo-feldspathic gneiss, marble, chert, quartzite, and amphibolite gneiss. The metaplutonic rocks include migmatitic (± hornblende / diopside, biotite, garnet) feldspathic gneiss of monzodioritic to syenitic composition, biotite granodiorite, and leucogranite.

1.7 Exploration

Significant exploration was carried out in 2011 that included activities of prospecting, grab and trench sampling, and diamond drilling. The exploration programme included the use of geophysical techniques to delineate additional graphite mineralisation. Initial graphitic carbon results from the 2011 trenching were encouraging in that they showed multiple graphic horizons present in each zone, of significant widths and grades. Additional trenching was undertaken on Molo during 2013 as part of a bulk sampling exercise. Subsequently an additional nine trenches, totalling 1,876m, have been excavated as part of the 2014 exploration programme.

The 2011 diamond drilling included several wide spaced holes on the Molo deposit. Most of these drill holes, over a strike length of 1.2 km, that intersected graphitic mineralisation to a vertical depth of 75m, with down-hole thicknesses of between 60m and 150m in width. Additionally, forty-one diamond drill holes, comprising 8,502.7m of diamond drilling was completed on Molo during 2012. During 2014 an additional 32 diamond drill holes, totalling 2,063m was completed. With this most recent drill programme, a total of 80 diamond drill holes, totalling 11,660m, was completed on Molo, and these were used for the mineral resource estimations.

No additional exploration was required for this FS.

1.8 Mineral Resource Estimate

The block model used to generate the current Mineral Resource Estimate for this FS has an effective date of 01 September 2014. The Resource is based on 80 core drill, 35 trenching, which produced 8,643 samples.

The Resource block model was produced using (Datamine Studio RM).

The current Resource was constrained by a 2% Cg cut-off grade, to a depth of 350m below surface.

Mineral resource estimates for the Molo Graphite Mine remains open along strike and to depth. (Table 1).

Table 1: Mineral Resource Statement

Molo Mineral Resource Statement – 1 September 2023				
Classification	**Material Type**	**Resource Tonnes**	**Grade**	**Contained Carbon Graphite**
		(kt)	**(% Cg)**	**(kt)**
Measured	"Low-Grade"	13,048	4.64	605
Measured	"High-Grade"	10,573	8.40	888
Total Measured		**23,622**	**6.32**	**1,493**
Indicated	"Low-Grade"	39,539	4.73	1,871
Indicated	"High-Grade"	37,207	7.86	2,925
Total Indicated		**76,746**	**6.25**	**4,796**
Measured + Indicated	"Low-Grade"	52,588	4.71	2,476
Measured + Indicated	"High-Grade"	47,780	7.98	3,813
Total Measured + Indicated		**100,367**	**6.27**	**6,289**
Inferred	"Low-Grade"	24,233	4.46	1,081
Inferred	"High-Grade"	16,681	7.70	1,285
Total Inferred		**40,915**	**5.78**	**2,366**

Notes:

(1) Mineral resources have been classified using the 2014 CIM Definition Standards.

(2) Mineral resources are reported inclusive of mineral reserves.

(3) "Low Grade" mineral resources are resources in a low-grade zone and stated at a cut-off grade of 2% Cg with no upper limit.

(4) "High Grade" mineral resources are resources in a high-grade zone and stated at a cut-off grade of 4% Cg with no upper limit.

(5) Eastern and western high-grade assays are capped at 15% Cg.

(6) A relative density of 2.36 tonnes per cubic meter (t/m^3) was assigned to the mineralized zones for the mineral resource tonnage estimation.

(7) Totals may not represent the sum of the parts due to rounding.

(8) Mineral resources are defined as surface mineable only.

(9) Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that any mineral resource will be converted into a mineral reserve.

(10) % Cg = percentage Carbon Graphite.

(11) The mineral resource estimates may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The total Measured and Indicated Resource is estimated at 100.37 Mt, grading at 6.27% carbon. Additionally, an Inferred Resource of 40.91 Mt, grading at 5.78% carbon is stated.

1.9 Mineral Reserve Estimate

The Molo deposit will be mined using conventional open pit mining methods consisting of drilling, blasting, loading, and hauling. Ore will be hauled to the primary crusher and waste rock and tailings will be placed in a co-disposal tailings facility (TSF).

The Mineral Reserves for Molo were prepared by Eugene de Villiers, Pr.Eng., ECMA Consulting; a Qualified Person as defined under National Instrument 43-101.

Development of the LoM plan included pit optimization, pit design, mine scheduling and the application of modifying factors to the Measured and Indicated Mineral Resources. The reference point for the Mineral Reserves is the feed to the primary crusher. The tonnages and grades reported are inclusive of mining dilution, geological losses, and operational mining losses.

Table 2 below shows the Mineral Reserves that have been estimated for the Molo Mine, which include 21.3 Mt of Proven Mineral Reserves at an average grade of 6.25% Cg and 32.4 Mt of Probable Mineral Reserves at an average grade of 6.09% Cg for a total of 53.7 Mt of Proven and Probable Mineral Reserves at an average grade of 6.15% Cg. To access these Mineral Reserves, 19.2 Mt of waste rock must be mined, resulting in a strip ratio of 0.3:1.

Table 2: Mineral Reserves

Molo Mineral Reserve Statement – 1 September 2023				
Classification	**Material Type**	**Ore**	**Grade**	**Contained Carbon Graphite**
		(kt)	**(% Cg)**	**(kt)**
Proven	“High Grade”	15,489	7.00	1,085
	“Low Grade”	5,845	4.25	248
	Total	21,334	6.25	1,333
Probable	“High Grade”	24,734	6.64	1,642
	“Low Grade”	7,677	4.32	331
	Total	32,412	6.09	1,973
Total Reserves		**53,746**	**6.15**	**3,306**

Notes:

(1) Mineral reserves have been classified using the 2014 CIM Definition Standards

(2) Assumes that all modifying factors have been applied, including mining losses of 5% and mining dilution of 3%.

(3) Assumes a reserve cut-off grade of 3% Cg has been applied, with all material below this cut-off grade treated as waste."Low Grade" mineral reserves are classified as ore with a grade ≥3% Cg and ≤5% Cg.

(5) "High Grade" mineral reserves are classified as ore with a grade >5% Cg.

(6) Totals may not represent the sum of parts due to rounding.

(7) % Cg = percentage Carbon Graphite.

(8) The estimate of mineral reserves may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

1.10 Mining

The mine design, scheduling, and costing for the project was completed by independent specialists. During the development of Molo Phase 1, open pit mining was identified as the most appropriate and cost effective method of mining. The open pit mining method that is currently employed at Molo is proven to be adequate and, has been extended for the FS due to the surficial, lateral expanse and the massive nature of the Molo deposit.

Mining will be carried out by means of drilling and blasting 4.0m thick benches, and loading of the 4m benches will be done by means of FEL and Excavators. The current loading fleet consists of 3 FEL with $3m^3$ buckets and 2 diesel-powered hydraulic excavators equipped with $1.3m^3$ buckets, and will increase by an additional 2 diesel powered hydraulic excavators equipped with x $4.5m^3$ buckets to support full production. The hauling fleet of 37t rigid frame mining trucks will increase to 14, 37t rigid frame mining trucks to support full production. A Front End Loader and 2 x 37t rigid frame mining trucks will be utilized to load and haul a blend of ore from the Run of Mine (ROM) stockpile to the ROM tip.

The mine will operate on two 8 hour shifts, 5 days per week, while the process plant will operate 24 hours per day, 365 days per year. A crushed ore bin will be filled before the mine shuts down for the evenings and weekends, and can be fed by the front-end loader at the ROM stockpile.

The ultimate pit designed for the Project considers 15m wide haul ramps for double-lane traffic, a maximum ramp grade of 10%, and a minimum mining width of 30m.

Open House Management Solutions (OHMS) carried out an open pit slope investigation and stability assessment in 2014.

The final pit is approximately 1,500m long and 550m wide on surface. The total surface area of the pit is approximately 83 ha. The pit contains 4 independent ramp systems which are required for pit phasing. The deepest part of the pit is at the 345m elevation, in the middle of the pit, where the total depth of the pit from surface reaches 219m.

The deposit will be mined from south to north to maximize the ore grade to be processed.

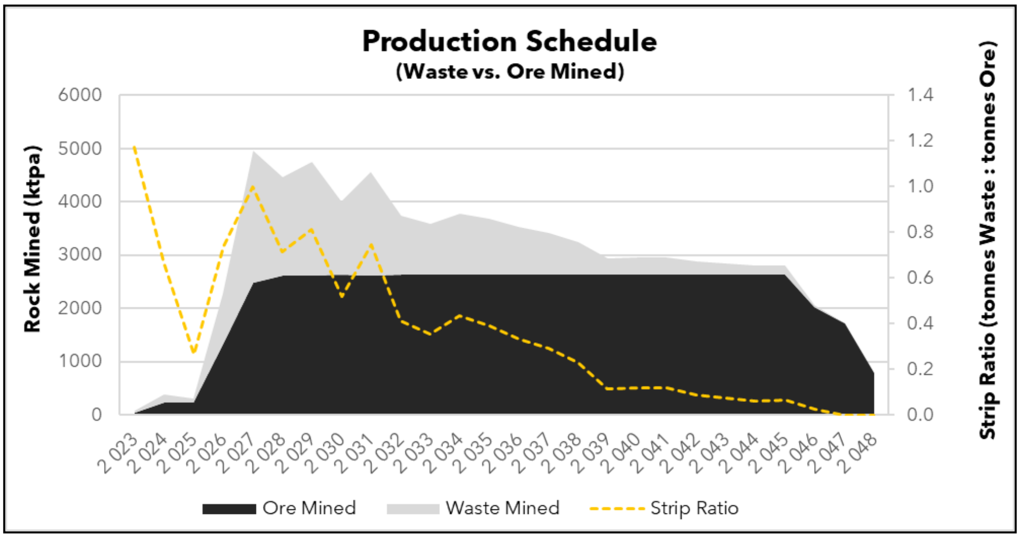
A mine production plan has been prepared using Deswik software. The mine plan has been scheduled and reported on a monthly/annual basis for the LoM of 25 years. The mine plan starts from September 2023 and incorporates the Phase 1 production.

The mine plan aims to produce 150,000 tpa of concentrate, and targets the nominal mill throughput capacity of 347 tph. This results in a maximum mill feed of 2.6 Mtpa considering an overall mill utilization of 85%.

During the 25 year LoM, the total ore mined from the open pit peaks at 2.64 Mt in Year 2040 and averages 2.47 Mtpa for Phase 2 (June 2026 – June 2048). The average diluted Cg grade ranges from 6.7% to 9.7% for the first 2.75 years (Phase 1), and averages 6.0% in the final 22.25 years. The mine plan is successful at achieving the targeted concentrate production.

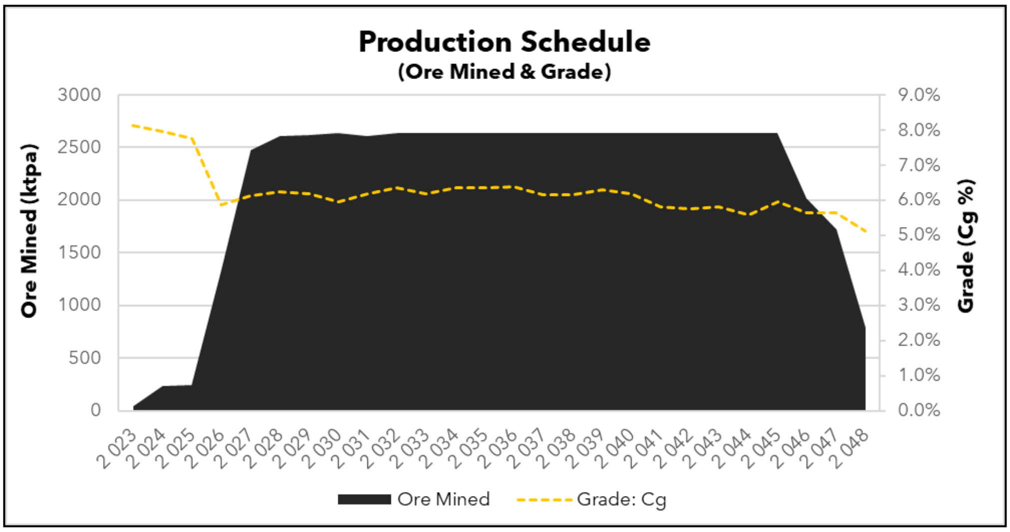
The production schedule graph provides a visual representation of the planned extraction of graphite ore over the LoM time frame, as tabulated in the "Mine Physicals" Table 3 below.

The annualised LoM waste and ore production profile is illustrated in Graph 1 below.



Graph 1: Production Schedule (Waste vs. Ore Mined)

The annualised LoM ore mined and grade profile is illustrated in Graph 2 below.



Graph 2: Production Schedule (Ore Mined & Grade)

A summary of mine physicals for the total LoM is shown in Table 3 below.

Table 3: Mine Physicals

Metric	Unit of Measurement	Result
Waste Mined (LoM Total)	Mt	19.2
Ore Mined (LoM Total)	Mt	56.3
Strip Ratio	ratio (t_{waste}: t_{ore})	0.3
Average ROM Grade	% Cg	6.07%
Contained Graphite (LoM Total)	Mt	3.094
Life of Mine	years (active)	24.8
Life of Mine	years (at steady-state production)	21.7
Nameplate Ore Production	Mtpa	2.6
Average Annual Ore Production	Mtpa	2.2

Ore mined from the open pit is fed to the onsite concentrator plant(s), subject to minimum levels of emergency ore stockpiles maintained on surface. An emergency stockpile of 15,000t of ROM is maintained during Phase 1, which is increased to 25,000t of ROM for full production.

1.11 Metallurgical Test Work

The FS is based on a full suite of metallurgical test work performed by SGS Canada Metallurgical Services Inc. in Lakefield, Ontario, Canada for the 2014 FS and remains the same for this FS.

These tests included laboratory scale metallurgical work and a 200t bulk sample / pilot plant program. The laboratory scale work included comminution tests, process development and optimization tests, variability flotation, and concentrate upgrading tests. Comminution test results place the Molo ore into the very soft to soft category with low abrasivity. A simple reagent regime consists of fuel oil number 2 and methyl isobutyl carbinol at dosages of approximately 120 g/t and 195 g/t, respectively. A total of approximately 150 open circuit and locked cycle flotation tests were completed on almost 70 composites as part of the process development, optimization, and variability flotation program.

The metallurgical programs culminated in a process flowsheet that is capable of treating the Molo ore using proven mineral processing techniques and its robustness has been successfully demonstrated in the laboratory and pilot plant campaigns.

The metallurgical programs indicated that variability exists with regards to the metallurgical response of the ore across the deposit, which resulted in a range concentrate grades between 88.8% total carbon and 97.8% total carbon. Optical mineralogy on representative concentrate samples identified inter-layered graphite and non-sulphide gangue minerals as the primary source of impurities. The process risk that was created by the ore variability was mitigated with the design of an upgrading circuit, which improved the grade of a concentrate representing the average mill product of the first five years of operation from 92.1% total carbon to 97.1% total carbon.

The overall graphitic carbon recovery into the final concentrate is 88.3%.

The average composition of the combined concentrate grade is presented in Table 4. The size fraction analysis results were converted into a grouping reflecting a typical pricing matrix, which is shown in Table 5.

All assays were completed using control quality analysis and cross checks were completed during the mass balancing process to verify that the results were within the estimated measurement uncertainly of up to 1.7% relative for graphite concentrate grades greater than 90% total carbon.

Vendor testing including solid-liquid separation of tailings and concentrate, screening and dewatering of concentrate, and drying of concentrate was completed successfully.

Table 4: Metallurgical Data - Flake Size Distribution and Product Grade

Product Size	% Distribution	Product Grade (%) Carbon
+48 mesh (jumbo flake)	23.6	96.9
+65 mesh (coarse flake)	14.6	97.1
+80 mesh (large flake)	8.2	97.0
+100 mesh (medium flake)	6.9	97.3
+150 mesh (medium flake)	15.5	98.1
+200 mesh (small flake)	10.1	98.1
-200 mesh (fine flake)	21.1	97.5

Table 5: Pricing Matrix - Flake Size Distribution Grouping and Product Grade

Product Size	% Distribution	Product Grade (%) Carbon
>50 mesh	21.8	96.9
-50 to +80 mesh	27.3	97.0
-80 to +100 mesh	8.0	97.2
-100 mesh	42.9	97.6

1.12 Recovery Methods

The process design is based on an annual feed plant throughput capacity of 2,640,00 tpa at a nominal head grade of 6.07% C (t) producing a combined product output of 150,000 tpa at steady state.

The ore processing circuit consists of two stages of crushing which comprises of a cone crusher in the primary crushing circuit, followed by a secondary cone crushing circuit. The secondary cone crusher is operated in a closed circuit with double deck classification screens. Crushing is followed by primary milling and screening, graphite recovery by froth flotation and concentrate upgrading circuit by attritioning, and graphite product and tailings effluent handling unit operations. The crusher circuit is designed to operate 365 days per annum for 24 hours per day at ±55% utilization. The crushed product (P_{80} of approximately 13 mm) passes through a surge bin from where it is fed to the milling circuit.

The milling and flotation circuits are designed to operate 365 days per annum for 24 hours per day at 92% utilization. Two single stage primary ball milling circuits running in parallel are employed, incorporating a closed-circuit classifying screen and a scalping screen ahead of the mill.

The scalping screens undersize feeds into a flash flotation cells before combining with the mill discharge material. Scalping and classification screen oversize are the fed to the primary mills.

Primary milling is followed by rougher flotation which, along with flash flotation, recovers graphite to concentrate from the mainstream. Rougher flotation employs six forced-draught trough cells. The recovered concentrate is then upgraded in the primary, fine-flake and attritioning cleaning circuits to an estimated final product grade of above 94% C (t). The primary cleaning circuit consists essentially of a dewatering screen, a polishing ball mill, a column flotation cell and flotation cleaner / cleaner scavenger trough cells.

The primary cleaner column cell concentrate gravitates to a 212 µm classifying screen, from where the large-flake oversize stream is pumped to a high-rate thickener located in the concentrate attritioning circuit whilst the undersize is pumped to the fine-flake cleaning circuit.

The fine flake cleaning circuit consists primarily of a dewatering screen, a polishing ball mill, a column flotation cell and flotation cleaner / cleaner scavenger trough cells.

The attritioning cleaning circuit employs a high-rate thickener, an attritioning stirred media mill, a column flotation cell and flotation cleaner / cleaner scavenger trough cells. Fine flake column concentrate is combined with the +212 µm primary cleaner classifying screen oversize as it feeds the attritioning circuit thickener. Concentrate from the attrition circuit is pumped to the final concentrate thickener.

The combined fine flake cleaner concentrate and the +212 µm may also be processed through the secondary attrition circuit which consists of a dewatering screen, an attrition scrubber, column flotation cell and cleaner scavenger trough cells. Concentrate from this circuit is pumped to the final concentrate. The secondary attrition circuit is optimal.

Combined rougher and cleaner flotation final tailings are pumped to the combined tailings sump. The material from the combined tailings sumps is pumped to the final tailings thickener. Thickened final tailings then gets pumped to the filter press to produce a filter cake to be stockpiled. Thickened final concentrate is pumped to a filter press for further dewatering before the filter cake is stockpiled prior to load and haul.

The concentrate thickener underflow is pumped to a linear belt filter for further dewatering and fed to a diesel fired rotary kiln for drying. The dried concentrate is then screened into four size fractions:

- +48 mesh.
- -48 + 80 mesh.
- -80 +100 mesh.
- -100 mesh.

The various product sizes are bagged and readied for shipping.

Chemical reagents are used throughout the froth flotation circuits and thickeners. Diesel fuel is used as collector and liquid Methyl Isobutyl Carbinol (“MIBC”) is used as frother within the flotation circuits.

Diesel collector is pumped from a diesel storage isotainer, from where it enters a manifold system which supplies multiple variable speed peristaltic pumps which discretely pump the collector at set rates to the various points-of-use within the flotation circuits.

MIBC frother is delivered by road to an isotainer. A manifold system on the storage isotainer supplies multiple variable speed peristaltic pumps, which discretely pump the frother at set rates to the various points-of-use within the flotation circuits.

Flocculant powder (Magnafloc 24) is delivered by road to the plant reagent store in 25 kg bags. The bags are collected by forklift as required and delivered to a flocculant mixing and dosing area. Here the flocculant is diluted as required using parallel, duplicate vendor package automated make-up plants, each one being dedicated to supplying the concentrate and tailings thickeners due to the flocculant types required being different for each application. Variable speed peristaltic pumps discretely pump the flocculant at set rates to the thickeners’ points-of-use.

Coagulant powder, (Magnafloc 1707), for thickening enhancement is handled similarly to the flocculant as described above, the exception being that a single make-up system is provided to supply both the concentrate and tailings thickeners. Again, variable speed peristaltic pumps discretely pump the coagulant at set rates to the thickeners’ points-of-use.

1.13 Infrastructure

Due to the remote location, the key infrastructure requirements for the project include tailings storage facility, power generation, water collection from well fields, explosive magazine and workforce accommodation, as well as all non-process buildings and infrastructure.

1.13.1 Tailings Storage Facility

Development of the TSF is to be a dry-stack facility with an upstream construction method. The waste material is to be transported to the TSF from the plant area by an overland conveyor system, then mechanically placed and compacted within the footprint of the TSF. The waste rock will form the outer containment wall region with the fines being spread and compacted behind this containment wall. The contact region between the coarse and fines will generally be a mix of material and is viewed as a transition zone between the course outer and the finer inner zone.

1.13.2 Mine Infrastructure

The mine infrastructure required to support the mining operation established during Phase 1 comprises of the explosive magazine, the access road to the explosive magazine, main haul roads (ex-pit) and the ROM tip ramp. Minor upgrades required for the expansion are:

- Two additional explosive magazine storage containers.
- New ROM tip ramp.

1.13.3 Process Plant Infrastructure

The process plant infrastructure required to support the operation established during Phase 1 comprises of offices, workshop, laboratory, change house and stores. Upgrades required for the expansion are:

- New offices for 90 people.
- Expanded laboratory and storage.
- New change house.
- Additional reagent and product storage.

1.13.4 Shared Infrastructure and Services

The shared infrastructure and services required to support the operation established during Phase 1 comprises of water including wellfield, power generation from solar and thermal sources, access roads, gate house, accommodation camp, storage yards, fuel storage and distribution and information and communication backbone. Upgrades required for the expansion are:

- Expanded wellfield water from 3 to 25 boreholes and associated reticulation.
- Expanded thermal and solar / BESS installation.
- Expanded accommodation camp adding 248 beds.
- Additional storage facilities.

- Expanded fuel storage and re-fuelling station.
- Expanded information and communication backbone.

1.13.5 Product Transport

The Port of Tulear is viewed as the primary option on the basis of its proximity to Molo with an economical and logistical advantage. In previous studies completed prior to the commissioning of Molo Phase 1, the port at Fort Dauphin was expected as the primary port, however the existing road from Molo to Tulear Port was effectively used for transportation of product and goods during the commissioning and ramp up of production and proved to be viable for Phase 1 volumes of cargo movement.

A logistics company, currently managing Phase 1 of the Molo project, has been selected to manage the global logistics requirements of the expansion project. Additionally, port operational partnership and transportation partnership has been established for Phase 1 and these will be extended for the expansion project.

After completion of the expansion project the combined concentrate for exportation will be 150,000t. The all-road route from the mine to Tulear port has a bridge restriction of 18t across the river at Tongabory.

To eliminate the bridge restrictions at Tongabory the option to barge loaded trucks from Soalara to Tulear Port was identified for the expansion project. This increases the loading capacity of trucks from 18 to 28t and reduces the total trucking distance.

The barging operation is envisaged to be implemented for the expansion project or, even before during Phase 1 of the project, reducing total fleet requirements and daily convoys. This route, at 210 km to Soalara, is significantly shorter than any other route, thus yielding lower fuel costs and shorter on road transit times. From Soalara the trucks will be moved on the landing craft to Tulear Port, with a short hauling distance from landing dock to warehouse, or direct container loading and exportation.

During implementation of the expansion project the Port of Tulear and Port d’Ehoala at Fort Dauphin will be used, the latter for heavyweight equipment importation.

1.14 Marketing

Independent marketing studies have been sourced as reference material for the FS. These credible sources, (Benchmark Minerals Intelligence and Fastmarkets), have provided short and long term forecasts for an array of flake graphite products for various commercial size fractions and purities. The FS focuses on the four main product sizes, with a premium applied for higher Cg content.

A LoM average selling price of US$1,191.00/t is the volume weighted average sales price for the various flake sizes and grades of graphite concentrate that are expected to be produced from the Molo deposit. This price used was based on current market prices provided by UK-based, commodity price reporting agencies Benchmark Minerals Intelligence and Fast Markets, who are recognized as leaders in providing independent and unbiased market research, pricing trends and demand and supply analysis for the natural flake graphite market.

1.15 CAPEX and OPEX

A Class 3 estimate was produced in support of the brownfield's expansion of the Molo mine, with a target accuracy of –15% to +25%. An area-level summary of the total Project CAPEX is provided in Table 6, equating to US$161.7 million (Real). Capitalised OPEX of US$25.2 million is excluded from Table 6 but, is capitalised for the purposes of the economic analysis.

Table 6: Project CAPEX (LoM Total)

Item	Total (US$ '000, Real)
Direct Capital Costs	**95,659**
Open-Pit Mining	3,625
Processing Plant	58,359
On-Site Infrastructure	33,675
Indirect Capital Costs	**44,814**
Project Management	18,395
Owner's Cost	20,325
Other Capitalised Cost (excl. Capitalised OPEX)	6,094
Provisions	**21,227**
Contingency	21,227
Total: Project CAPEX (excl. Capitalised OPEX)	**161,700**

Total sustaining capital amounts to US$205 million over the LoM and includes the staged development of the various TSF lifts (US$171 million), as well as replacement of key equipment in the processing plant (US$25.3 million) and the open-pit fleet (US$9.2 million).

The LoM average all-in-sustaining cost (FOB, Tulear) is included in Table 7.

Table 7: Operating Costs (LoM Average)

Item	LoM Total	Unit Cost	Unit Cost
	(US$M, Real)	(US$ / t ore mined)	(US$ / t concentrate)
Open-Pit Mining	190	3.38	61.41
Processing	454	8.06	146.66
On-Site Infrastructure	430	7.65	139.13
G&A (Site)	140	2.50	45.39
Minesite Operating Cost (EXW)	**1,215**	**21.58**	**392.59**
Royalties	301	5.34	97.14
Selling Cost	460	8.18	148.80
Total Cash Cost (FOB)	**1,976**	**35.10**	**638.53**
G&A (Corporate)	16	0.29	5.31
Reclamation & Closure Cost	13	0.23	4.13
Sustaining Capex	205	3.65	66.36
All-in Sustaining Cost (FOB)	**2,210**	**39.27**	**714.33**

Specialised trailers and equipment for transporting out-of-gauge items are limited. The design of equipment / plant would have to consider above mentioned limitations to ensure equipment can be transported to site from port.

1.16 Economic Analysis

The economic analysis presented in this chapter contains forward-looking information with regards to the commodity prices, foreign exchange rates, proposed mine production plan, projected mass yield and recovery ratios, CAPEX and OPEX costs. The results of the economic analysis are subject to various known and unknown risks, uncertainties and other factors that cause actual results to differ materially from those presented here.

The economic analysis and investment evaluation was carried out using a discounted cash flow approach. Results based on the evaluation are presented on a post-tax and pre-tax basis.

A Techno-Economic Model ("TEM") was developed using real, post-tax, unlevered, (ie. assumed to be financed 100% by equity), undiscounted free cash flows, discounted at a real discount rate to determine the NPV of the project. The economic analysis and accompanying free cash flow profile generated by the TEM have been presented in United States dollars.

Table 8 lists the basis of evaluation assumptions associated with the Project.

Table 8: Basis of Evaluation Assumptions

Factor	Assumption
Method of Analysis	Discounted Cash Flow Analysis
Cash Flow Terms	Real Terms
Base Currency	United States Dollar (US$)
Base Date of Evaluation	1 September 2023
Discount Rate	8.0% (US$ Real)

Headline results are presented in terms of a scenario that encompasses forward-looking macro-economic assumptions (Forecast Scenario). Results for two other scenarios are also presented that rank *pari-passu* with the headline scenario, that consider commodity prices and foreign exchange rate assumptions at three-year trailing prices and at spot prices respectively.

Table 9 summarises the commodity price assumptions and foreign exchange rate assumptions that are applicable to each of the scenarios used to generate economic results.

Table 9: Macro-Economic Scenarios

Metric	Unit of Measurement	Expert Forecast	3 Year Trailing Average	Spot
Weighted Average Superflake® Graphite Price	US$/t of concentrate	1,191	1,124	1,011
Foreign Exchange Rate: MGA to US$	MGA/US$	4,224	4,312	4,225
Foreign Exchange Rate: ZAR to US$	ZAR/US$	17.67	16.01	16.00

Key economic results are shown in Table 10 below.

Table 10: Key Economic Results

Metric	Unit of Measurement	Forecast	3 Year Trailing Average	Spot
$NPV_{8\%}$ (Post-Tax)	US$ million	370.0	299.8	182.7
IRR (Post-Tax)	%	29.0%	26.0%	19.7%
$NPV_{8\%}$ (Pre-Tax)	US$M	424.1	345.7	216.2
IRR (Pre-Tax)	%	31.1%	27.8%	21.2%
Undiscounted Payback Period	Years (from first Phase 2 concentrate production)	3.1	3.5	4.8
Undiscounted Payback Period	Years (from first Phase 2 Capex)	5.8	6.1	7.5
Peak Funding Requirement	US$ million	178.5	177.9	183.5

The business case is value accretive across all three macro-economic scenarios.

The sensitivity analyses have been performed on a post-tax basis to assess the impact of variations in input parameters to the project's key economic results. Figure 1 depicts the results of the deterministic sensitivity analysis, which assesses the sensitivity of the project's NPV to changing input parameters.

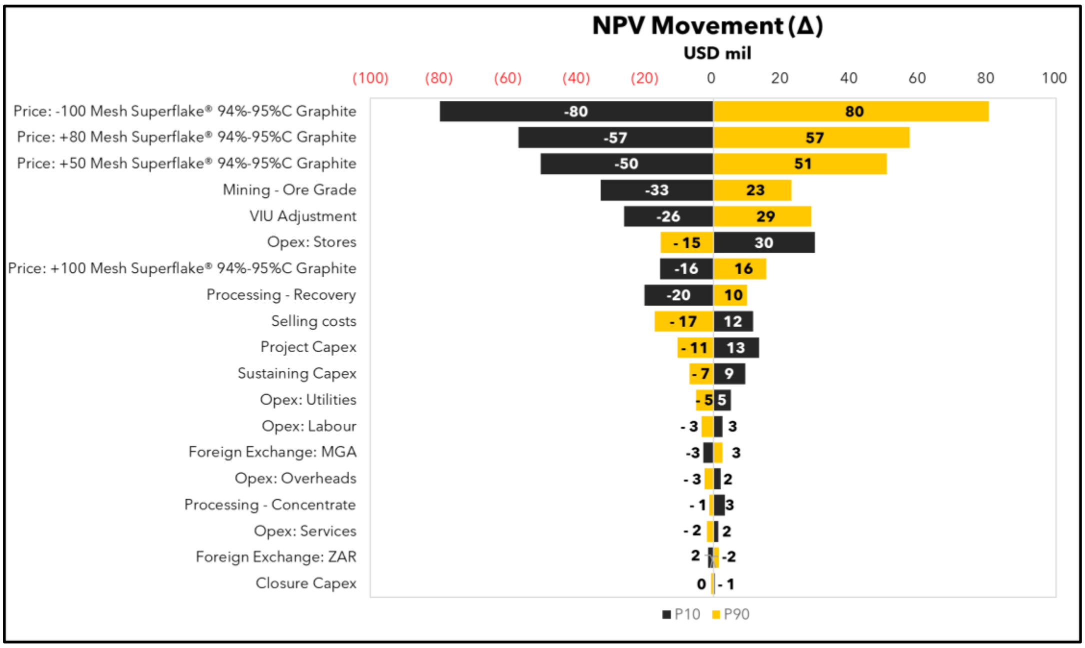


Figure 1: Tornado Chart - NPV Movement

The Project NPV is the most sensitive to the following key drivers:

- Commodity Prices (specifically to -100 Mesh Superflake® graphite concentrate).
- Mining: Ore Grade.
- Operating Cost: Stores.
- Processing: Recovery.

An analysis of robustness has been performed on the forecast macro-economic scenario set of results. The analysis shows that the Molo Expansion Project is value accretive in all scenarios examined, which is indicative of a robust business case.

1.17 Environmental and Permitting

The Global Environmental and Social Impact Assessment (ESIA) for Molo Phase 1 was approved by the Office National Pour l'Environnement (ONE) in Madagascar, April 8th, 2019. The Molo Expansion will according to ONE requires an amendment to the Phase 1 ESIA. The original Global Environmental Authorization remains valid. However, an application for an amendment for the following current documents will be required:

- Specific Environmental Management Plans (SEMPs) for Energy Generation, Camp, Mining, Processing, Tailings, Roads and Pipelines and Dams, and the Conservation Site.
- Relocation Action Plan (RAP).
- Social Development Plans (SDP).

In addition, a completely updated ESIA is required in terms of the International Finance Corporation and Equator Principles, and its findings and recommendations will need to be integrated into the Cahier des Charges Environnement (CCE) which will then be adjusted accordingly.

As part of this ESIA update, a detailed assessment of surrounding aquifers and well field abstraction yields remains to be undertaken and its results integrated into an updated water management plan. As part of the TSF expansion design a dam breach assessment will be required in line with the Global Industry Standards on Tailings Management (GISTM) guidance and the findings incorporated into the Molo risk register and management plans.

Flowing from this, all on-site management plans covering the monitoring and management of direct and indirect environmental impacts will require updating. The timeline for this will be between 12 and 18 months and should, therefore, be initiated in a timely manner.

1.18 Interpretation and Conclusions

This Feasibility Study confirms that the expansion to 150,000 tpa is technically feasible as well as economically viable. There is no certainty that the economic forecasts on which this FS is based will be realized. There are a number of risks and uncertainties identifiable to any project and usually cover the mineralization, process, financial, environment and permitting aspects.

This FS includes an AACE Class 3 estimate with an accuracy level of -15% to +25% at an 80% confidence level, is based on the technical and economic aspects for the expansion project and concluded that, it is viable to expand the mining operation to produce 150,000 tonne of concentrate per year. The specialists engaged in compiling this study are all Qualified Persons according to the NI 43-101 criteria and capable of providing opinions in relation to their expertise.

The Company's 2011 exploration programme delineated a number of new graphitic trends in southern Madagascar. The resource delineation drilling undertaken during 2012-2014 focussed on only one of these, the Molo Deposit, and this has allowed for an Independent, CIM compliant, updated resource statement for the Molo deposit.

The total Measured and Indicated Resource is estimated at 100.37 Mt, grading at 6.27% C. Additionally, an Inferred Resource of 40.91 Mt, grading at 5.78% C is stated.

When compared to the November 2012 resource statement, (Hancox and Subramani, 2013), this shows a 13.7% increase in tonnage, a 3.4% decrease in grade, and a 9.8% increase in graphite content. The reason for the increase in tonnage is due to the 2014 drilling on the previously untested north-eastern limb of the deposit, which added additional new resources. Additionally, 23.62 Mt, grading at 6.32% Carbon, have been upgraded by infill drilling from the Indicated to Measured Resource category.

Mineral reserves of 53.75 Mt have been declared in the Feasibility Study Report with an average head grade of 6.15% C.

The mine design, scheduling and costing for the expansion project was completed by independent specialists. During the development of Molo Phase 1 open pit mining was identified as the most appropriate and cost effective method of mining. The open pit mining method is currently employed at Molo and was proven adequate and extended for the expansion project due to the surficial, lateral expanse and the massive nature of the Molo deposit.

Comprehensive metallurgical test programs culminated in a process flowsheet that is capable of treating the Molo ore using conventional and established mineral processing techniques. Process risks associated with the variability of the ore have been mitigated in the process design to reduce graphite flake degradation, improve recovery and allows for processing flexibility to accommodate mine geology.

The concentrator is designed to increase the total concentrate output of the Molo operations to its full capacity of 150 ktpa. Lessons learned from the Phase 1 concentrator, that is capable of producing 17 ktpa, has been incorporated into the conceptual design for the project expansion. The process plant design described in this report includes two identical streams for milling operations in order to reduce equipment sizing for importation. Therefore, to accommodate importation limitations modular concentrators may be reviewed during detail design to replace the single concentrator identified for this FS.

Infrastructure has been established at Molo to accommodate for Phase 1 operations and this proved to be sufficient for current operations. The Phase 1 scope included access roads, site roads, aircraft landing strip that are all adequate for the expansion with minor improvement on roads. Areas where infrastructure expansion is required are, power supply, water supply and storage, accommodation and associates services, offices, workshops and stores for operational requirement.

Tailings will be deposited on a newly established tailings disposal site that was identified for the expansion. Tailings will be dried and co-disposed with the waste rock generated from operations mining. Tailings will be conveyed from the processing facility to the TSF. It is envisaged that all waste rock will be crushed and conveyed with the fines waste to the TSF. In the next phase of the study a detailed design will be completed, complete with environmental and social impact assessment and closure.

All mining permits are in place for Phase 1 operations and included the Global Environmental and Social Impact Assessment (ESIA). The Molo Expansion will according to ONE requires an amendment to the Phase 1 ESIA. Additional specific Environmental Management Plans (SEMPs) for Energy Generation, Camp, Mining, Processing, Tailings, Roads and Pipelines and Dams, and the Conservation Site will be required.

For the expansion project a completely updated ESIA is required in terms of the International Finance Corporation and Equator Principles, and its findings and recommendations will need to be integrated into the Cahier des Charges Environmental (CCE) which will then be adjusted accordingly.

1.19 Recommendations

Whilst the results of this FS demonstrate that the expansion to 150 ktpa is financially sound there are implementation options which may further optimise the economic results. The following recommendations have been identified throughout the FS:

- Infill drilling to convert the Inferred Resource to Indicated.
- Process design optimisation to incorporate current knowledge base and industry best practice.
- Progress the TSF design and testwork to optimise the LoM design.

The expansion will require application for amendments as and when applicable to the stand-alone documentation from the original Phase 1 ESIA, SEMPs, RAP, SDP, Tree Clearance Permit, and Water Uses.

2 INTRODUCTION

NextSource Materials Inc. is a strategic materials development and mining company based in Toronto, Canada.

Prior to April 24, 2017 the Company was called Energizer Resources Inc. ("Energizer") which was changed from Uranium Star Corporation, ("Uranium Star"), on December 9, 2009. The Company commenced trading on the Toronto Stock Exchange ("TSX"), under the symbol NEXT on April 24, 2017. The Company is currently focused on becoming an integrated, global supplier of critical battery and technology materials, and key to this strategy is the development of its Molo graphite deposit, ("Molo" or "the Project"), in south-western Madagascar.

During 2023 NextSource has been ramping up its Phase 1 project to nameplate capacity of 17,000 tpa of flake graphite. This FS considers an expansion to the Molo Graphite Mines' current production capacity with the construction of an additional processing plant and mining infrastructure that increases the steady-state production rate to 150,000 tpa of SuperFlake® graphite concentrate over a 25 year LoM.

This National Instrument 43-101 (NI 43-101) Technical Report was prepared by Erudite Strategies (Pty) Ltd and issued by Nextsource Materials Inc.

2.1 The Technical Team

This Technical Report has been prepared by a combined technical team made up of people, who by virtue of their education, experience and professional association, are considered qualified people as defined by NI 43-101.

The team is comprised by individuals from the following organizations:

- Caracle Creek International Consulting (Proprietary) Limited – Geology.
- Geostratum (Pty) Ltd – Water and Geohydrology.
- Globesight (Pty) Ltd - Environmental, Social, Permitting.
- SGS Lakefield – Metallurgical Test work.
- Metpro – Metallurgical Test work analysis.
- ECO - Elementum – Tailings Storage Facility.
- Deswik (Pty) Ltd – Mining Design and Mining Capital and Operational costs.
- Erudite Projects – Earthworks, Civils, and Infrastructure.
- Erudite Projects (Pty) Ltd – Process Engineering. The process engineering was completed for the Molo 2017 FS by Met63 with the competent person being Mr. Paul Harvey supported by Mr A Mokwena. Neither of these experts is currently employed by Met63 and as such was not able to provide further comment on the process engineering as part of the FS. Mr Hector Mapheto, Pr.Eng who holds specific graphite processing knowledge and is currently employed by Erudite Projects (Pty) Ltd, has reviewed the process design and his comments are incorporated as appropriate.
- ISS, ASSTRA, Velogic – Logistics.
- OHMS – Mine Geotech.
- RLH – Port Trade-off Study (From Molo 2015 FS).

2.2 Report History

The report history for the Project is as shown in Table 11 below:

Table 11: Report History for the Project

Date	Description of Event
February 2022	Finalisation of FS report based on NI 43-101 requirements.
March 2021	Initiation of construction of the Expansion of the Molo Graphite Mine.
February 2021	Binding agreement with Vision Blue to provide financing for construction of the Expansion of the Molo Graphite Mine.
May 2019	Technical Report NI 43-101 FS report consisting of the Expansion with processing capacity of 240 ktpa and Expansion of 720 ktpa.
April 2019	Environmental License for the Molo Graphite Project granted.
February 2019	Official permit is granted to the Company by the BCMM.
	Environmental and Social Impact Assessment completed and submitted to the Malagasy government.
January 2019	Permit PR #3432 was transformed into two Exploration Permits (PR #39806 and PR #39810) and an Exploitation Permit (PE #39807) by the Ministry of Mines.
July 2017	Molo 2017 FS (Detailed Engineering and Design) undertaken by Met 63.
May 2017	Molo 2015 FS Technical Report NI 43-101 Technical Report issued by DRA.
April 2017	Energizer Resources rebrands to NextSource Materials Incorporated.
February 2015	Molo 2015 DFS undertaken by DRA.
September 2014	Technical Report NI 43-101 issued by Hancox and Subramani.
August 2014	Updated resource statement published.
2014	Further exploratory drilling and sampling undertaken by CCIC – additional 32 diamond drill holes and 9 trenches.
April 2014	Energizer signs a Sale and Purchase Agreement and a Mineral Rights Agreement with Malagasy to acquire the remaining 25% interest.
April 2013	FS Technical Report NI 43-101 issued.
2013	FS undertaken by DRA.

Date	Description of Event
October 2013	Energizer signs an MOU" with Malagasy to acquire the remaining 25% interest in the land position.
December 2012	Maiden resource statement published.
May – November 2012	Resource delineation, drilling and trenching on Molo takes place.
Early 2012	Seven holes drill tested.
January 2012	Energizer announces Molo discovery.
December 2011	Energizer enters into a JVA with Malagasy Minerals Limited to acquire a 75% interest to explore and develop a group of industrial minerals.
Late 2011	Molo discovered by Energizer.
December 2009	Uranium Star Corporation rebrands to Energizer Resources.
2007	Exploration works of the Molo graphite deposit completed by Uranium Star.

2.3 Terms of Reference and Purpose

The terms of references for this phase was to re-use the results of the previous NI 43-101 compliant studies, augment them with current actual data stemming from the construction of the Molo graphite mine, (designed to produce 17 ktpa of flake graphite), and consider the addition of a stand-alone expansion with a plant capacity to process 2,640,000 tpa of ore and increase the steady-state graphite concentrate production to 150 ktpa. This study was prepared in accordance with Form 43-101F1for the proposed Molo Graphite Expansion 150 ktpa concentrate Project. The work covers the following key activities:

- To engineer and design a concentrator for the expansion project with all supporting infrastructure.
- To compile CAPEX and OPEX estimates for the Project.
- To build a financial model and perform an economic analysis of the Project.

2.4 Sources of Information

The following sources of information have been used in this Report. Refer to Table 12 below.

Table 12: Source of Information

Report or Document	Author / Organization
Physical Data	
Site Topographical Data	The Company
Exploration Permit 3432	Malagasy Government
Civil Geotechnical Reports	EPOCH Consulting and more recently CERMAT
Climate Report	GCS Water and Environment
Geochemical Analysis	GCS Water and Environment
Geological Block Model	CCIC
Mine Geotech Report	OHMS
Mining and Reserves	
Mine Schedule	Mr. M Flannigan
Equipment Tenders	Various
Process Plant and Metallurgy	
Process Design Criteria (PDC)	Erudite Strategies
Process Flow Diagrams (PFD)	Erudite Strategies
Mechanical Equipment List (MEL)	Erudite Strategies
Metallurgical Test Work Reports	SGS
Metallurgical Test Work Analysis	Metpro
Vendor Test Work Reports	Modular Supplier
Plant Layout Drawings	SYNCS
Block Plan	Erudite Strategies
Rheology Test Work	SGS
Production Schedule	Mr. M Flannigan
Tailings Disposal and Storage	
Site Selection Report	ECO - Elementum
TSF Report	ECO - Elementum
Infrastructure	
Site Plan	Erudite Strategies
Area Plan	Erudite Strategies

Report or Document	Author / Organization
Infrastructure Equipment List (IEL)	Erudite Strategies
Surface Water Supply Report	Geostratum
Power	
Power Plant Layout Drawings	Cross Boundary
Running Load Estimate	Erudite Strategies
Single Line Diagram	Erudite Strategies
Vendor Quotes	Various
Fuel quotes	Various
Water	
Detailed Water Report	Geostratum
Water Balance	Geostratum and Erudite
Environmental and Social	
Specialist Studies	GCS Water and Environment and Agetipa
ESIA (Environmental & Social Impact Assessment)	GCS Water and Environment and Agetipa
MOU (Memorandum of Understanding)	Agetipa
TOR (Terms of Reference)	Agetipa
Certification of Investment Amount	GCS Water and Environment and Agetipa
Sensitivity Maps	GCS Water and Environment
Permitting and Stakeholders	
Permit Register	Globesight
Stakeholder Register	Globesight
HR and Operational Readiness	
Malagasy Legislation	The Company
Malagasy Labour Rates	The Company
Staffing Plan	Erudite Strategies
Transport and Logistics	
Route Surveys	Asstra, Velogic

2.5 Personal Inspections

The following personal inspections were done on the property:

- Dr. Philip John Hancox, (QP for sections 6 to 11), between the 8 and 11 May, 2012. During this visit the graphite occurrences at Molo, Seta and Fotsy were investigated and various trenches on Molo were inspected. A second site visit was undertaken between the 19 and 21 May 2013, during the bulk sampling exercise. During this visit a number of collar beacons from the 2012 drilling campaign were inspected, as well as the trenching, logging and sampling methodologies.
- Desmond Subramani visited the Molo site between the 15 and 19 February 2014, during the 2014 drilling campaign. The main aim of this visit was to plan the layout of the additional drilling and trenching required for the resource upgrade. This visit also covered the inspection of various borehole collars and open trenches, as well as a review of the drilling, logging and sampling procedures.
- Dave Thompson visited the Molo site on 11 to 13 March 2014. The aim of this visit was to assess the site from a mining perspective.
- Oliver Peters and John Stanbury have not visited the site.
- Although not an author of this report, a principal engineering geologist, Mr. Colin Wessels, visited the site from 5 to 13 April and 17 to 21 June 2014 to conduct a reconnaissance of the potential tailings dam sites and to supervise geotechnical investigations for the tailings dam, borrow pits and plant site.

The following QP's on the Project, spent time on site and contributed to the study in their various areas of expertise:

Nico Hamman	Various dates in 2023.
Alkie Marais	March 11 to 13, 2014.
Eugene de Villiers	During 2023.
Hercu Smit	No existing infrastructure to influence the design. (Did not visit).
Schalk Pienaar	No existing infrastructure to influence the design. (Did not visit).
John Hancox	Visited site on various dates in May 2012 and 2013.
Desmond Subramani	Visited site from February 15 to 19, 2014.
Hector Mapheto	Visited site during 2023.
Oliver Peters	No existing process to inspect. (Did not visit).
Ruan Daffue	Did not visit site, reliance was therefore made on previous site visits undertaken by Qualified Persons.
Mr Johann de Bruin	(QP) visited the site from the 9 to the 11 of May 2023 during the commissioning of the Expansion Plant, as well as in 2015, 2016, 2017 and 2022.

2.6 Currency and Units

This Report, unless otherwise stated, uses Système international d'unités ("SI") metric units, the standard Canadian and international practices, including metric tonnes ("tonnes", "t") for weight, and kiLoMetres ("km"), or metres ("m") for distance.

All currency amounts, unless otherwise stated, are expressed in United States Dollars ("USD" or "US$").

3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared for the Company and the information, conclusions, opinions and estimates contained herein are based on:

- Information available at the time of preparation of this report, (effective date being the September 1, 2023).
- Assumptions, conditions and qualifications as set forth in this report.
- Data, reports and other information as supplied by the Company and other third-party sources.
- Experts involved in the run-up to the Molo Phase 1 execution and the execution recently completed.

For the purpose of this report, the authors have relied on ownership information provided by the Company. In consideration of all legal aspects, relating to the Project, Erudite Strategies and CCIC places reliance on the Company that the information relating to the legal aspects, and the status of surface and mineral rights, are accurate.

At the time of writing this report all mining permits and other statutory and regulatory permits and documents are in place as they apply to Phase 1 and are also in respect to all environmental requirements by the ONE, including the necessary permits to extract water by ANDEA.

Property information in this report is sourced from previous works supplied by the Company and the authors are not responsible for the accuracy of any property data and do not make any claim, or state any opinion, as to the validity of the property disposition described herein. The ongoing operation of Phase 1 execution bears witness to the validity of all statutory, regulatory, and environmental permitting and their validity at the time of writing.

For the preparation of this report, the authors relied on maps, documents and electronic files generated by the current and past exploration crews, contributing consultants and the technical team of the Company.

The Company has received specialist input from the following organizations in the preparation of this report:

- Eco-Elementum (Co-disposal Tailings Storage Facility).
- Geostratum – (Water Management Consulting and Geohydrology).
- Globesight (Environmental, Social, Permitting).
- ASSTRA, Velogic (Logistics).
- OHMS (Mine Geotechnical).
- Practara (Financial Modeling and Taxation).
- SGS Lakefield (Process Analytics).
- Caracal Creek International (Resource Definition).
- Erudite Projects (Pty) Ltd (Infrastructure and Process).
- Deswik (Mining, Pit selection and Optimization).
- Lebedev Consult (Pit-to-port Dynamic Simulation).
- John W Ffooks and Co. (Eligibility of the project to the Large-Scale Mining Investment law).

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Madagascar Overview

The Republic of Madagascar, ("Madagascar"), has the potential to play an important role in mining and contributes to the sustainable development goals with more exploration in planning phases. Madagascar, not a traditional, or particularly well known exploration and mining destination and is mostly under explored, with the country subject to very little modern era systematic exploration.

4.1.1 Country Overview

Madagascar is the largest island in the Indian Ocean (Figure 2) and is the 4th largest island in the world, (after Greenland, New Guinea and Borneo). It is located to the south-east of the African continent, which is separated by the Mozambique Channel. The country extends over 1,570 km from north to south, and is over 575 km wide, with a surface area of 587,040 km^2 and a 5,000 km long coastline.

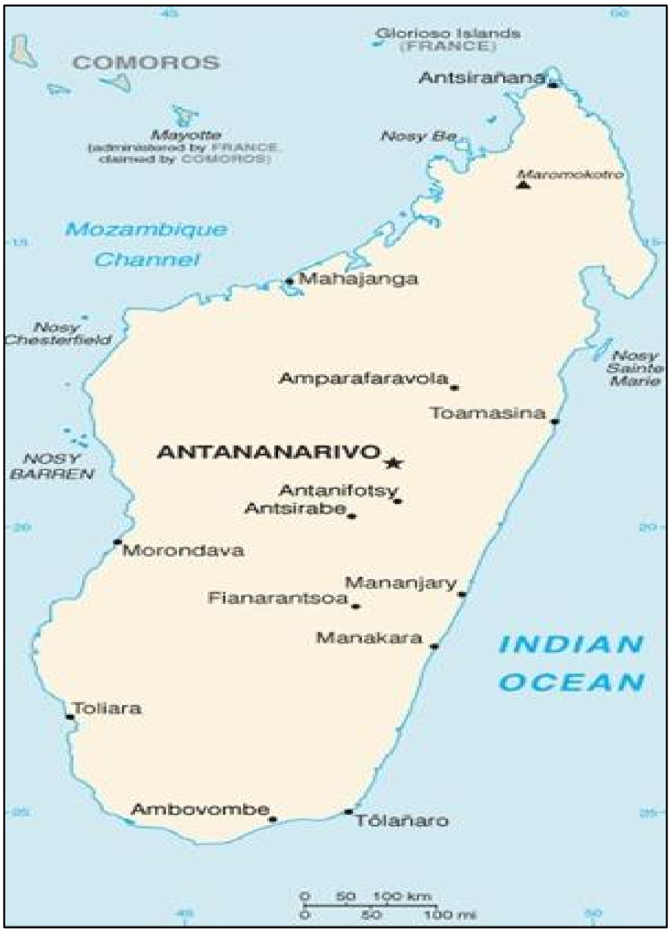


Figure 2: Map of Madagascar

Source: http://goafrica.about.com/library/bl.mapfacts.madagascar.htm

The capital of Madagascar is Antananarivo, a city with approximately 3,900,000 people and is located in the central eastern area of the island approximately 150 km inland from the central-east coast, at an elevation of just over 1,200m above mean sea level ("mamsl").

Madagascar is officially bilingual, with French being the language of the government and business. Malagasy (Malgache), a language of Malayo-Polynesian origin, is the official national language. English is taught in schools, however, it is not widely spoken outside of business and government circles.

Madagascar is one of the world's hotspots of endemism and is recognized as one of the planets 17 recognized mega-biodiversity countries, (Razafindralambo and Gaylord, 2005). Over 80% of the country's plant and animal species are unique to the island. Vegetation is varied and ranges from dense tropical rain forest in the east, Savannah in the central plateau and western coastal plain and spiny dry vegetation in the southern areas, which is where the Project is located.

4.1.2 Government Policy and Outlook regarding the Mining Industry

The government of Madagascar embarked on an economic revival plan in 2000. At that time the Ministry of Energy and Mines ("Ministry"), had already initiated reform through the Projet de Réforme du Secteur Minier ("PRSM") program, with the introduction of the new Mining Code in 1999, and the establishment of the Mining Titles (Cadastral), Registry (Bureau du Cadastre Minier de Madagascar, or BCMM) in 2000. These initiatives attracted new investors to Madagascar, including both junior and senior mining companies.

During 2003, in furtherance of its economic policy, the Ministry commenced the 5 year Projet de Gouvernances des Ressources Minérales program, with the following objectives:

- To further improve and enforce the legal and statutory framework, particularly with respect to mining.
- To promote investment in the minerals sector through the dedicated Agence de Promotion du Secteur Minier.
- To improve the geoscientific knowledge of Madagascar through geophysical surveys, geological mapping, and remote sensing, with appropriate staff training to support mapping Projects.
- To address environmental health and safety issues and to contribute to poverty reduction.

4.2 Project Location

The Project is located approximately 160 km south-east of Madagascar's port city of Toliara, in the Tulear region and roughly 220 km north-west of Fort Dauphin. Refer (Figure 3) below.

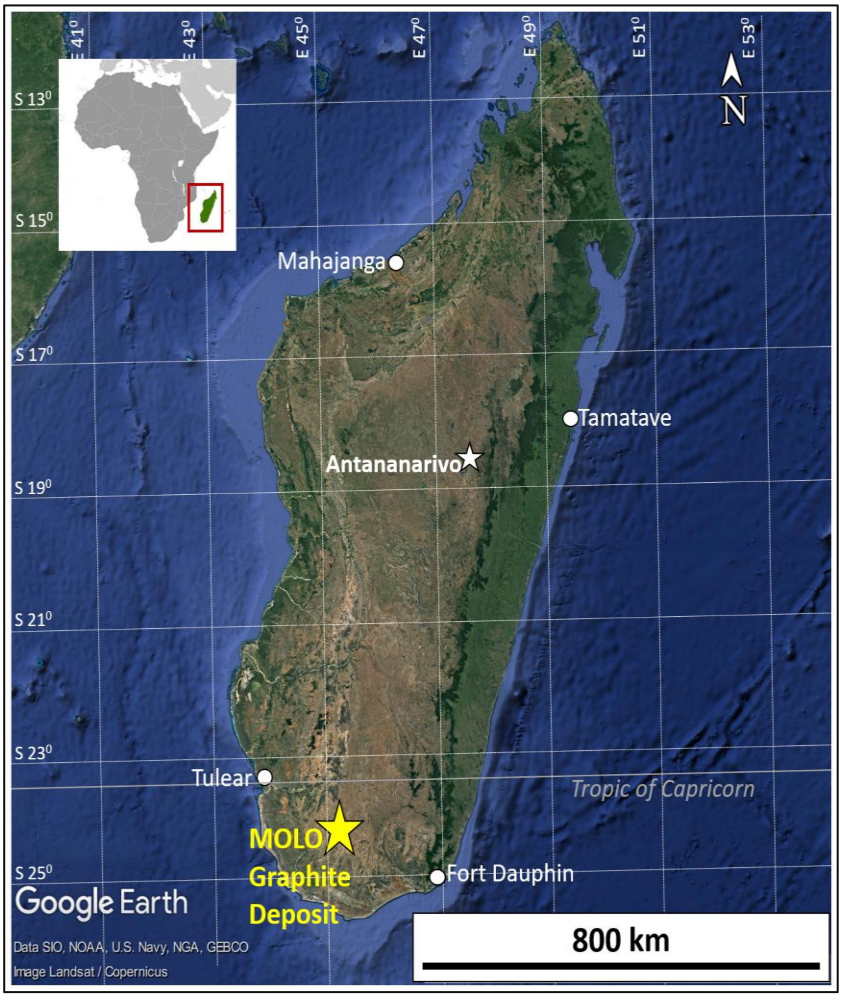


Figure 3: Molo Project Location

4.3 Property Area

The Project includes 790 claims and an area totalling 308.6 km^2. Refer (Figure 4) below. The Project is centred on the UTM co-ordinates 495,289 easting and 7,345,473 northing (UTM 38S, WGS 84 datum). The Project is located 11.5 km east-north-east of the town of Fotadrevo within Exploitation / Mining Permit PE #39807 which covers an area of 175 km^2, or 17,500 hectares ("ha") and Exploration Permits PR #39806 and PR #39810 which covers areas of 96.1 km^2 (9,609 ha) and 37.5 km^2 (3,750 ha) respectively. Further to the preceding 2 permits, a Land Lease Permit which provides legal access was also granted for a period of 50 years, which commenced on February 25, 2022. The Government of Madagascar designates individual claims by a central LaBorde UTM location point, comprising a square measuring 625m x 625m. (Refer to Figure 4).

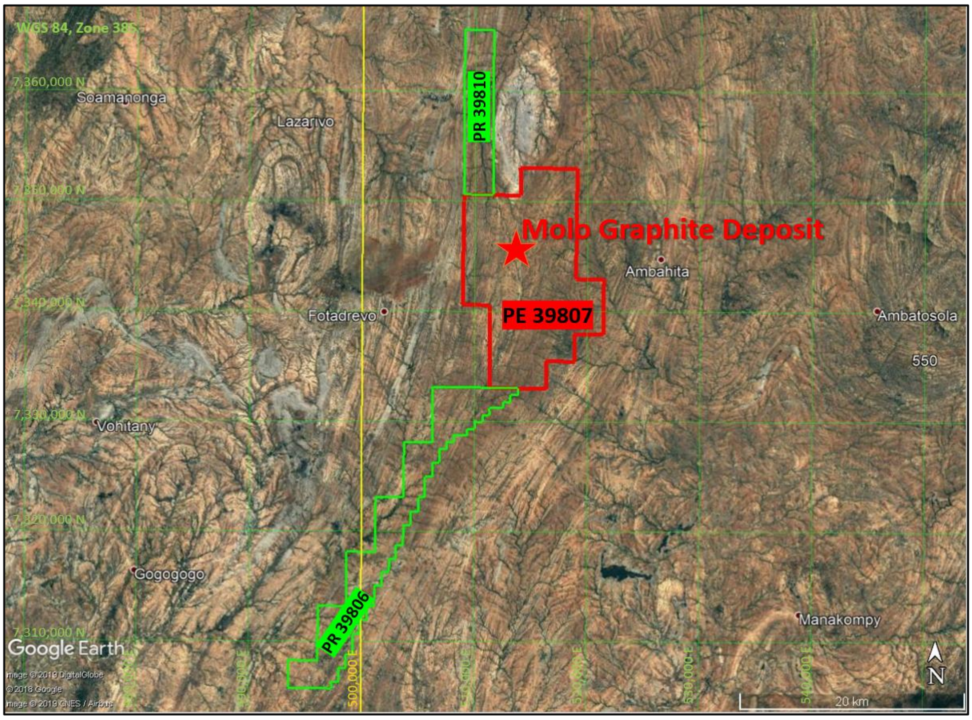


Figure 4: Molo Project - Exploitation Permit PE #39807

4.4 Mineral Tenure and Title

On December 14, 2011, the Company entered into a Definitive Joint Venture Agreement ("JVA") with Malagasy Minerals Limited ("Malagasy"), a public company on the Australian Stock Exchange, to acquire a 75% interest to explore and develop a group of industrial minerals, including graphite, vanadium and approximately 25 other minerals. On October 24, 2013, the Company signed a Memorandum of Understanding ("MOU") with Malagasy to acquire the remaining 25% interest in the land position.

On April 16, 2014, the Company signed a Sale and Purchase Agreement and a Mineral Rights Agreement with Malagasy to acquire the remaining 25% interest. Malagasy retains a 1.5% net smelter return royalty ("NSR").

CCIC reviewed a copy of the Contrat d'Amodiation pertaining to this right and are satisfied that the rights to explore this permit have been ceded to the Company, or one of its Madagascar subsidiaries.

The Project was located within Exploration Permit PR #3432 as issued by the BCMM pursuant to the Mining Code 1999 (as amended), and its implementing decrees. On January 18, 2019, Permit PR #3432 was transformed into two Exploration Permits (PR #39806 and PR #39810) and an Exploitation Permit (PE #39807) by the Ministry of Mines, with the official permit being granted to the Company by the BCMM on February 14, 2019.

Mineral Resources and Reserves delineated in Sections 14 and 15 of this report are entirely within the bounds of Exploitation Permit PE #39807.

The Company holds the exclusive right to exploit / mine and explore for graphite within this license area for a period of 40 years and can renew the license several times for a period of 20 years upon each renewal.

The Company holds the exclusive right to explore for a defined group of industrial minerals within Exploration Permits PR #39806 and PR #39810. These industrial minerals includes the following:

- Vanadium.
- Lithium.
- Aggregates.
- Alunite.
- Barite.
- Bentonite.
- Vermiculite.
- Carbonatites.
- Corundum.
- Dimensional stone, (excluding labradorite).
- Feldspar (excluding labradorite).
- Fluorspar.
- Granite.
- Graphite.
- Gypsum.
- Kaolin.
- Kyanite.
- Limestone / Dolomite.
- Marble.
- Mica.
- Olivine.
- Perlite.
- Phosphate.
- Potash / Potassium minerals.
- Pumice Quartz.
- Staurolite.
- Zeolites.

ERG (Madagascar) Ltd S.A.R.L.U first applied for an exploration mining permit with the BCMM, a government agency falling under the authority of the Minister of Mines. Permits under usual circumstances are generally issued within a month. The number of squares varies widely by claim number.

The updated Decret requires the payment of annual administration fees of Permits Research of 15,000 Ariary (MGA), for exploitation permits in years one and two. Annual fees increased by multiplying by a factor equivalent to the number of years (plus 1), that the company has held the permit. Exploration permits have an updated duration of 5 years, with the possibility of two renewals of an additional 3 years each. Payments of the administration fees are due each year on 31 March, along with the submission of an activity report.

Each year the Company is required to pay a similar, although increasing, amount in order to maintain the claims in good standing.

Reporting requirements of exploration activities carried out by the title holder on an Exploration Permit are minimal. A title holder must maintain a diary of events and record the names and dates of the persons currently active on the Project. In addition, a site plan with a scale between 1/100 and 1/10,000 showing “a map of the work completed” must be presented. CCIC is of the opinion that the Company is compliant in terms of its commitments under these reporting requirements.

The Project has not been legally surveyed, however, since all claim boundaries conform to the pre-determined rectilinear LaBorde Projection grid, these can be readily located on the ground by use of Global Positioning System (“GPS”) instruments. Most current GPS units and software packages do not, however, offer LaBorde among their available options, and therefore, defined shifts must be employed to display LaBorde data in the WGS 84 system. For convenience, all the Company’s positional data is collected in WGS 84 and if necessary, converted back to LaBorde.

4.5 Royalties

Prior to the expansion 5% of revenue and 3.5% of the FOB value of sales is payable to the Government of Madagascar. After the expansion, and successful application of the LGIM, the royalty decreases to 2.5% of revenue and 3.5% of FOB value of sales.

Vision Blue retains a 3% gross revenue royalty, and Malagasy retains a 1.5% net smelter return royalty on the Project.

4.6 Permits

Exploitation Permit PE #39807 (175 km^2) and Exploration Permits PR #39806 and PR #39810 are held under the name of a subsidiary of the Company called ERG (Madagascar) Ltd. S.A.R.L.U. and were granted to the Company by the BCMM on February 14, 2019.

The Madagascar Ministry of Environment’s Office National pour l’Environnement (the National Office for the Environment), or “ONE”, granted the Company its Environmental License for Phase 1 of the Molo Graphite Mine on April 8, 2019.

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

5.1 Access

Access to Molo from Toliara, is initially via a 70 km paved road to the village of Andranovory (Figure 5) below. From Andranovory, a secondary all-season roads continue to Betioky, a distance of 93 km. From Betioky the Property area can be reached via Ambatry to Fotadrevo, a distance of 105 km (268 km total), or from Betioky to Ejeda, then onwards to Fotadrevo, a distance of 161 km (324 km total). This alternate route from Ejeda to Fotadrevo is used by heavy transports and by all vehicles during portions of the rainy season, as the primary route quickly becomes impassable. At the height of the rainy season, both routes to Fotadrevo may be largely impassable. Molo may be reached from Fotadrevo by a fairly well-maintained dirt road.

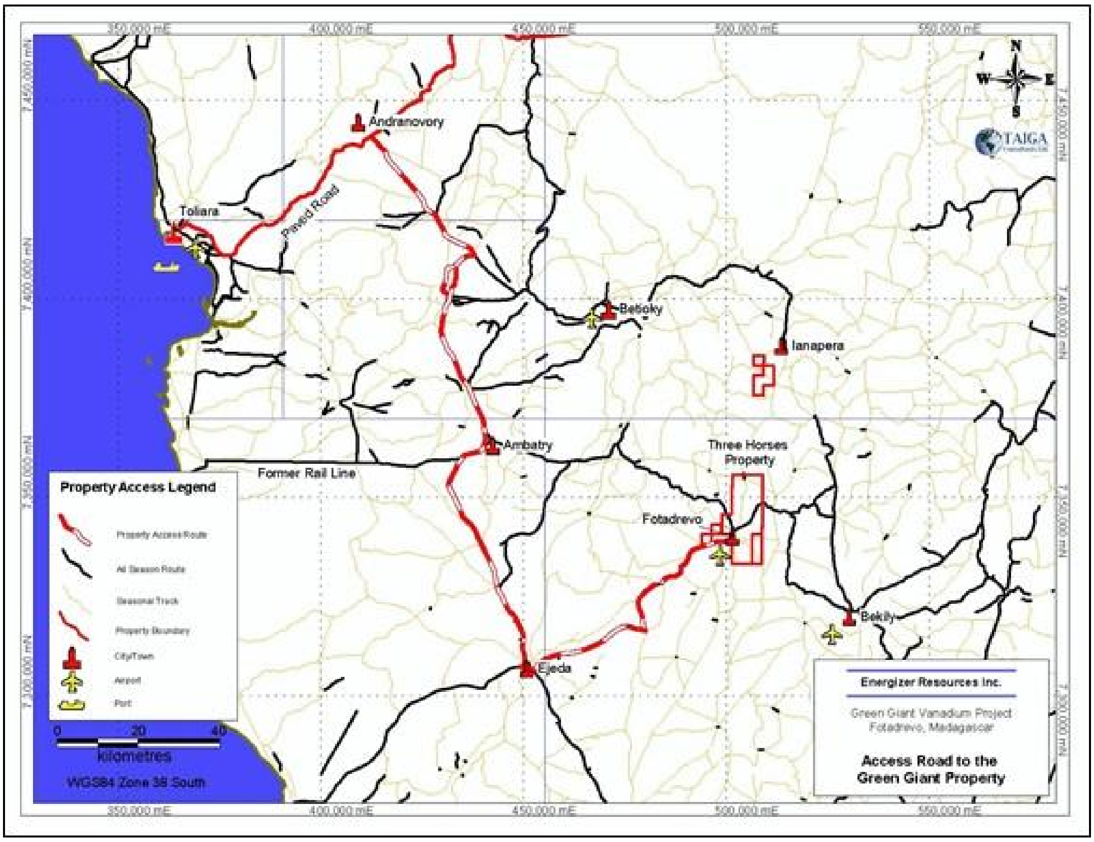


Figure 5: Road Access to the Molo Area from the Town of Toliara

Access to the Molo area from the Port of Ehoala is via the RN 13 over a total distance of 372 km and passes through the municipality of Andalatanosy, Ambatofotsy then through Beraketa and later on through the city of Bekily. (Refer Figure 6 below). The bulk of the road is severely rutted bush track, which during the rainy season from November to March can at times become impassable.

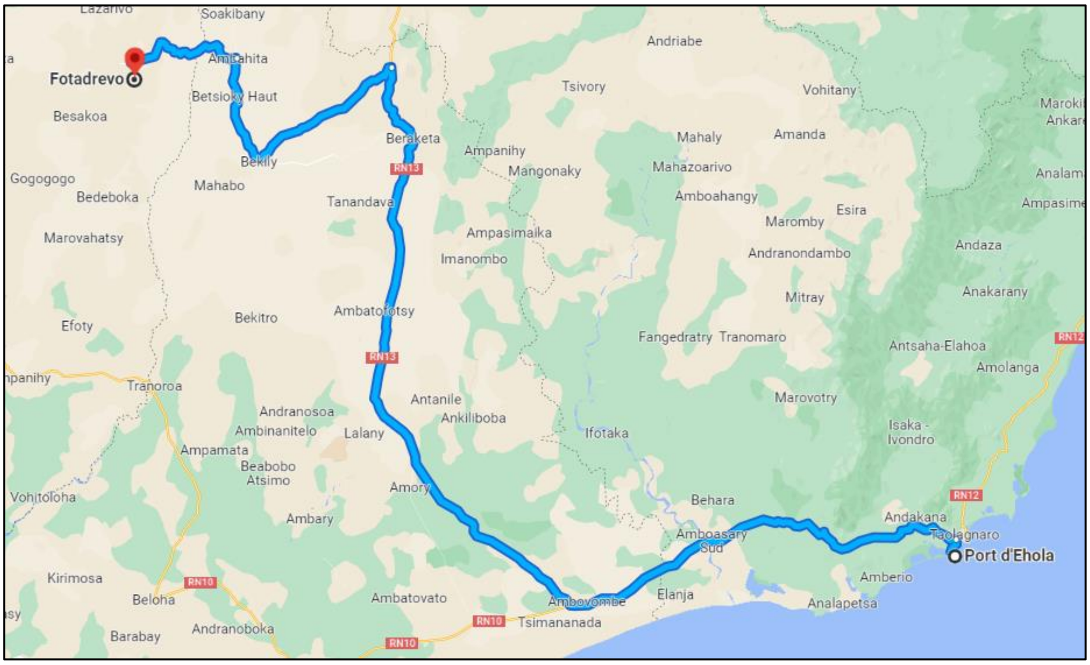


Figure 6: Road access to site from Port Ehoala

With the upgrading of an existing airstrip at Fotadrevo (Photo 1) to an all-weather airstrip during the 2008 exploration programme, the Project area is now accessible year round, (except under special circumstance caused by continuous, or multiple days of heavy rain), by air, using private aircraft out of Antananarivo. Flying times to Fotadrevo are roughly 2.5 hours from Antananarivo and 45 minutes from Toliara.

Photo 1: Airstrip At Fotadrevo

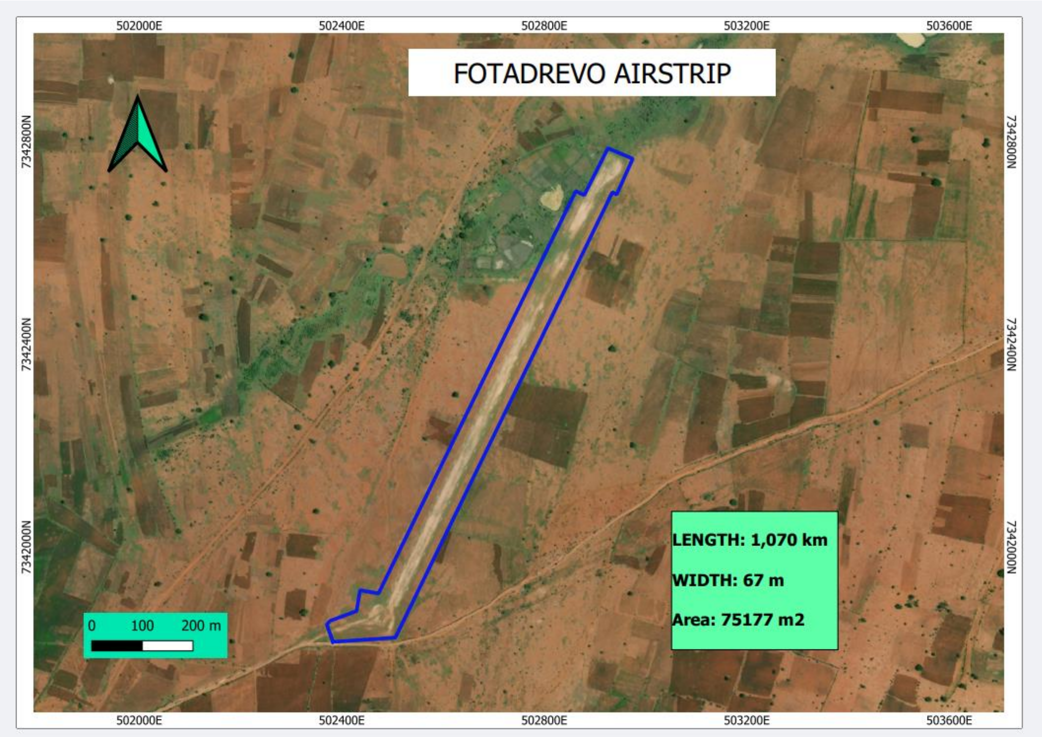


Figure 7: Fotadrevo airstrip with important dimensions

Antananarivo is currently serviced by Air France (from Paris), Air Mauritius, (Sir Seewoosagur Ramgoolam International Airport in south-east Mauritius), Air Ethiopia flying from Addis Ababa, (Addis Ababa Bole International Airport), Kenya Airways flying from Nairobi, Kenya (Jomo Kenyatta International Airport) and recently Airlink flying from Johannesburg (O.R. Tambo International airport). Air Madagascar also has regularly scheduled domestic jet and propjet flights throughout the country, including daily flights between Antananarivo and Toliara.

5.2 Physiography

The Molo deposit area is covered by sparse vegetation (Photo 2) below. Grass cover is widespread, and trees are widely spaced overall, with accumulations near drainage lines and stream beds. In areas of lower relief, alluvial cover is generally shallow, and bedrock and/or floats are readily observable. Elevations range between 536m to 565m above mean sea level (“mamsl”).

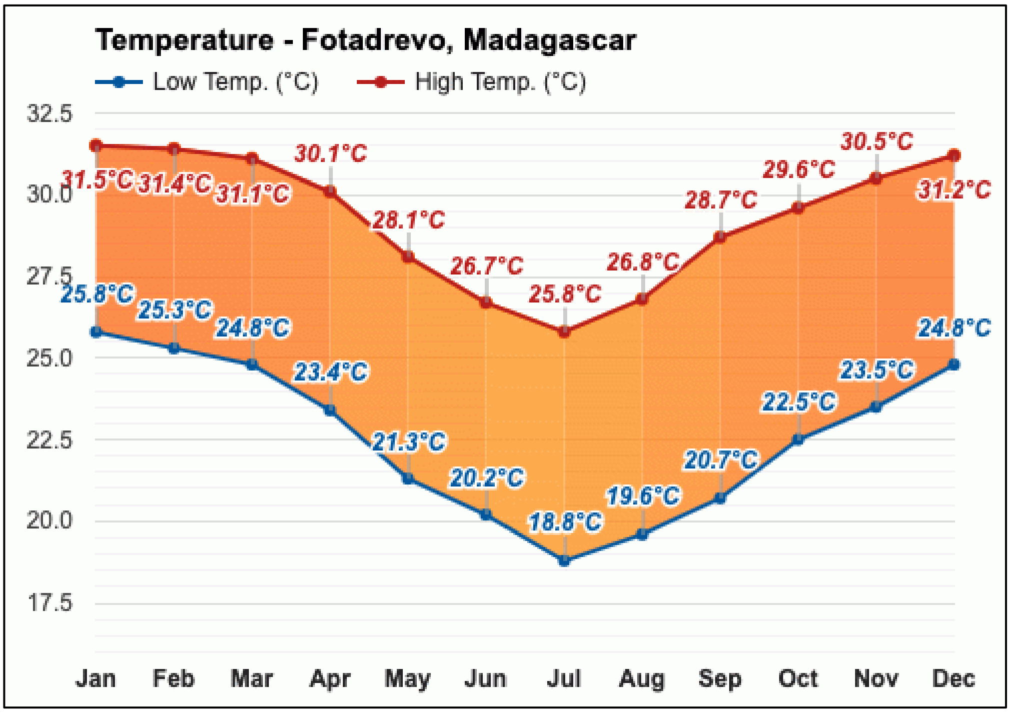
Photo 2: View of the Molo Project Area showing the Nature of the Vegetation

Typical of the tropics, the surface is subject to lateritic weathering. However, full laterite profiles are rarely developed within the southern climatic zone. Previous drilling on the Property indicated that the weathered profile is typically less than 10m thick in the region, which is roughly one third of that seen in other parts of Madagascar and on the adjacent African continent.

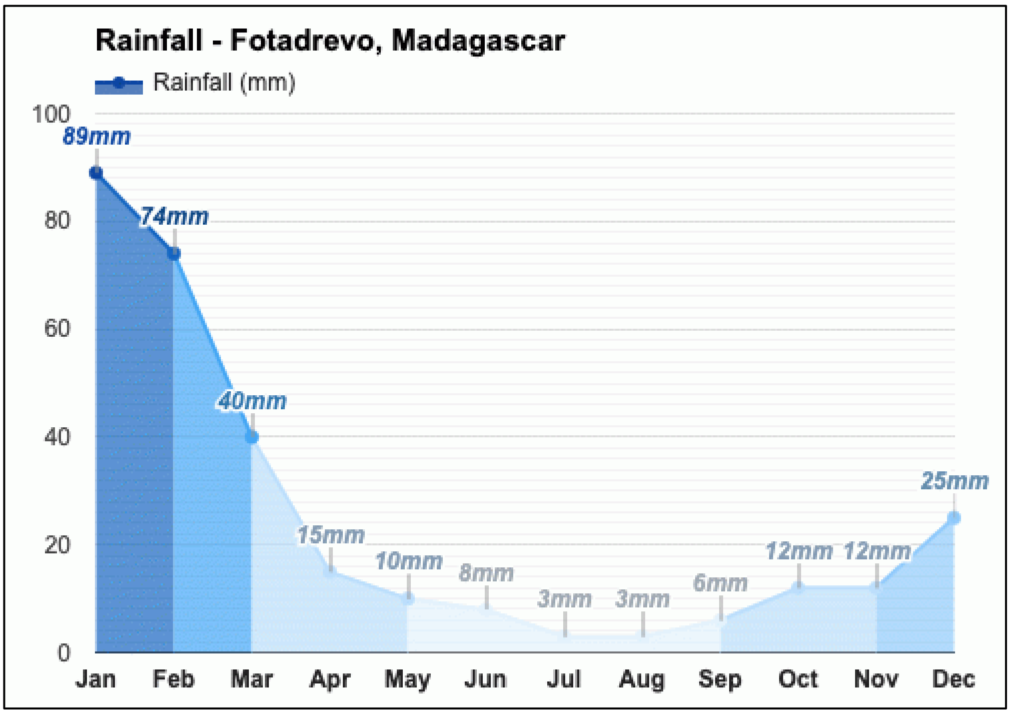
5.3 Climate

Five climatic zones divide Madagascar. The Molo deposit area falls within the semi-desert south zone, with elevated temperatures all year-round peaking in the hot season at an average of over 30°C. The climate is dominated by south-eastern trade winds originating in the Indian Ocean anticyclone, a centre of high atmospheric pressure that seasonally changes its position over the ocean. Madagascar has two seasons, a hot, rainy season from December to March / April, and a cooler dry season from April / May to November. During the rainy season access to site by road is more difficult than during the dry season due to the overall general condition of the infrastructure. Total rainfall is sparse within the Molo area, with yearly precipitation ranging from 300 mm to 500 mm. The rainy season, which is mainly concentrated in the November to March period, causes difficulty in travelling off the main highways for exploration, logistics and operations.

Refer to Graph 3 and Graph 4 below.



Graph 3: Average Temperature Graph



Graph 4: Average Rainfall Graph

5.4 Local Resources and Infrastructure

The village of Fotadrevo is located to the west of the Molo deposit area. The village has been a source of labour during the exploration programmes on Molo and a portion of the workforce for Molo Phase 1 and subsequently also planned for the expansion. Unskilled labour is available locally, while to some extent, semi-skilled resources are available regionally. Skilled labour is available either on a national basis or as expats. A few basic goods are commercially available in the village. However, the main centres for support are the cities of Toliara and Antananarivo.

Molo Mine now operates its own 70 person mine camp immediately to the east of the Molo plant. Power is supplied to the camp by 2 dedicated diesel generators of 320 kVA, one standby and one running. In October 2023 the generators will be removed and an 11kVA overhead line will supply the camp from the main generator farm at the plant, and/or the newly completed solar farm.

A cellular telephone tower located in Fotadrevo, which provides convenient coverage, and during Phase 1 execution an Airtel tower was added on site. Currently there are 3 existing boreholes on the Molo property. The expected requirement for Expansion is that as many as a further 22 holes might be drilled to provide for water requirements.

5.5 Security

Madagascar is an island where no border issues, or conflicts are known that might affect operations, security, or title in the region. Security of personnel is a company policy directed by management. Considering that the area is predominantly rural, with few police, or other security patrols common in the area, there is always a small possibility that local criminal activity might affect operations, and to mitigate this, the company employs the local military forces to accompany field parties away from secure areas. The Madagascar government provides a requested number of regular military troops, at a cost to the Company, to ensure security on the Property, on the work site and for the company’s equipment.

6 HISTORY

The region around the Property has primarily been explored for base metal type occurrences, although colonial geologic services were alert to all kinds of mineral potential in the region. In 1985 the Bureau de Recherches Géologiques et Minières ("BRGM") (http://www.brgm.fr/) produced a three-volume country scale compilation of all exploration and mineral inventory data in their files. Relatively little exploration and development work has been completed in south-western Madagascar after that of BRGM, and therefore, these volumes are key to retracing any historical data. Archival research by the Company has not revealed evidence of mineral exploration in the past fifty years within the Project area prior to the exploration work completed by the Company.

6.1 Property-Scale Exploration History

Prior to the exploration work completed by the Company, (then Uranium Star) in 2007 there is no record of any previous exploration activity within the Project area and no historical resource estimates exist for the area, or for Molo. Between 2007 and 2011 the Company retained Taiga Consultants Limited ("Taiga") to manage exploration activities on the Project. Table 13 shows a summary of the historical exploration activities previously on the property. This FS was informed by all previous exploration work and execution of Phase 1 of the Molo Graphite Project, and no further exploration work was completed.

Table 13: Historical Activities on the Project

Date	Activity	Company Responsible
2007	Stream sediment sampling (182 samples). Soil sampling (7.5 lines km for 1,684 samples). Prospecting (226 grab samples). Property wide reconnaissance mapping (1:25,000 scale). Detailed geological mapping on selected targets (1:5,000 scale). Trenching (11 trenches for 525m). Construction of camp. Construction of gravel airstrip. Repair road from camp to airstrip.	Taiga
	Remote sensing interpretation.	Earth Resource Surveys Inc. (ERSI)
	Airborne DIGHEM EM and magnetic survey (7,856 lines km).	Fugro Airborne Systems

Date	Activity	Company Responsible
2008	Geological mapping entire project at 1:10,000 scale. Prospecting (391 grab samples). Soil sampling (110 lines kms for 3,509 samples). Stream sediment sampling (311 samples). ScintilLoMeter survey (18 km strike length).	Taiga
	Diamond drilling (33 holes for 4,073m).	Cartwright Drilling
	Ground HLEM survey (152 km). Ground magnetics survey (419 km).	Spectral Geophysics
2009	Soil XRF survey on lines 200m apart covering 18 km of strike length.	Taiga
	Ground scintilLoMeter surveys on lines 200m apart covering 18 km of strike.	Taiga
	Trenching programme (140 Trenches for 17,105m).	Taiga
	Diamond drilling on Jaky Deposit (27 holes for 4,166m).	Boart Longyear
	Diamond drilling for metallurgical samples on Jaky (3 holes for 344m).	Boart Longyear
	Diamond drilling on Manga Deposit (24 holes for 4,422m).	Boart Longyear
2010	Diamond drilling on Manga, Manga North, Manga South, and Mainty deposits (46 holes for 8,952m).	Boart Longyear
	Prospecting (20 grab samples).	Boart Longyear
	Geologic mapping over Manga and Mainty Deposit at 1:5000 scale.	Taiga Consultants Ltd
	ERT ground geophysical survey (5.64 km). MAG ground geophysical survey (169.53 km). Gradient Array EM ground geophysical survey (128.82 km).	SOING
2011 (Phase 1)	Trenching programme (20 trenches for 1,912 m) over Fotsy and Fondrana areas.	Energizer Resources

Date	Activity	Company Responsible
	Diamond Drilling on Fondrana, Fotsy and Boko areas (10 holes for 1,157m). Metallurgical samples taken from Fond-11-01 and Fotsy 11-05.	Boart Longyear
2011 (Phase II)	Diamond Drilling on Fondrana, Fotsy, Molo and Jaky areas (19 holes for 1,701m). Metallurgical samples selected from Molo-11-07.	Boart Longyear
	Prospecting (538 grab samples) over areas of historical graphitic occurrences (BRGM) on both Green Giant and JV Property. Trenching programme (two trenches for 258m) over Fotsy and Molo areas. Geologic mapping over Fotsy, Fondrana, Jaky and Molo areas at 1:1000 scale. EM-31 ground geophysical survey (52.2 lines kms).	Energizer Resources

Exploration work undertaken in 2011 led to the discovery of the Molo deposit, which then became the focus of the 2012-2014 exploration programmes addressed in the Molo Phase 1 FS and this Expansion Project FS.

There has been no exploration on the property since 2014 that is material to the mineral resource estimate outlined in Section 14.

6.2 Metallurgical Testing

Whilst the detailed metallurgy is covered later in this Technical Report, the history of metallurgical test-work is covered here for sake of completeness.

In 2012 a 200 kg sample was taken from a test-pit over the Molo Deposit and sent to Mintek (South Africa) for carbon recovery testwork. A duplicate sample from the same test pit was sent to Lac Des Iles for analysis.

In 2013 a 200t bulk trench sample was collected for pilot plant testing by SGS Lakefield (Photo 3). It is the same bulk sample on which the expansion process design is based and covered in section 12.

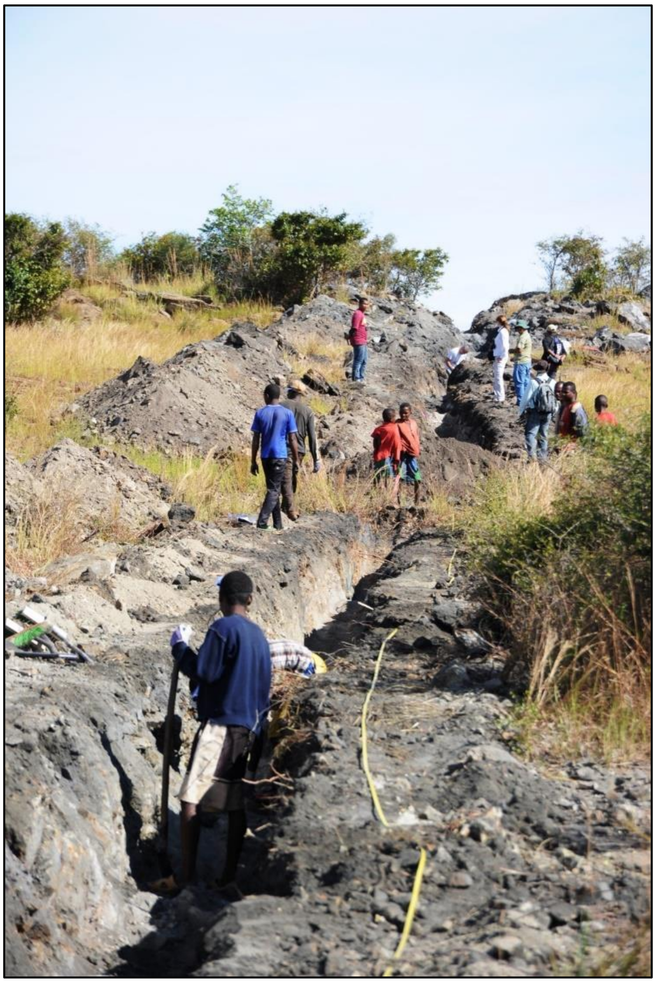


Photo 3: Trenching for bulk sampling May 2013

6.3 Mining Studies

Previous mining studies undertaken on the Project include a Preliminary Economic Assessment undertaken by DRA Mineral Projects (Pty) Limited in April of 2013. This study evaluated an open pit mining approach combined with processing through a sequence of crushing and screening, grinding, flotation, thickening and drying, producing a primary concentrate of graphite of various grades and flake sizes.

The open pit-pit mining option was evaluated on the basis of contractor mining. It was estimated that the pre-production period prior to implementation would be approximately two years and the LoM plan obtained from the pit optimization study would be 27 years.

In addition to the plant design, allowances were made for the purification of graphite concentrate through a caustic hydration and hydrochloric acid dehydration process. The plant processing rate was to be 1.16 Mtpa, with a concentrate grade of 88% without chemical treatment.

6.4 Historical Production

During the completion of this FS, Molo Graphite Phase 1 construction was completed and entered a commissioning phase. For the commissioning mining activity included the stripping of waste materials to further expose the ore body. In order to commission the process facility low-grade ore and high-grade ore was extracted from the ore body and stockpiled at the process plant, totalling 8,160t by end August 2023.

7 GEOLOGICAL SETTING AND MINERALISATION

The regional geology and geology of the Molo deposit presented below is taken mainly from Hancox and Subramani (2013). It should be noted, however, that the local scale geology of southern Madagascar and of the Molo deposit has more recently been described by Scherba et al. (2018).

7.1 Regional Geology

Madagascar comprises a fragment of the African Plate, which rifted from the vicinity of Tanzania at the time of the breakup of Gondwana, some 180 million years ago. At that time Madagascar remained joined with India, moving east-by-south until the late Cretaceous, (approximately 70 million years ago), where upon the two land masses split apart. On a regional scale Madagascar can be described as formed by two geological entities, a Precambrian crystalline basement, and a much younger Phanerozoic sedimentary cover (Figure 8) below that hosts potentially economic coal deposits. The central and eastern two-thirds of the island are mainly composed of Neoproterozoic-aged, crystalline basement rocks, composed of a complex mélange of metamorphic schist and gneiss intruded by younger granitic and basic igneous rocks. The Phanerozoic sedimentary cover is largely restricted to the western side of the island and is Carboniferous to Permian-Triassic. These rocks correlate with the Karoo Super Group successions of sub-Saharan Africa, which was widespread in the former super continent of Gondwana.

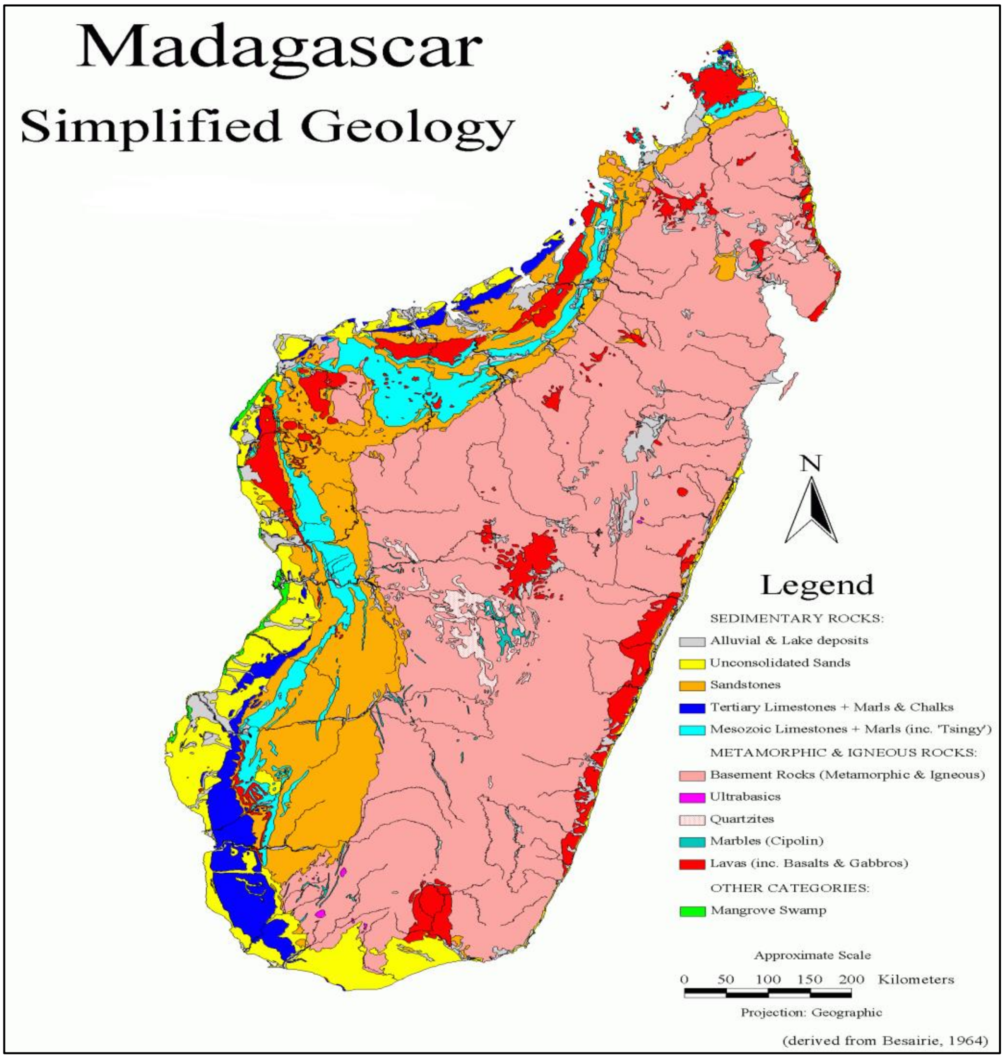


Figure 8: Geological Map of Madagascar Showing the Distinctive Crystalline Basement and Sedimentary Basins in the West (Source: Besarie (1964)

The geology of the basement of Madagascar is composed of inter-continental tectonic blocks made up of ancient poly-deformed, high-grade metamorphic rocks and later igneous intrusions. The tectonic and metallogenic basement framework was originally sub-divided into four blocks (Besarie, 1967), these being the:

- Northern Bemarivo Block.
- North-eastern Antongil Block.
- Central Antananarivo Block.
- Southern Bekily Block.

The Molo deposit lies entirely within the bounds of the Bekily Block (Figure 9) below. Later authors (e.g., Pitfield et al., 2006) divided the Precambrian basement of Madagascar in a somewhat different manner, with nine tectono-metamorphic units (Figure 10).

In the case of the region around Molo, the tectonic blocks and the tectono-metamorphic units cover a nearly identical area, and as such these divisions can be used inter-changeably.

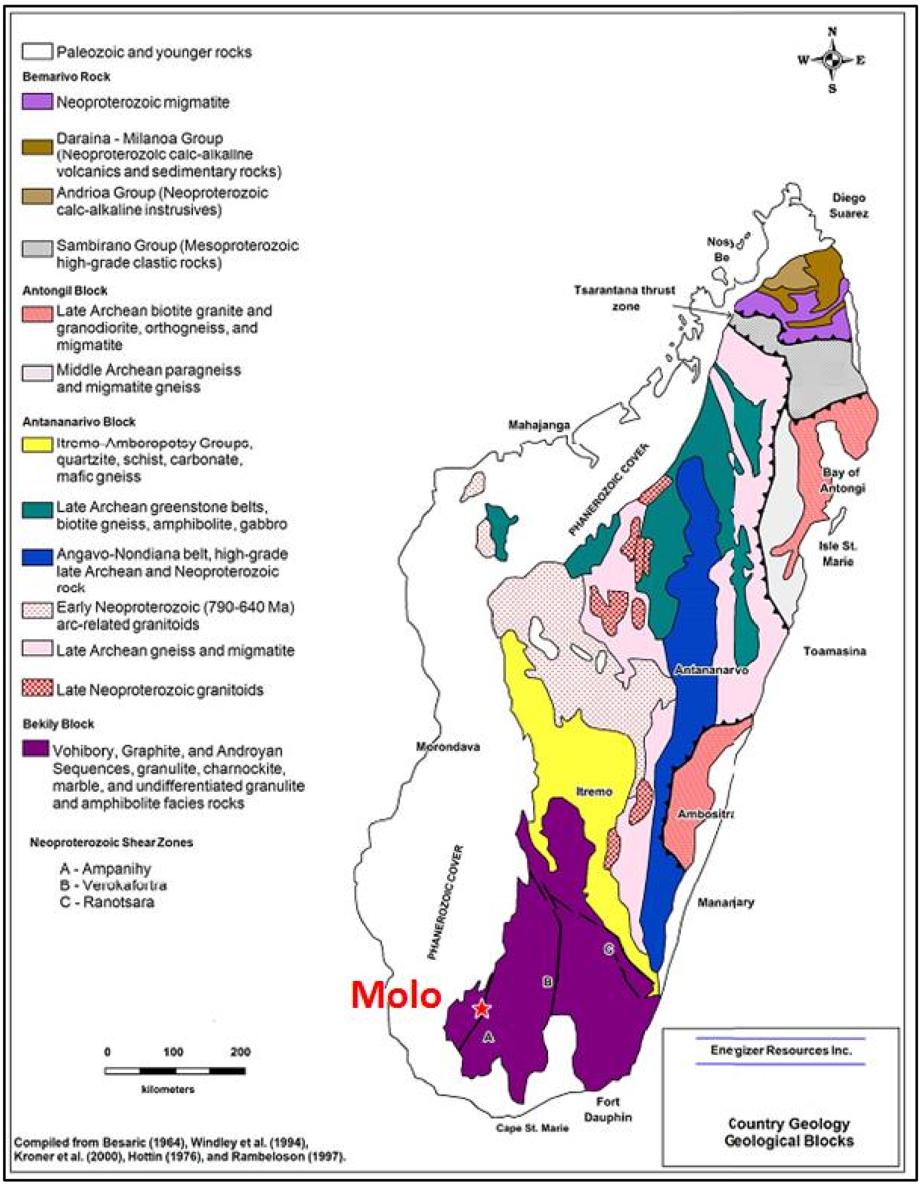


Figure 9: Country Geology: Geological Blocks (source: AGP Mining Consultants (2011)

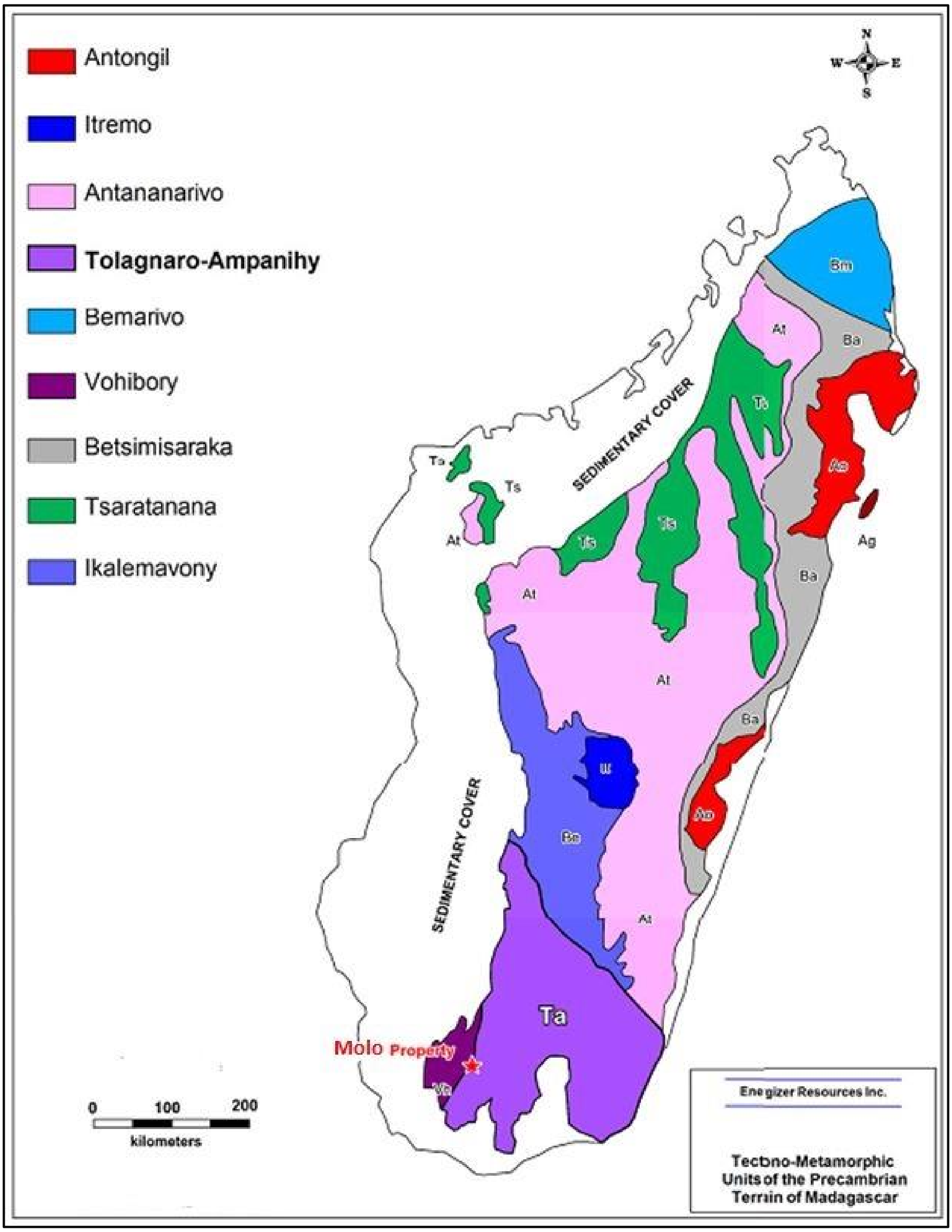


Figure 10: Tectono-Metamorphic Units of the Precambrian Terrain of Madagascar (source: AGP Mining Consultants, 2011)

7.1.1 Tectonic History of Southern Madagascar

Southern Madagascar forms part of the Mozambique Mobile Belt and is made up of a section of lower Proterozoic crust that underwent granulite-facies metamorphism during the Pan-African Orogeny (Paquette et al., 1994). Three crustal units separated by north-south trending vertical shear zones make up this area (Figure 11).

Each of these units experienced granulite facies metamorphism with temperatures between 750˚C and 800˚C. The associated pressures during the Pan-African Orogeny in this area range between 3 to 11 kilobars ("kbar") with a decreasing trend from west to east (Pili et al., 1997).

The Bekily block, (also referred to as the Androyen region, or the Tolagnaro-Ampanihy tectono-metamorphic unit), forms a vast high-grade meta-sedimentary (paragneiss) terrane that has been metamorphosed to granulite facies conditions. This region comprises a complex Neoproterozoic terrain of high-grade metamorphic rocks, with a history of polyphase deformation. Two prominent north-south trending late Neoproterozoic ductile shear zones, the Ampanihy and Vorokafotra shear zones, crosscut the region. A third set of en-echelon shears forms part of the early Palaeozoic Ranotsara Shear Zone that cuts the basement in a north-west-south-east direction over a strike length of over 400 kms.

- De Wit et al. (2001) recognize four episodes of deformation and metamorphism. The two early episodes of simple shear deformation (D1 and D2), during which north-east verging recumbent sheath folds and ductile thrusts were formed, are dated between 647 Ma - 627 Ma. Early prolate mineral fabrics (L1/L2) are preserved in massif type anorthosite bodies and their marginal country rocks.
- D1 and D2 deformation was followed by a 10 Ma to 15 Ma period of static, annealing metamorphism when bulk shortening (D3) took place. D2 and D3 deformations are coaxial, but are separated in time by leucocratic dykes that intruded between 620 Ma and 610 Ma. Between 609 Ma and 607 Ma, D3 deformation was focused zonally, forming the prominent north–south shear zones. Oblate strain resulted in a strong composite D2/D3 fabric defined by sub-vertical S-tectonites and sub-horizontal intersection lineations.

A variety of post-D3 pegmatites accompanied the following 85 million years of relatively static annealing and metasomatic / metamorphic mineral growth. Numerous occurrences of phlogopite, uranium, and rare earth elements are associated with these pegmatitic bodies. A continuum of concordant monazite dates of between 605 Ma and 520 Ma suggests that this thermal event is part of an extended period of low-pressure (3k to 5k bar) charnockite producing event.

The D4 deformational event recorded within the Ranotsara Shear zone overlaps with the youngest parts of the regional metamorphic conditions. Between 530 Ma to 490 Ma, prevailing low pressure, high temperature amphibolite-granulite facies rapidly gave way to greenschist facies conditions. (Figure 11 below).

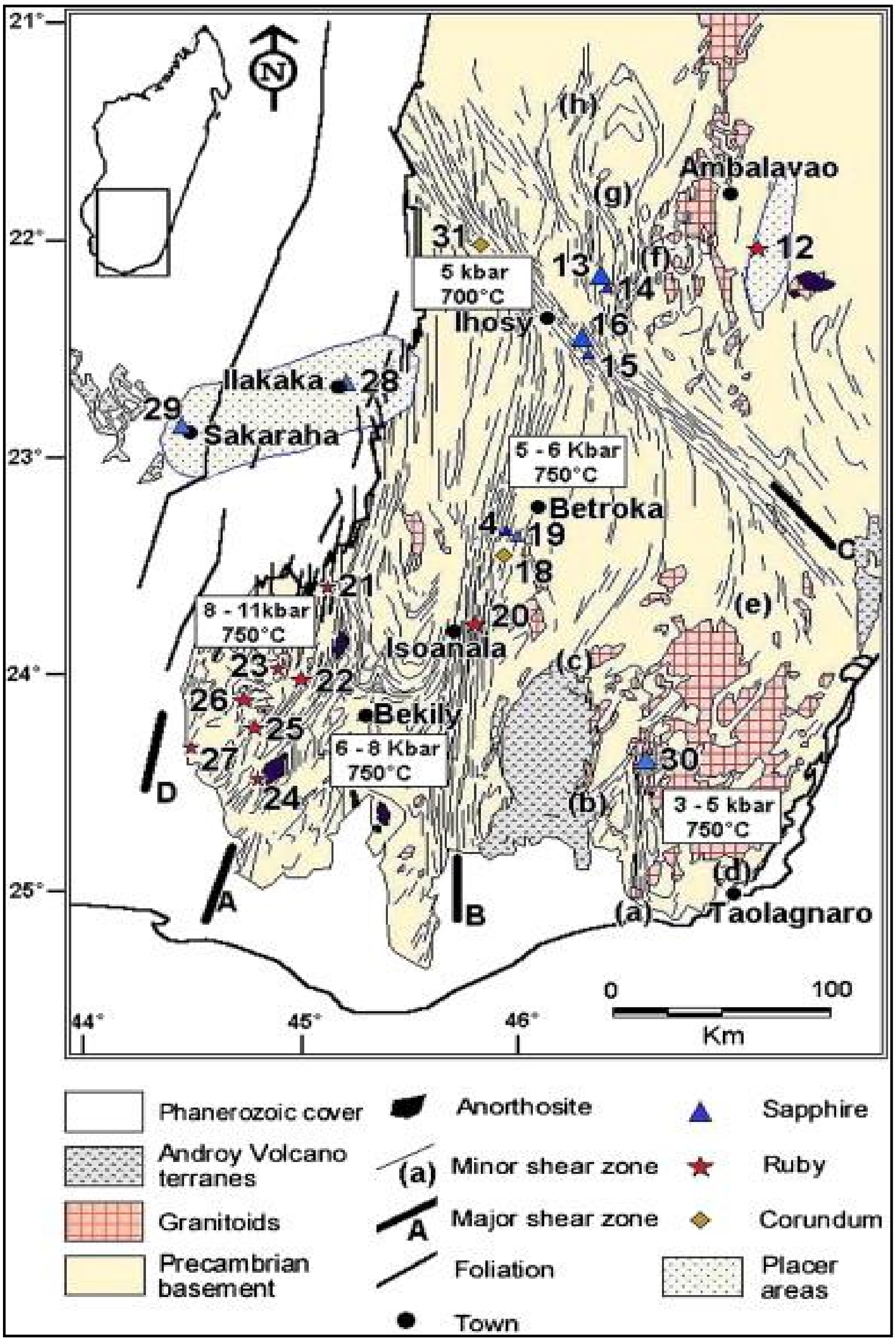


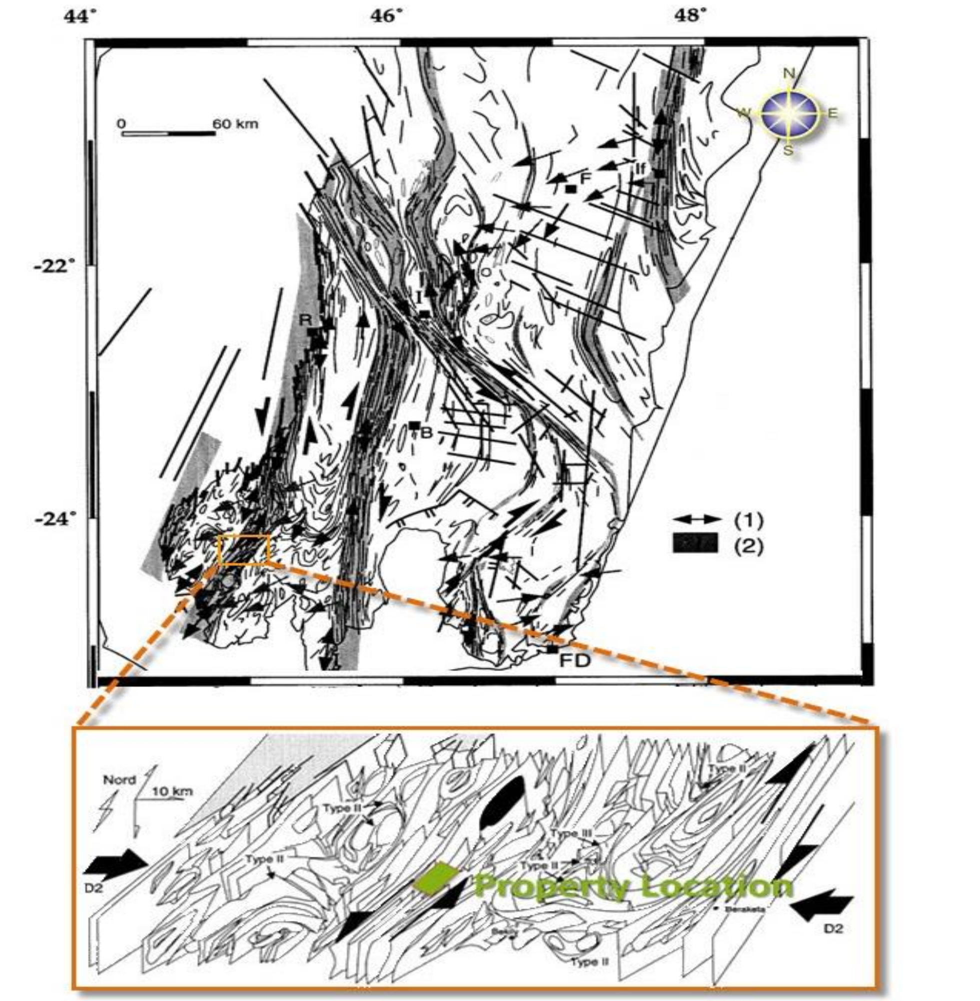
Figure 11: Structural and Lithological Sketch Map of South-East Madagascar Showing the Positions of the Major Shear Zones (source: Rakotondrazafy et al.,2008)

7.2 Regional Geology as it Relates to the Molo Deposit

The Molo deposit occurs within the regional Ampanihy Shear Zone (Figure 12Figure 12) which delineates the western edge of the Androyen domain. The most conspicuous feature of rocks found within this shear zone is their well-developed north-south foliation and vertical to sub-vertical nature.

Martelat et al. (2000) state that this observed bulk strain pattern is clearly related to a transpressional regime during bulk horizontal shortening of heated crust, which resulted in the exhumation of lower crustal material. Figure 12 below illustrates the general position of Molo relative to the D2 regional strain pattern and the resulting Ampanihy Shear zone.

The Project area is underlain by supracrustal and plutonic rocks of late Neoproterozoic age that were metamorphosed under upper amphibolite facies and deformed with upright north-northeast-trending structures. The supracrustal rocks involve migmatitic (± biotite, garnet) quartzo-feldspathic gneiss, marble, chert, quartzite, and amphibolite gneiss. The metaplutonic rocks include migmatitic (± hornblende / diopside, biotite, garnet), feldspathic gneiss of monzodioritic to syenitic composition, biotite granodiorite, and leucogranite.



Finite strain pattern of southern Madagascar with Ampanihy transpressive lithospheric shear zone shown in inset image

Modified from Martela, 2000

Figure 12: Position of the Molo Project Area within the Overall Strain Pattern Documented for Southern Madagascar

Descriptions of the individual lithological units identified by the Company, which are relevant to Molo, are included below.

7.2.1 Lithological Descriptions of Individual Rock Formations

7.2.1.1 Amphibolitic Gneiss

Dark grey to black, mesocratic to melanocratic, medium to coarse grained, sub-equigranular to porphyroblastic amphibolitic gneiss and amphibolite. Amphibolitic gneiss forms one, or more major continuous bands in the eastern part of the permit, intercalated with quartzo-feldspathic gneiss and spatially associated with marble. In the central portion of the detailed map area, amphibolitic gneiss forms local bands, or lenses intercalated with quartzo-feldspathic gneiss and marble.

7.2.1.2 Meta-quartzite

White to greyish white, weakly to moderately layered and foliated, coarse to medium grained quartzite. Brecciated quartzite with isoclinally folded layering is locally associated with dark brown ferruginous gossan. Un-brecciated quartzite very locally contains narrow, concordant, and discontinuous seams of gossan.

7.2.1.3 Grey-white Chert

Mottled greyish-white, massive to brecciated, hyalocrystalline graphite-bearing chert, (or possibly siliceous rhyodacite). Grey-white chert displays evidence of polyphase brecciation, involving cm to mm scale, angular white siliceous fragments in a relatively early translucent grey siliceous (chalcedony) breccia matrix, and/or a later opaque brown ferruginous gossan breccia matrix.

7.2.1.4 Brown Fe-carbonate Chert

Tawny (yellowish) brown to reddish brown and chocolate brown, massive, hyalocrystalline opaque, graphite-bearing Fe-carbonate chert, variable biotite, and/or specularite. Brown chert, like grey-white chert, contains a small amount (≤1%) of fine-grained disseminated graphite, as well as variably small amounts of fine-grained disseminated biotite and/or specularite. Brown chert represents a widespread Fe-carbonatized alteration facies of grey-white chert, and both occur within the same chert masses. Brown chert is intimately associated with brown marble and ferruginous gossan.

7.2.1.5 Ferruginous Gossan

Dark purplish brown to black, dense, massive to brecciform and quasi-layered, aphanitic to fine-grained, siliceous ferruginous gossan. The gossan is variably highly siliceous to moderately siliceous and pitted, composed in part of Fe carbonate, (siderite-ankerite), and generally contains disseminated to clustered, fine-grained specularite, biotite, and/or graphite. Siliceous ferruginous gossan occurs as:

Breccia matrix of late-stage chert breccia and quartzite breccias.

Concordant layers inter-calated with chert and marble and discontinuous concordant seams in quartzite discordant masses cutting regional structure in quartzo-feldspathic gneiss and marble.

Siliceous ferruginous gossan is locally associated with cm scale patchy masses of green, opaque calc-silicate, or bright green amorphous and resinous calc-silicate mineral.

7.2.1.6 *Quartz Feldspar Gneiss*

Light grey to white, migmatitic, well foliated, and locally lineated, leucocratic to hololeucocratic, generally medium-grained, (to fine, or coarse grained), ubequigranular to porphyroblastic biotite-garnet Quartzo-feldspatic gneiss comprises a mixture of fundamental constituent lithologies, dependent on the relative abundance, or absence of biotite and garnet.

7.2.1.7 *Feldspathic Gneiss*

Pinkish grey to pink, migmatitic, foliated, medium to coarse grained, leucocratic (± hornblende / diopside, biotite, garnet) feldspathic gneiss. The feldspathic gneiss is comprised of a mixture of quartz-poor constituent lithologies.

7.2.1.8 *Structural Geology*

In 2010 a structural interpretation was undertaken based on the 2007 Fugro Airborne Surveys (http://www.fugroairborne.co.za/) ("Fugro") helicopter-borne frequency domain electromagnetic (DIGHEM V) multi-coil, multi-frequency, electromagnetic and high sensitivity cesium magnetometer geophysical survey (Butler, 2010; in Desautels et al., 2011). Only magnetite bearing units were capable of being interpreted, except where putative intrusions cross cut the main fabric in a magnetised area. This work showed the Green Giant Property to be dominated by structures associated with the Ampanihy Shear Zone. Butler (2010; in Desautels et al., 2011) specifically identified three magnetic domains associated with the Ampanihy shear system:

- Zones where magnetic units are parallel, or near parallel to the walls of the domain. In these regions, the shearing has reduced intrafolial folds into sheared out 'tectonic fish'. There are also broad zones where a low content of magnetite (± pyrrhotite), may be present which most likely represent a different metamorphic mineral assemblage, (different pressure and temperature conditions), or pre-metamorphic alteration and/or rock types.
- Zones where magnetic units vary from parallel to a high angle at the domain boundary. These regions are interpreted to be the intrafolial fold remnants of sheath folds. The boundary shear of these domains may represent a sheared-out early thrust, or high angle fault.
- Zones with refolded chaos folds in domain lozenges. These regions occur in the north-central portion of the study area and may be a remnant of a broad F3 episode enclosed within intense zones of ductility.

7.3 Molo Property Geology

The Molo graphitic zone is delineated over the western, isoclinal antiform of a surficially exposed, steeply plunging (84°), antiform-synform pair consisting of graphitic schist and graphitic gneiss exposed for a strike length of over 2 km and a width of 750m (Figure 13).

The limbs of the Molo antiform are parallel to the regional Ampanihy shear zone, and both dips steeply to the west at 85°. Outcrop mapping and trenching on Molo has shown the surface geology to be dominated by resistant ridges of graphitic schist (Figure 14) and graphitic gneiss, as well as abundant graphitic schist float.

Geological modelling (Figure 13) has shown that the deposit consists of various zones of mineralised graphitic gneiss, with a barren footwall composed of garnetiferous gneiss (Figure 13). The host rock of the mineralised zones is graphitic gneiss.

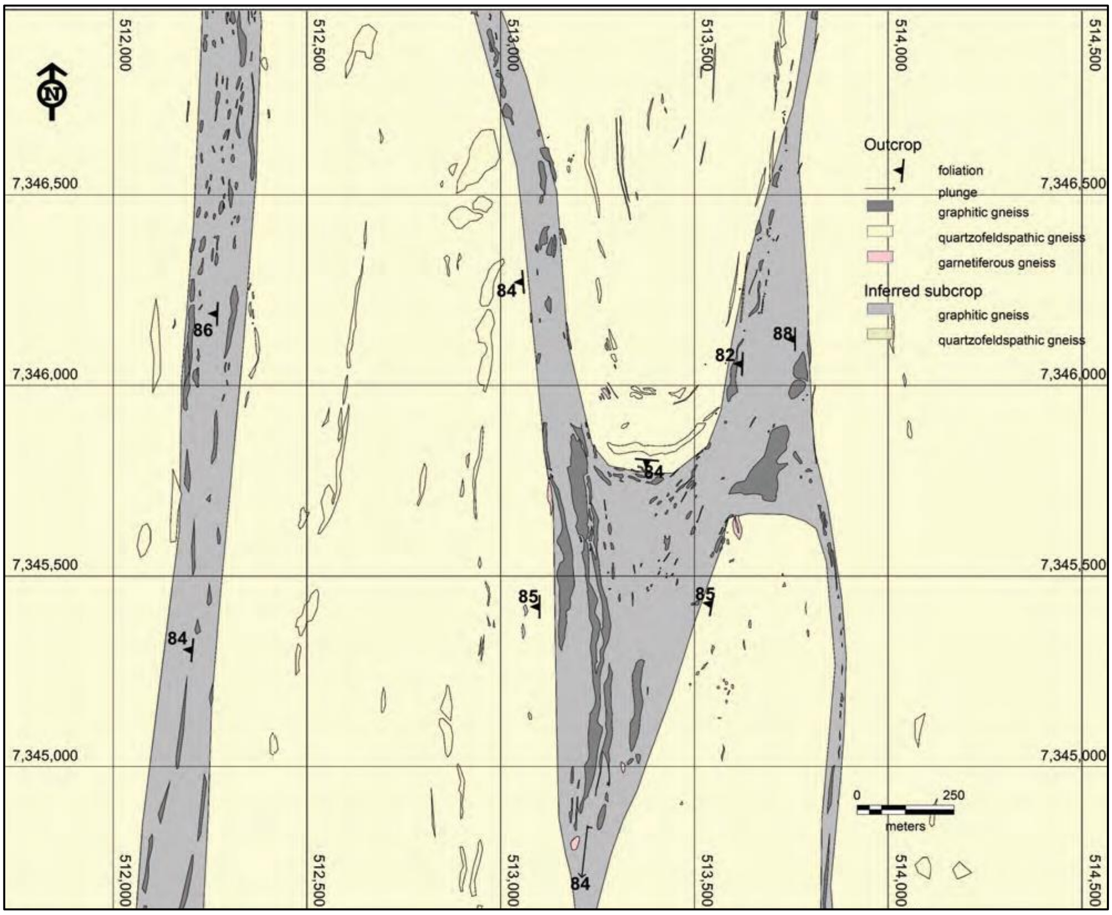


Figure 13: Map Showing the Surface Geology of the Molo Deposit

Figure 14: Outcrop Exposure of Graphitic Schist on the Molo Project

7.4 Graphite Mineralisation on Molo

Graphite mineralisation on the Molo Project is hosted in schists, believed to originally have been mud stones, silt stones, and sand stones. Graphitic mineralisation in the Molo Project area is bimodally distributed, with low-grade and high-grade zones having carbon cut-off grades of 2% and 4% C, respectively. High-grade mineralization is associated with metamorphosed silt stones and mud stones, while low grade mineralization is associated with rocks interpreted to represent metamorphosed sand stones, which are interpreted to be more favourable hosts for large and jumbo flake graphite (Scherba et al., 2018).

The Molo graphite deposit appears to have resulted from many mineralizing events, which extended over a period of time that may range from ca. 900 to ca. 490 Ma. These include the graphitization during the emplacement of anorthosite complexes, graphitization in a high-strain regime under high pressure and high temperature granulite facies metamorphism during the collision of the Androyen domain with the Vohibory domain, graphite refining and re-crystallization believed to have taken place during East Gondwana and West Gondwana collision, and the formation of post-collisional hydrothermal vein graphite during orogenic collapse. The super-imposition of the tectono-metamorphic history of southern Madagascar on a sedimentary sequence in which the protoliths were rich in organic carbon has resulted in flake graphite mineralization with high carbon purities and large flake sizes (Scherba et al., 2018).

(Figure 15 below).

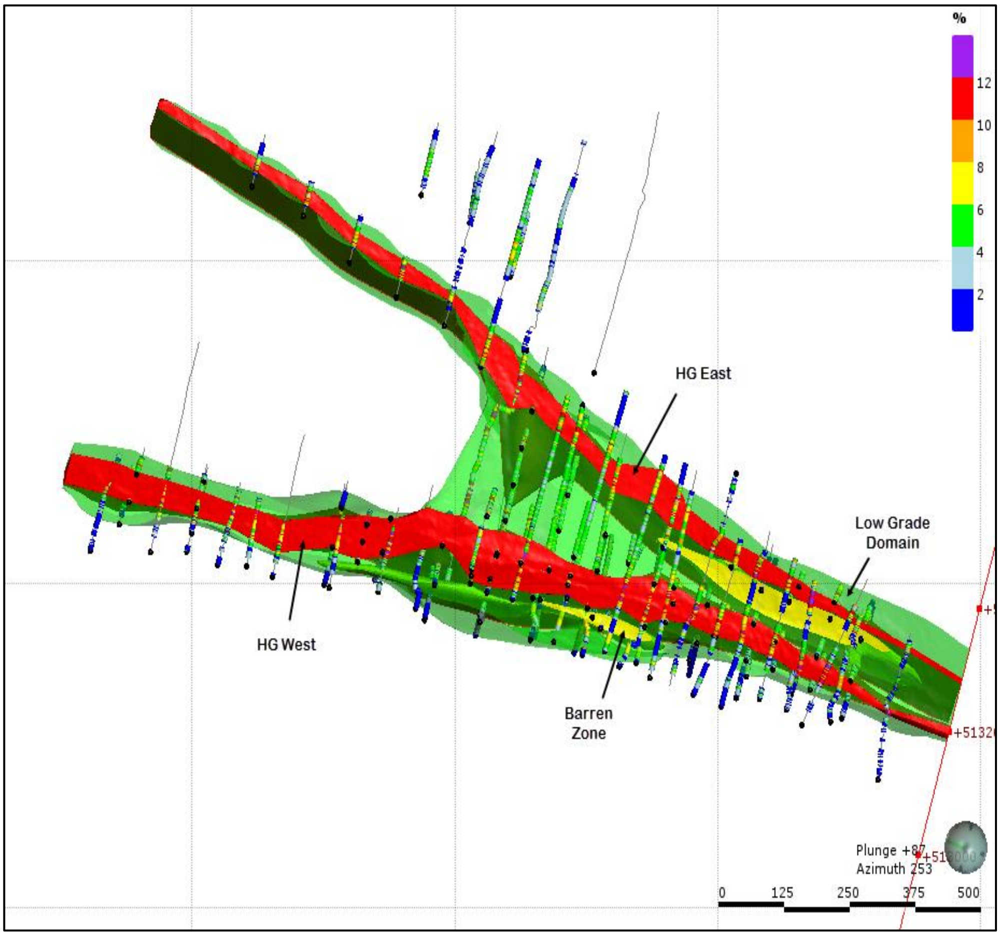


Figure 15: Geological Model of the Molo Deposit Showing the Nature of the Two Tightly Oppressed 'High" Grade (Hg East and Hg West) Mineralized Zones

8 DEPOSIT TYPES

The Molo Project hosts a flake graphite deposit, which is one of a variety of graphite deposit types, these are discussed below:

8.1 Graphite Deposit Types

Graphite is one of the three familiar naturally occurring forms of the chemical element Carbon ("C"). The other two varieties are amorphous carbon, (not to be confused with amorphous graphite) and diamond. Graphite may be synthetically produced, or extracted from a natural source. Most natural sources are considered one of three main types, amorphous, flake, or vein.

The following is taken mainly from Kogel et al. (2006):

- Graphite is widely distributed throughout the world, occurring in many types of igneous, sedimentary, and metamorphic rocks.
- Many occurrences, however, are of little economic importance.
- The more important occurrences are those found in metasomatic-hydrothermal deposits and in sedimentary rocks that have been subjected to regional, or contact metamorphism.

Identified Economic deposits of graphite include the following:

- Flake graphite disseminated in metamorphosed, silica rich rock.
- Sedimentary rock.
- Flake graphite disseminated in marble.
- Amorphous deposits formed by metamorphism of coal.
- Carbon-rich sediments.
- Veins filling fractures, fissures, and cavities in country rock.
- Contact metasomatic or hydrothermal deposits in metamorphosis.
- Natural graphite of economic value can be divided into two main classes, these being:
- Disseminated flake.
- Crystalline vein (fibrous, or columnar).

Most, if not all, of the world's deposits of disseminated flake graphite occur in metamorphic rocks of Precambrian age. Flake graphite is a lamellar form found in metamorphic rocks, such a marble gneiss, and schist. Each flake is separate, having crystallized as such in the rock. In many cases, pegmatitic veins have intruded the rocks. Flake graphite has recently been identified as a critical and strategic material due to its essential applications in the aerospace and energy sectors (Robinson et al., 2017) and its role as the primary anode component in lithium-ion batteries (Benchmark Mineral Intelligence, 2017a).

Crystalline vein graphite, (also called lump, or high crystalline graphite), is normally found in well-defined veins, or pocket accumulations along intrusive contacts of pegmatites with limestones. The enclosing wall rock is not necessarily graphitic.

This type of deposit assumes the character of a true lode. The graphite in these deposits is of two types, foliated and columnar. The Sri Lankan graphite deposits are of vein type.

8.2 Graphite Mineralisation on Molo

Petrographic descriptions undertaken on thin sections of selected rocks of the Manga vanadium deposit submitted for metallurgical analysis to Mintek (www.mintek.co.za/) in 2010 identified 17.17% modal graphite from the silicate composite, and 15.87% modal graphite from the oxide composite samples. Three additional composite samples were submitted to Mintek at the conclusion of the 2010 exploration program. The Quantitative Evaluation of Minerals by Scanning Electron Microscopy ("QEMSCAN") analysis of these samples quantified a graphite composition of 4.09%, while the head chemical analysis quantified a graphitic carbon content of 3.87%.

The identification of graphite as a potential credit to the Company's vanadium resources led the Company's geologists to conduct a reconnaissance exploration program with the goal of delineating new graphitic trends and comparing them to those associated with vanadium mineralisation. During the period of this reconnaissance exploration program various surficial graphitic trends were identified on the Green Giant Property. These graphite trends were visually determined to be of both higher carbon content, and larger flake size than those associated with the vanadium resource mineralisation. Samples from these mineralised zones were submitted to Mintek for analysis, as well as to the North Carolina State University ("NCSU") Minerals Research Laboratory in Asheville, North Carolina.

9 EXPLORATION

The identification of graphite as a potential credit to the Company's NI 43-101 compliant vanadium resources, (Scherba and Chisholm, 2008), led to a reconnaissance exploration programme being undertaken on the Property in September 2011, with the goal of delineating new graphitic trends. Activities during this phase of exploration included prospecting, grab and trench sampling, and diamond drilling. Based on the results of this programme, the Company launched a second phase of exploration in November 2011. The objective of this second programme was to use geophysical techniques to delineate additional graphite mineralisation, as well as to drill test the known graphitic horizons.

The signing of the JV agreement with Malagasy in November 2011 prompted additional exploration to ascertain the industrial mineral potential of the JV Property area. Exploration activities consisted of geologic mapping, prospecting, and sampling, (including metallurgical), ground geophysical surveying (EM-31), trenching, and diamond drilling. As a result of work undertaken during 2011, the Molo graphite prospect was identified and targeted for additional work, which was undertaken between May 2012 and June 2014.

9.1 Geological Mapping

A series of excellent 1:100,000 scale geological maps (1952-53) are available for the region surrounding Molo (Fotadrevo-Bekily, Ianapera, Sakamena-Sakoa), with the area covered by the 1:100,000 scale topographic map #H-60 Fotadrevo.

Various mapping Projects have been completed on the then entire Green Giant Property, with the major emphasis being strictly dedicated to the commodity of interest at that time. In 2007 Taiga completed a property wide reconnaissance mapping Project (1:25,000 scale). Greater detailed (1:5,000 scale) geological mapping was later undertaken to compliment the larger scale version in areas of geologic interest. During the 2008 field season, Taiga again completed a property wide mapping programme, however, at a notably smaller scale than before (1:10,000 scale).

9.2 Trenching

No known historical trenching is documented on Molo. During the 2011 field season a number of the new graphite rich areas were trenched including Molo. Initial graphitic carbon results from the 2011 trenching were encouraging in that they showed multiple graphic horizons present in each zone, of significant widths and grades. Because of this and coupled to the size of the electro-magnetic signature, the 2012 programme focussed on Molo and an additional 22 trenches were excavated.

Additional trenching was undertaken on Molo during May of 2013 as part of a bulk sampling exercise (Photo 4). Subsequently an additional nine trenches (totalling 1,876m), have been excavated as part of the 2014 exploration programme.

Photo 4: Trenching for the Bulk Sample on Molo, May 2013

A plan map showing the positions and grades of all the trenches excavated on the Molo deposit is provided as Figure 16 below.

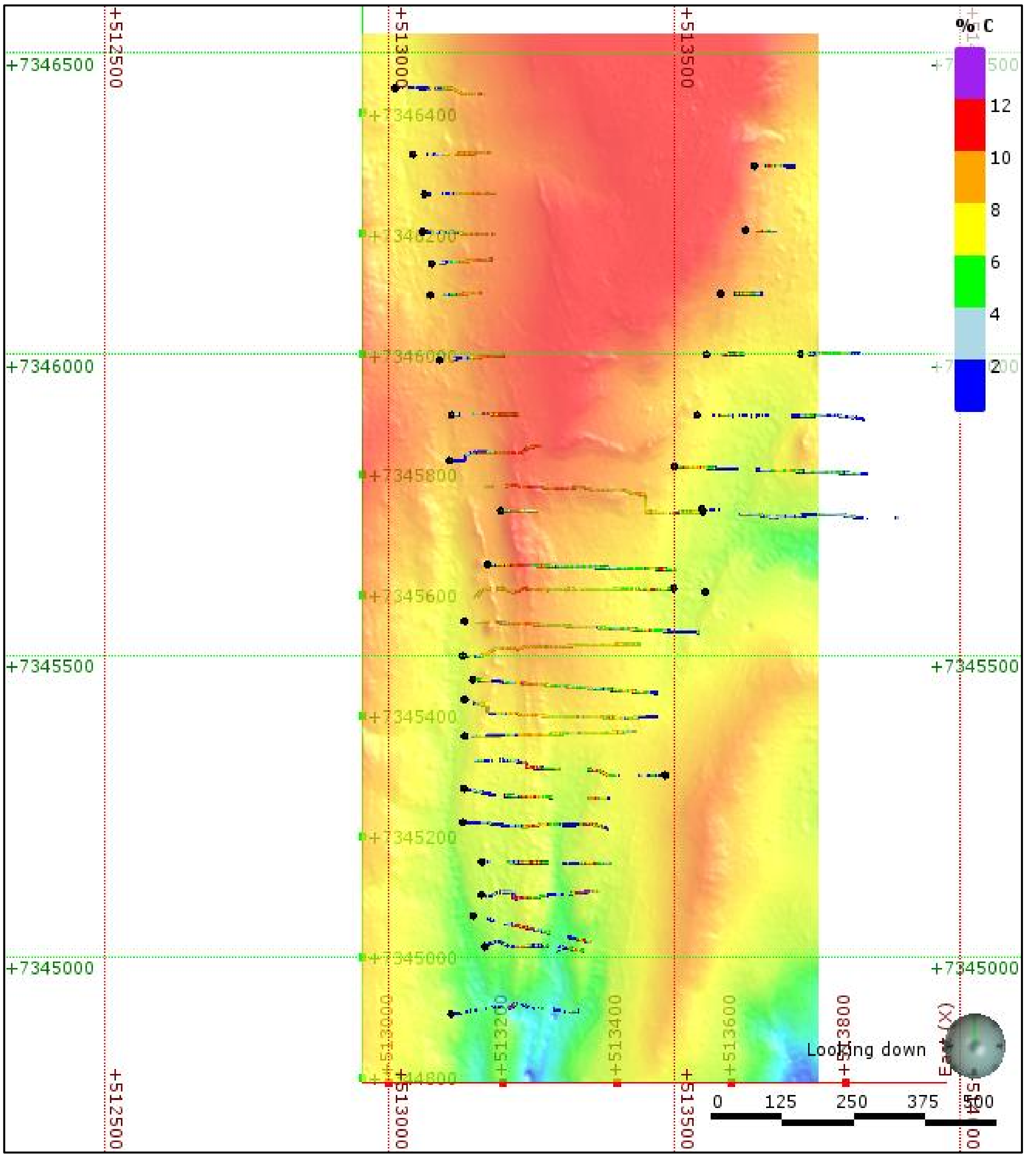


Figure 16: Plan Showing the Positions and Grades of All the Trenches Excavated on the Molo Deposit, Overlain on the Topographic Elevation Map

9.3 Trench Sampling

Standardized sampling methods includes 2m long continuous chip samples approximately four cm wide being collected along the northern edge of the trench floor, consisting of about 3 to 4 kg of material per sample. The following procedural steps were taken during the sampling and mapping process:

- Plastic sample bags were sequentially numbered with a unique series from pre-printed sample books. The Quality Assurance / Quality Control ("QA/QC") sample numbers are flagged at this point for later insertion.
- The trench floor is swept clean with hand brooms to ensure there is no contamination from rubble, or fines.

- Two technicians use hammers and chisels to gently dislodge the weathered rock along the channel profile.
- A third technician follows behind to collect the sample material, first verifying the sample tag is in the bag, then matching the sample bag number and the sample book interval.
- The sample bag was sealed with a zip tie, with the sample tag inside the bag.
- Two technicians follow behind the samplers and clean / scrape the north wall of the trench to allow better visual inspection of any structures, and to remove any debris, or 'polishing' which may have occurred during excavation.
- A geologist, or qualified technician using scaled paper inspects the north wall of the trench and records structures, mineralization, depth, and any other notable aspects.

All samples were brought back to the camp each night for storing in the secured facility at Fotadrevo until shipment.

Samples taken from the 2012-2014 trench exploration programmes were subject to stringent QA / QC and their lengths and percentage carbon are presented in Table 14 below. This data was used in the original estimation process.

Table 14: Samples taken from the 2012-2014 Trench Exploration Programmes

Trench	From (m)	To (m)	Length (m)	C%
MOLO-TH-12-01	28	318	290	6.58
MOLO-TH-12-02	2.5	358	355.5	6.01
MOLO-TH-12-03	5	380	375	7.74
MOLO-TH-12-04	37	119	82	8.89
MOLO-TH-12-05	32	90	58	7.4
MOLO-TH-12-06	45	125	80	6.56
MOLO-TH-12-07	52	138	86	7.34
MOLO-TH-12-08	88	140	52	7.65
MOLO-TH-12-08	186	286	100	7.25
MOLO-TH-12-09	38	128	90	6.92
MOLO-TH-12-09	166	220	54	8.79
MOLO-TH-12-10	92	121	29	7.18
MOLO-TH-12-10	214	230	16	5.06
MOLO-TH-12-11	34	94	60	5.01
MOLO-TH-12-11	158	194	36	5.74
MOLO-TH-12-12	84.5	257	172.5	5.68
MOLO-TH-12-13	16	52	36	4.51

Trench	From (m)	To (m)	Length (m)	C%
MOLO-TH-12-13	74	320	246	6.55
MOLO-TH-12-14	82	176	94	7.87
MOLO-TH-12-15	84	152	68	6.94
MOLO-TH-12-16	80	96	16	4.37
MOLO-TH-12-18	38	65	27	4.88
MOLO-TH-13	0	299	299	6.14
MOLO-TH-14	44	222	178	6.32
MOLO-TH-15	27.3	112	84.7	5.88
MOLO-TH-16	32.6	108	75.4	6.82
MOLO-TH-17	0	332	332	6.15
MOLO-TH-18	20	351	331	5.58
MOLO-TH-19	24	298	274	5.37
MOLO-TH-20	88	250	162	7.13
MOLO-TH-21	59	212	153	6.93
MOLO-TH-22	48.6	119	70.4	6.57
MOLO-TH-23	3.3	69.3	66	7.26
MOLO-TH-24	27	65	38	8.09
MOLO-TH-25	26	66.6	40.6	6.51
MOLO-TH-26	18.6	46	27.4	7.63
MOLO-TH-27	24.6	50.6	26	6.86

9.4 Ground Geophysical Surveying

During 2011 an EM31-MK 2 (“EM31”) ground conductivity instrument was obtained to aid determination of the overall extents of the graphitic horizons delineated during the previous year’s field mapping and prospecting activities. The EM31 geophysical tool was invaluable in delineating the extents of the graphitic zones, as well as their continuity. The Company’s geotechnical team conducted a survey consisting of 100m spaced lines and 25m stations. In total a 160.5 km line of EM31 surveying was completed over five target areas.

This data was used to plan the original (2011) boreholes drilled on Molo (Figure 17).

During the 2011 programme an EM31 geophysical survey was also conducted over the Molo deposit area concurrently with prospecting and subsequent geological mapping of the stronger graphitic zones. The survey (Figure 17) aided in outlining a zone length of over 2 km, with an aggregate EM31 measured strike length of 10 km. This, along with the confirmation of five strong graphitic horizons, supported further trenching and drilling.

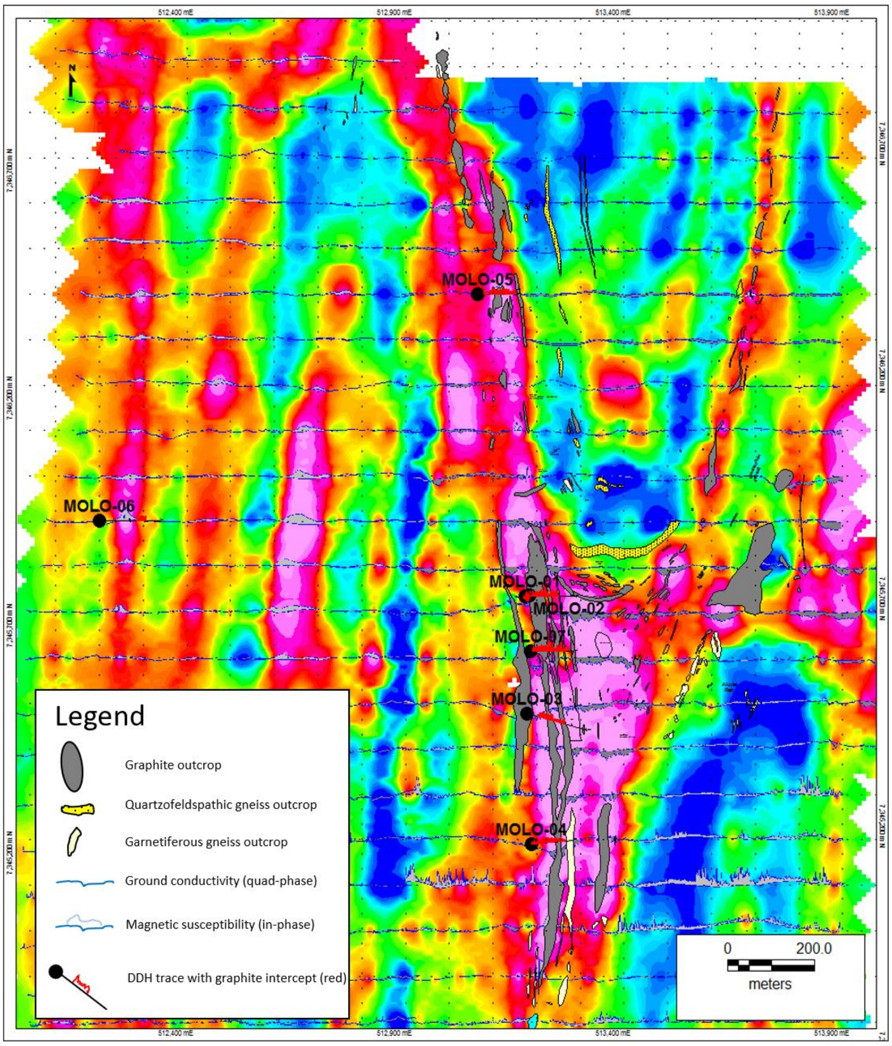


Figure 17: EM31 Geophysical Survey Map of the Molo Deposit Showing the Positions of Holes Molo-01 to Molo-07 Drilled During 2011

Bright Pink areas are interpreted graphitic areas and the grey areas represent surficial / mapped graphite outcrops. The north-eastern limb was the focus of additional work during 2014.

9.5 Prospecting and Sampling

Property scale prospecting and grab sampling was conducted during earlier exploration programmes. Prospecting typically consisted of a preliminary stage in which areas were covered on a large scale to determine Vanadium and Graphite potential.

Upon discovery of any notable potential mineralisation, a larger group of prospectors were sent to the area of significance. This then allowed much more of the Property to be 'ground truthed' and, where applicable, sampled in an intensive manner to gain an understanding of all the zones.

During 2011, the Company's employees thoroughly covered the Green Giant property with special interest in graphitic showings. With the addition of the JV Property, a much larger area had to be investigated, which required helicopter assistance to access remote and marginally accessible areas. Over the course of six days the Company's technical team visited a variety of notable graphite localities including Molo. During 2012, the Molo Project area was subject to an intensive grab sampling programme, which resulted in a total of 344 samples being collected.

10 DRILLING

No known historical diamond drilling is documented for the Property. Numerous diamond drilling programmes have been conducted over the Green Giant, (formerly the Three Horses) property in the past and these have previously been covered in previous reports (Scherba and Chisholm, 2008, McCracken and Holloway, 2009; and Desautels et al., 2010). Only work directly, or indirectly pertaining to the Molo deposit is, therefore, covered here.

Due to the realization that a significant graphitic resource probably existed on the Green Giant-JV Property, a reconnaissance diamond drilling programme was implemented in 2011 to test the viability and potential of the graphitic prospects.

The 2011 diamond drilling commenced in early November and ran through to mid-December. During this programme the Company entered into a JV agreement with Malagasy Minerals and this phase, therefore, was re-focussed on the new graphitic prospects on the JV areas including Molo.

Of the 2011 diamond drilling, seven (Molo-01 to Molo-07) wide spaced holes were drilled on Molo. Six of these (over a strike length of 1.2 km) intersected graphitic mineralisation to a vertical depth of 75m, with down-hole thicknesses of between 60m and 150m in width. Graphite mineralisation intersected in drill core was open along strike, and at depth. Forty-one diamond drill holes, comprising 8,502.7m of diamond drilling was completed on Molo during 2012. During 2014 an additional 32 diamond drill holes, (totalling 2,063m), were completed. With this most recent drill programme, a total of 80 diamond drill holes (Figure 18), (totalling 11,660m) has currently been completed on Molo, and these were used for the mineral resource estimations.

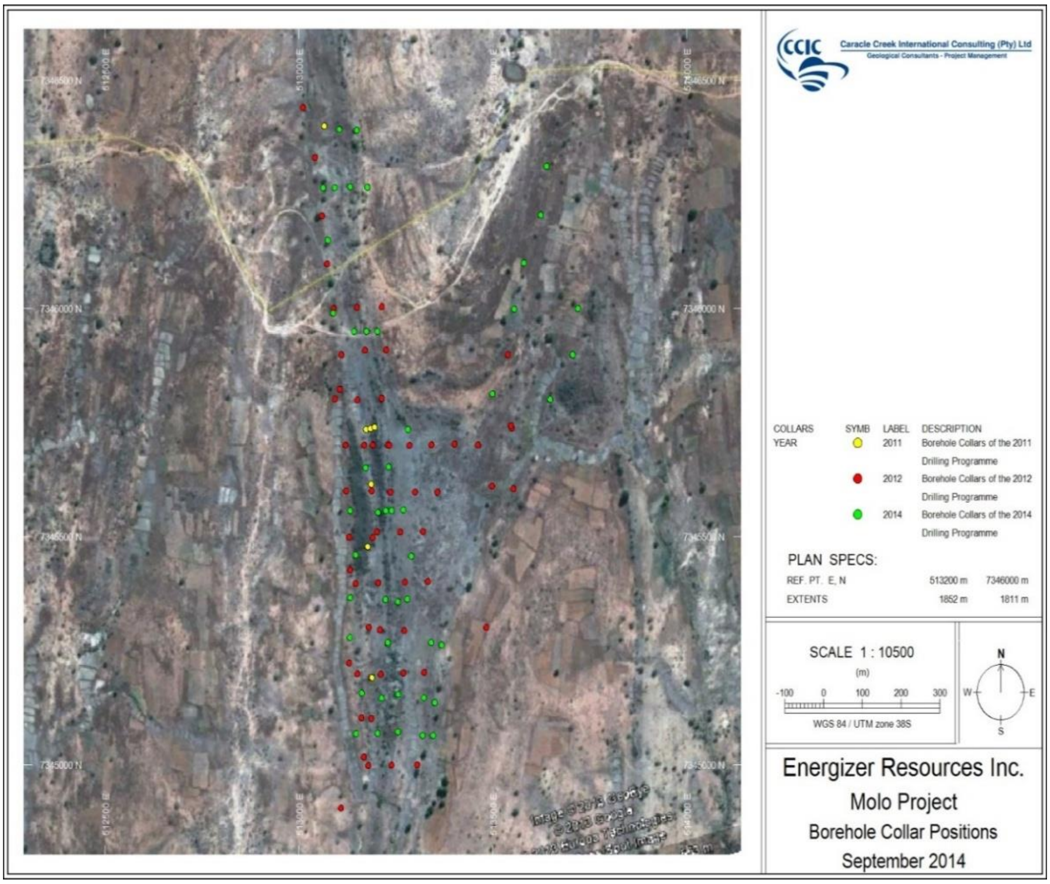


Figure 18: Borehole Collar Positions for All of the Known Boreholes Drilled on the Molo Deposit

10.1 Diamond Drill Contractor and Logistics

Between 2009 and 2014 all diamond drilling on the Green Giant-JV Project was carried out using a Boart Longyear 44 skid-mounted wire-line rig, and a Boart Longyear LF-90 skid-mounted wire-line rig, owned and operated by Boart Longyear™ (http://www.boartlongyear.com/), South Africa. The initial 70m to 100m of any borehole was generally completed with HQ core (63.5 mm diameter), and once reasonably competent rock was encountered, this was reduced to NQ (47.6 mm diameter core). On rare occasions, (as noted for the 2011 metallurgical hole), larger diameter PQ sized core (85.0 mm diameter) was extracted.

The drill moves were completed using Boart Longyear's John Deere skidder equipped with a blade and winch. Drill pads and sumps were prepared using a rental CAT 420D backhoe / loader. While drilling, all fluids were pumped directly from the sumps, with all overflow fluids directed back to the sumps.

These measures were taken to conserve drill fluids and to prevent site contamination by drill additives, or metals liberated by the drilling. Water for drilling operations was trucked from streams, creeks, or ponds sporadically located along the main drainages crossing the property and stored at the drill site until required.

10.2 Core Handling Procedures

Core is delivered from the Molo drill site to the Fotadrevo base camp by pickup truck at the end of every 12 hours shift, under the supervision of the drilling company, or an official designated by the Company. Drill core is stored in galvanized-steel core boxes 1m in length holding 3m PQ, 5m HQ, or 7m NQ core. The core boxes are laid out on constructed core benches in sequential order. A general review of the core is undertaken, and the core is washed, or rinsed of debris and drilling fluids. The Company's technicians then assess the overall condition and recovery of the core and complete a lithological 'quick log'. Errors in run markers are noted. All drill core is presently stored at the Company's Fotadrevo camp within a secure, 20m x 25m fenced enclosure.

10.3 Core Logging

The Company utilizes a logging system developed by Taiga, which has subsequently been altered slightly and has become the standard for all Company procedures. At this stage of exploration there is no restricted list of rock units for core logging as the stratigraphy is still being developed. The Company's geologists, however, attempt to utilize a standard set of units and aim to discuss and agree upon any new units prior to their utilization.

Core logging was previously recorded onto paper logs which were subsequently transferred to computer. The Company has, however, recently purchased Panasonic field laptops and thus all data is currently entered directly into these units. Core logs contain observations of geology, structure, mineralogy, alteration, and sample interval descriptions.

10.4 Core Recovery

Trained technicians are responsible for collecting geotechnical data such as Rock Quality Description ("RQD") and core recovery. The data is recorded onto paper forms with entry into computer logs at the end of the day. The core recoveries are considered as being good and fit for resource estimation purposes.

10.5 Core Photography

The core is photographed in groups of two boxes (Photo 5), and then forwarded for cutting. Core is typically photographed wet.

Photo 5: Core Photographs of Borehole Molo-05 Core (Boxes 17 and 18) as Supplied to CCIC by the Company

10.6 Collar Survey

Borehole collar locations are initially established in the field using a hand-held Global Positioning System ("GPS") instrument. Following completion of the hole the collar locations were re-measured, also using a hand-held GPS. The nominal accuracy of these positions, as stated by the manufacturer of the GPS units, is ±3m. The bore hole collars have not been surveyed by a registered surveyor.

All drill collar sites have been reclaimed and collars marked, with nothing left in the ground, or on the drill site. All holes are plugged and cemented to approximately 1m down the hole. Furthermore, all holes are identified by engraving the collar name into the fresh cement, which when dry is very difficult to destroy (Photo 6).

Photo 6: Concrete Marker Showing the Collar Position of Borehole Molo 12-02 (as Supplied to CCIC by the Company)

10.7 Down Hole Surveys

From 2009 till present, Boart Longyear uses single shot Reflex equipment on all diamond drill holes to measure down the hole azimuths and inclinations. Measurements were taken below the level of the surface casing, (generally 10m to 15m depth), every 50m, (unless hole conditions dictated otherwise), with a final measurement taken at the end of the hole.

10.8 Geotechnical Logging

Geotechnical logging consisted of RQD measurements and core recovery calculations. The data was collected on paper forms and later transcribed into an Excel spreadsheet. RQD measurements are calculated using a minimum 10 cm core length according to the following formula:

$$RQD = Run\ Length - \frac{(\sum Pieces\ of\ core < 10\ cm)}{Run\ Length}$$

10.9 Diamond Drill Core Sampling

The methodology utilized by the Company for diamond drill core sampling was first established by Taiga during the 2008 exploration programme and then modified to accommodate specific programme requirements as needed.

The 2012-2014 diamond drill core sampling procedure may be described as follow:

- Sample interval was set at a maximum of 1.5m (run length), and shortened based on lithological breaks.
- Sample intervals were recorded in the drill log and in pre-printed sample books. QA/QC samples numbers were flagged at this point for later insertion.
- Plastic sample bags were numbered sequentially with the appropriate sample number.
- Core was cut by a technician using a clean water spray table rock saw and both halves of the sawn core was placed back in the box.
- The geologist who logged the core verified the sample tag with the sample book and placed half of the cut core into the sample bag.
- The sample bag was sealed with a zip tie, placed in another larger bag, (i.e., double bagged), with a duplicate sample number, and a sample tag was inserted between the sample bags to mitigate against the destruction of the sample tag.
- All the samples were stored in a secure facility at the Company's Fotadrevo camp site until shipment.

10.10 Diamond Drill Results

All results from the 2011, 2012 and 2014 diamond drill programmes are presented in (Table 15) below. The authors of this Report are of the opinion that there are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the nature of the obtained samples.

Table 15: Collar Co-Ordinates, Drilled Length and Orientation for the Molo Drill Holes

BHID	EASTING	NORTHING	ELEV	LENGTH	DIP	AZIMUTH
MOLO-01	513173	7345735	553.96	166	-45	90
MOLO-02	513184	7345737	554.52	47	-45	265
MOLO-03	513177	7345478	549.88	207.5	-45	105
MOLO-04	513188	7345191	545.79	137	-45	85
MOLO-05	513065	7346400	551.60	131	-45	90
MOLO-06	512199	7345902	543.51	200	-45	90
MOLO-07	513186	7345615	550.84	142.5	-45	90
MOLO-12-01	513121	7345600	550.78	463.6	-45	90
MOLO-12-02	513185	7345601	550.58	89	-45	270
MOLO-12-03	513236	7345598	551.52	311	-45	90

BHID	EASTING	NORTHING	ELEV	LENGTH	DIP	AZIMUTH
MOLO-12-04	513300	7345600	553.24	221	-45	90
MOLO-12-05	513360	7345600	553.96	170	-45	90
MOLO-12-06	513150	7345400	548.59	364.5	-50	90
MOLO-12-07	513210	7345400	547.65	299	-50	90
MOLO-12-08	513270	7345400	547.86	212	-50	90
MOLO-12-09	513330	7345400	549.46	137	-50	90
MOLO-12-10	513151	7345199	545.65	320	-50	90
MOLO-12-11	513210	7345197	545.81	221	-50	90
MOLO-12-12	513270	7345200	544.46	182	-50	90
MOLO-12-13	513322	7345202	543.49	95	-50	90
MOLO-12-14	513177	7344998	541.12	260	-50	90
MOLO-12-15	513238	7345001	541.87	194	-50	90
MOLO-12-16	513305	7344999	543.16	80	-50	90
MOLO-12-17	513092	7345801	553.85	258.4	-50	90
MOLO-12-18	513151	7345799	554.37	132	-50	90
MOLO-12-19	513204	7345805	556.96	77	-50	90
MOLO-12-20	513089	7346000	552.99	234	-50	90
MOLO-12-21	513150	7346002	552.38	156.5	-50	90
MOLO-12-22	513215	7346007	554.77	54	-50	90
MOLO-12-23	513202	7345517	550.20	126.4	-50	270
MOLO-12-24	513245	7345513	550.21	177.5	-50	270
MOLO-12-25	513210	7345292	546.50	85.5	-50	270
MOLO-12-26	513274	7345295	545.81	186.5	-50	270
MOLO-12-27	513226	7345702	554.22	212	-50	270

BHID	EASTING	NORTHING	ELEV	LENGTH	DIP	AZIMUTH
MOLO-12-28	513170	7345909	554.69	84	-50	270
MOLO-12-29	513215	7345907	555.68	143	-50	270
MOLO-12-30	513188	7345713	553.67	126.8	-50	270
MOLO-12-31	513322	7345513	552.23	293	-50	270
MOLO-12-32	513119	7345697	553.26	327.5	-50	90
MOLO-12-33	513169	7345702	553.10	252.5	-50	90
MOLO-12-34	513232	7345702	554.33	236	-50	90
MOLO-12-35	513285	7345700	554.94	185	-50	90
MOLO-12-36	513341	7345702	554.59	200	-50	90
MOLO-12-37	513403	7345703	553.26	200	-50	90
MOLO-12-38	513465	7345706	549.95	152	-50	90
MOLO-12-39	513190	7345498	550.05	329	-70	90
MOLO-12-40	513179	7345301	546.95	326	-70	90
MOLO-12-41	513187	7345102	542.80	329	-70	90

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

At all times during sample collection, storage, and shipment to the laboratory facility, the samples were in the control of the Company, or their agents.

Note that no further sample preparation and analysis were conducted for this report as all the required work was carried out prior to Molo Phase 1.

When sufficient sample material, (grab, trench, or core) has been collected, the samples are trucked, or flown to the Company's storage location in Antananarivo, at all times accompanied by a Company employee. From there, samples are then shipped to either South Africa (Mintek, or Genalysis), or Canada (Activation Labs) for ICP-MS analysis.

Drill core samples collected during 2011 were directed to two major laboratories. All samples collected during Phase I of 2011 were sent to Mintek, South Africa. Samples were then tested for Carbon content, (Total Organic Carbon and overall Carbon content), as well as the full range of elements available through ICP-OES (Mintek code FA5) and XRF analysis. The elements tested included Al, Si, P, S, Cl, K, Ca, Ti, V, Cr, Mn, Fe, Co, Ni, Cu, Zn, Ga, Ge, As Se, Br, Rb, Sr, Y, Zr, Nb, Mo, Ag, Cd, in, Sn, Sb, Te, I, Cs, Ba, La, Ce, Hf, Ta, W, Hg, Tl, Pb, Bi, Th and U.

The remainder of samples collected during Phase 2 of the 2011 exploration programme were submitted for analysis to Actlabs, Canada. Samples were again submitted for analysis of Carbon content, as well as for a large range of elemental analysis.

During 2012, all samples were submitted to Intertek Genalysis, (www.intertek.com/) ("Genalysis"). All work undertaken by Intertek is performed in accordance with the Intertek Minerals Standard Terms and Conditions of which can be downloaded from their web page.

All analytical results were e-mailed directly by both Genalysis and Mintek to the Green Giant Project Manager, as well as the Company's executive staff, and were posted on a secure website and downloaded by the Company's personnel using a secure username and password. Following the site inspection in May 2012, all analytical results were also e-mailed directly to Dr. Hancox (CCIC) and these were compared against the final data set as presented by the Company.

All the laboratories that carried out the sampling and analytical work are independent of the Company.

The authors are of the opinion that following on the validation and verification checks, the sample preperation and analysis were adequate.

11.1 QA/QC

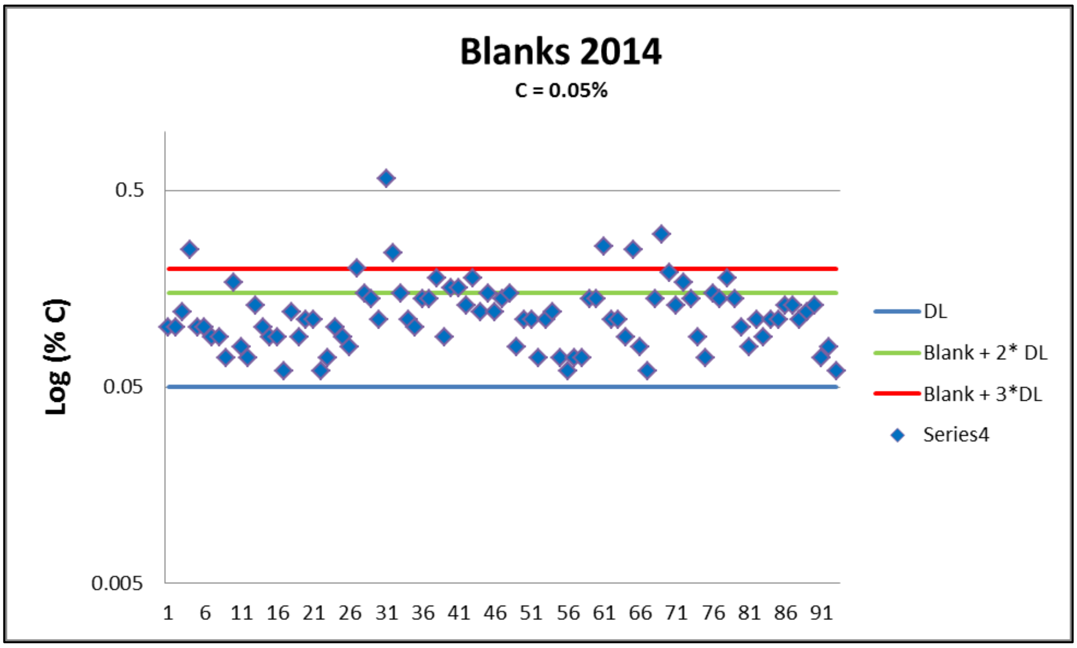
To carry out QA/QC protocols on the assays, blanks, standards and duplicates were inserted into the sample streams. This was done once in every 30 samples, representing an insertion rate of 3.33% of the total.

11.2 Blanks

Since the 2009 drill programme, the Company has rigorously implemented a blank protocol. For Molo fine grained quartz sand sourced from a hardware store in Antananarivo was used as the blank material for the sampling campaign.

An additional 93 blanks have been submitted during this campaign, taking the total number of blanks to 301. A detection limit of 0.05% Carbon was used for the purpose of this exercise.

To verify the reliability of the blank samples, the detection limit and the blank + 2, and 3 times the detection limit was plotted against the date and Graph 5 represents the previous and current campaigns respectively). Blanks for the 2014 campaign reflects the majority of samples have concentrations that lie within the blank + 3 times detection limit threshold, with the maximum outlier being 0.57% C.



Graph 5: Plot of Log % C Versus the Date of the Analysis 2014 for the Blanks

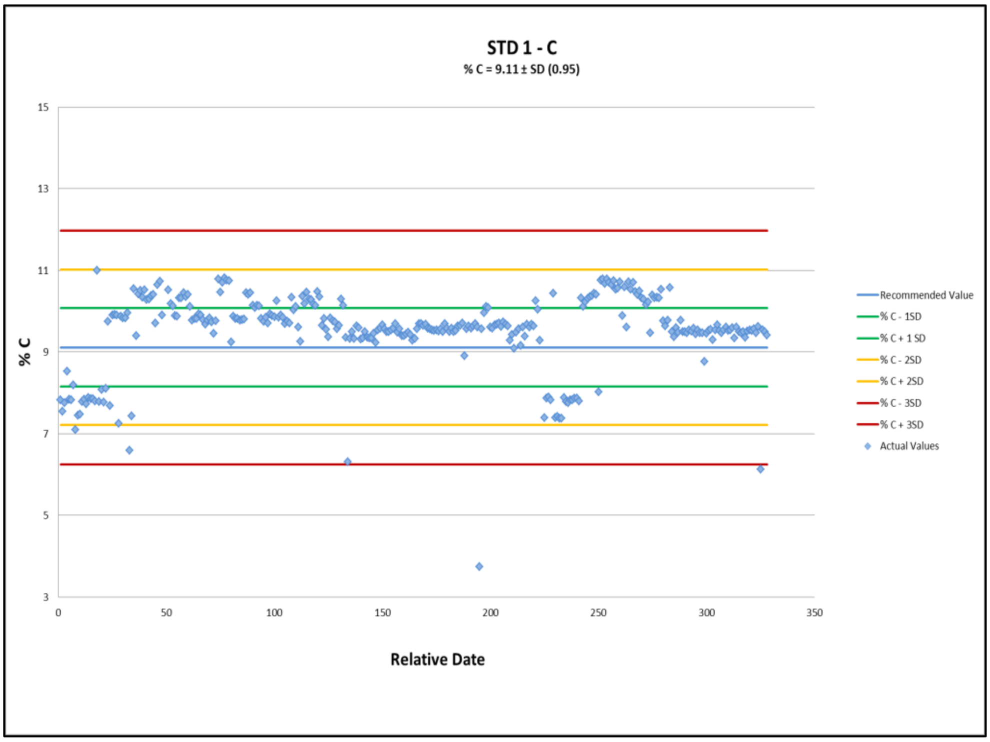
11.3 Standards

Because of the difficulty in sourcing CRM during the 2012 campaign, the Company commissioned Actlabs, Canada to create a CRM from the remaining Molo drill core pulps from the 2011 programme. As certified the Actlabs standard (STD 1 C), has a recommended value of 9.11% Carbon. For the 2014 campaign, the Company sourced two additional CRM's from GEOSTATS (Proprietary) Limited ("GEOSTATS"), namely GGC-01 and GGC-07. The recommended values for GGC-01 and GGC-07 are 24.97% C and 0.56% C respectively.

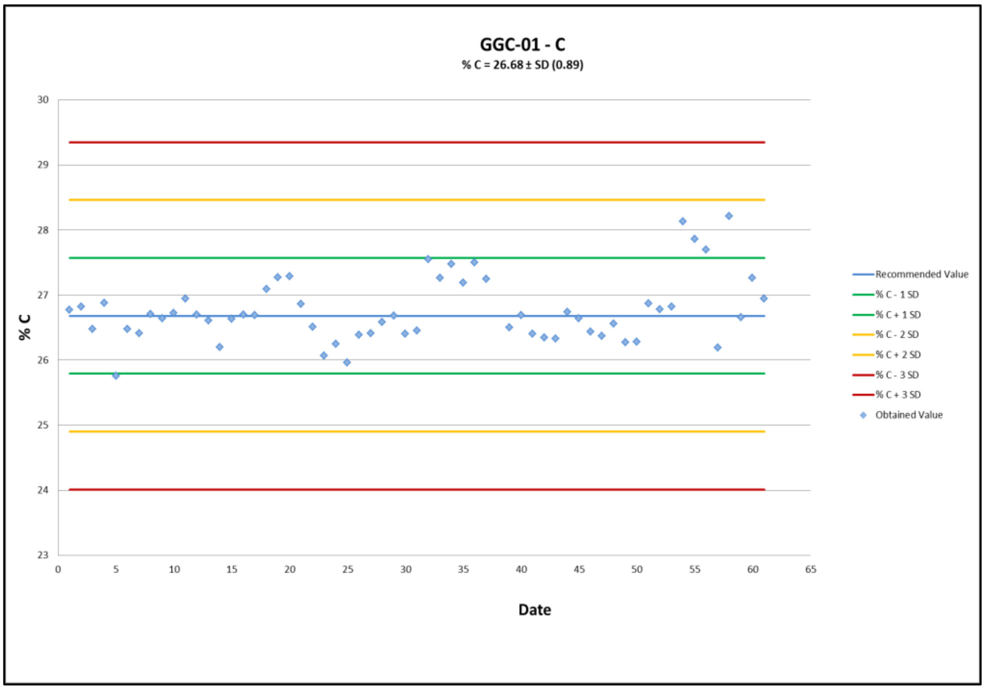
To check the reliability of the standard, a plot of the recommended CRM value versus date was created (Graph 6, Graph 7 and Graph 8) below.

The upper and lower limits of one, two and three times the standard deviations of the recommended value are also included in the plot. For STD 1 C, all but two results fall outside the acceptable limit of three times the standard deviation.

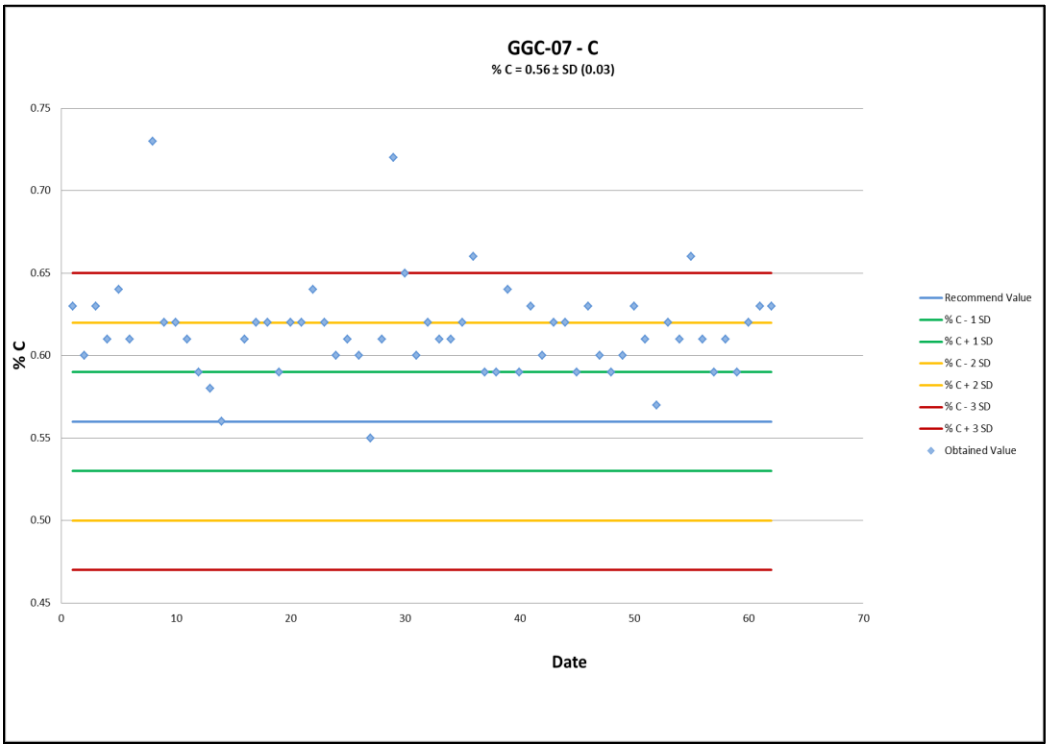
It is, however, worth noting that the obtained results indicated a slight positive bias when compared to the recommended value. For GGC-01, all the obtained results occur within two standard deviations. For GGC-07, four samples occur outside of three standard deviations. This CRM has a mean value of 0.56% C whilst the mean of the obtained values is 0.61% C, indicating a positive bias at low concentrations.



Graph 6: Showing Carbon Concentration as Analysed in Std 1C



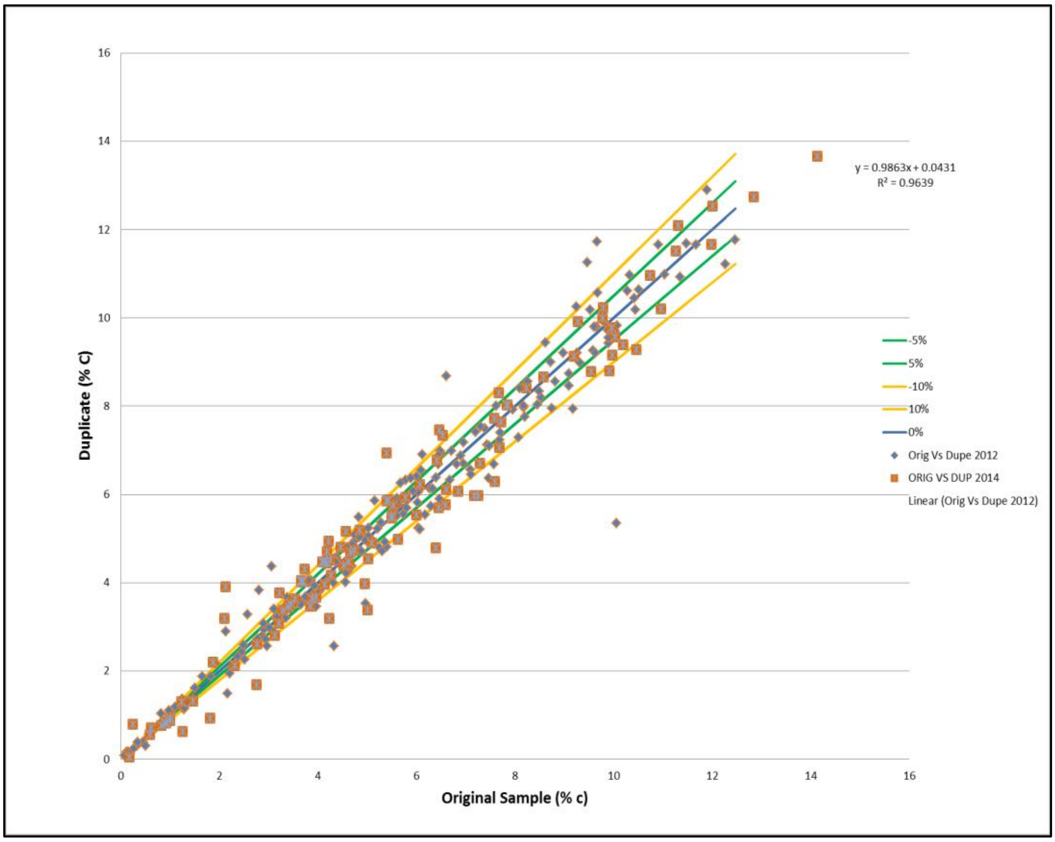
Graph 7: Showing Carbon Concentration as Analysed in GGC-01



Graph 8: Showing Carbon Concentration as Analysed In GGC-07

11.4 Field Duplicates

A total of 254 duplicate samples were submitted. To check how close these were to the original samples, a plot of the original samples with a zero, five, and ten per cent difference of the original samples was created. (Graph 19) 2012 and 2014 samples are shaded in blue and red respectively. Most of the samples are within the 10% difference limit. The plot also shows a good correlation between the original value and the duplicate, as is evident from the regression line with an R2 value of 0.96.



Graph 19: Original (Orig) Versus Duplicate (Dupe) Plots

11.5 Relative Density

No additional samples were collected for density measurement during the 2014 drill programme. A total of 226 relative density measurements were contained in the database presented to CCIC for Molo.

The process to measure the relative density was as follows:

- Pieces of whole core were collected, and the rock types documented.
- The selected section of core was then dipped into wet paraffin wax and allowed to dry. This sealed the core to avoid the absorption of moisture.
- The pieces of core were weighed dry, followed by weighing in a water bath.

- All data was collected on paper forms and transferred to a spreadsheet for future calculations.

11.6 Data Verification

Prior to CCIC's involvement with the Company and the Project, all information published regarding the 2011 exploration programme was reviewed by an independent Qualified Person as it became available.

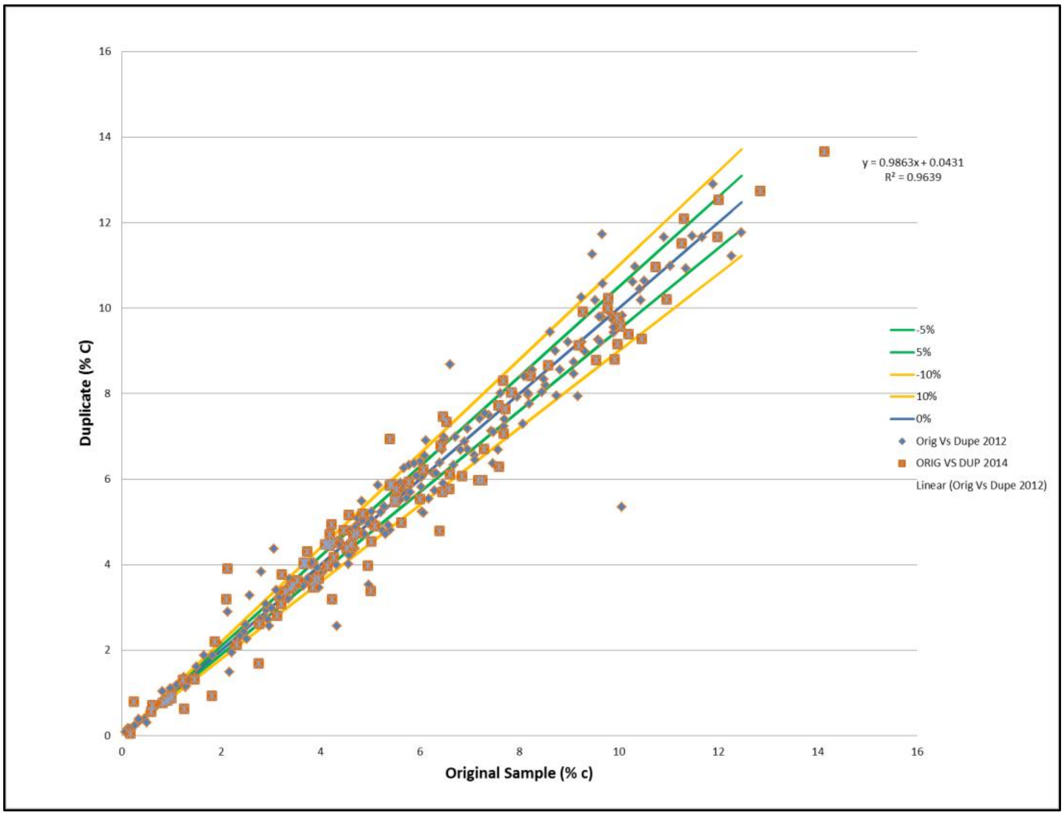
The database received by CCIC from Energizer, (now NextSource; "the Company"), contained 80 drill holes totalling 11,660m and data from 35 trenches totalling 8,492m.

With regards to the database, CCIC performed various tests to verify the integrity of the collar co-ordinates, logging and sampling procedures, and assay results. Leapfrog™ Geo software (www.leapfrog3d.com/) ("Leapfrog") was used for most of the checks.

11.7 Collar and Down Hole Surveys

During a site visit in 2014, Desmond Subramani randomly selected four drill hole collars to validate. All four drill hole collars were physically located and plotted within the accuracy of the handheld GPS unit being used for validation. While on site, the Company was in the process of undertaking a topographic survey and a DGPS re-survey of all drill hole collars.

To verify the correct position of the re-surveyed drill hole collars with respect to their elevation, collar co-ordinates were plotted against the surface re surveyed topography of the area (Graph 20). The results showed that the re-surveyed collars were within a 25 cm of the surface topography, and therefore, all collar co-ordinates were deemed to be correct and were used for the geological modelling.



Graph 20: East-West Cross-Section Showing the Collar Positions with Respect to Topography

12 DATA VERIFICATION

12.1 Drill Logs

During the initial 2012 site visit Dr. Hancox randomly selected two of the 2011 drill holes (Molo-07 and Fotsy-06) to review the log's data against the drill core (Photo 7). The holes were check logged to verify that the intervals in the logs matched the drill core. No discrepancies were observed. Molo-04 and Molo-05 was also examined.

Photo 7: Borehole Fotsy-06 - One of the Bore Holes Check Logged during May 2012 Site Visit (as Supplied to CCIC by the Company)

For the 2014 drilling campaign, a CCIC Geologist (Mr W. Ngangolo) supervised the drilling, logging and sampling. Drill hole logs were checked in the field, prior to uploading into the Company's Database, managed by Eric Steffler. Dr. Schneiderhan also undertook various check logs during the 2014 site visit.

Database checks undertaken in the Leapfrog™ software ("Leapfrog") included, (but were not limited to), gaps consistency in the logging codes, and overlaps in the depth 'From' and 'To' entries. No gaps were encountered in the database.

There seemed to be some lack of consistency in the sage of the logging codes as initially the graphite bearing unit was termed as either gneiss (Gn) coding, or graphitic gneiss, (coded as GfGn). This was later edited by the Company and the graphite bearing gneiss was finally coded GpGn.

Whilst a few irregularities were encountered with the data supplied, the authors are of the opinion that following on the validation and verification checks, the data is adequate for the level of geological modelling and resource estimation undertaken.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A series of metallurgical and mineralogical investigations was completed at Mintek in South Africa and SGS Lakefield in Canada on samples originating from the Molo deposit amongst others, which included Fotsy and Fondrana deposit material in the case of the Mintek test work. The primary purpose of the programs from the perspective of the Molo material was to:

- Develop a robust process flowsheet that produces a combined graphite concentrate grading of at least 95% total carbon.
- Demonstrate the robustness of the proposed flowsheet for the expected mill feed during the first few years of operation.
- Generate concentrate and tailings samples for downstream evaluation by potential vendors and off-take partners.

The first scoping level program was completed at Mintek in 2012. This was followed up with a flowsheet development program at SGS Lakefield in June to August 2013. The proposed process flowsheet was employed in a pilot plant campaign, again conducted at SGS Lakefield, in September / October 2013 to confirm the robustness of the flowsheet and to generate concentrate for downstream testing.

A process optimization program was conducted at SGS Lakefield between June and September 2014 to simplify parts of the circuit. This optimized process flowsheet was validated in a concluding variability flotation program between October 2014 and January 2015. All process flowsheet optimization was conducted under the assumption that the average ore mined per annum would be 856,071t at a head grade of 7.04% C. The resulting flow sheet was subsequently tested for higher head grades and is, therefore, deemed suitable for both Phase 1 and the Expansion project development.

The key economic factors for a graphite Project from a metallurgical point of view are graphite recovery, flake size distribution and concentrate grade.

While there is a market for graphite concentrates grading as low as 80% carbon, the price of product increases with carbon grade. For a flotation graphite concentrate without further purification a product grading between 94% and 97% total carbon is typically targeted.

Large graphite flakes demand higher prices due to a limited supply on the market, while concentrates containing graphite flakes smaller than 200 mesh (-75 microns) are available in abundance and, therefore, create a much lower revenue on a per tonne of concentrate basis. Consequently, a process development program must focus on flake size preservation to maximize the amount of medium and large flakes in the concentrate, while minimizing the percentage of small flakes.

It is pertinent that the decisions made in the process development programs consider these economic factors. While not all of them can be optimized at the same time, a balanced approach is required.

13.2 Flowsheet Development – Mintek

A scoping level metallurgical and mineralogical program was completed on five composites at Mintek in South Africa in 2012. These composites originated from the Fondrana, Fotsy, and Molo zones. Only the results for the Molo composite are summarized in this section, which was specified as trench sample Molo-TH-11-01.

The Molo composite graded 10.6% carbon. Three rougher kinetics tests were completed on the composite with a primary grind size of P50 of 150 microns. A carbon recovery of almost 99% was achieved on average with rougher tailings grades of 0.1% to 0.2% carbon. The rougher concentrate grades yielded 53.5% to 59.0% carbon. The reagents used in the program were illuminating paraffin as the graphite collector and Dowfroth 200 as the frother.

Two cleaner tests using conditions that were previously established for Fotsy and Fondrana composites produced concentrate grades of 77.4% carbon with the dispersant sodium silicate and 77.5% carbon without sodium silicate at carbon recoveries of 95.9% to 97.5%.

A final cleaner flotation test was conducted using the flowsheet depicted in Figure 21. The second regrind was conducted in a pebble mill with smaller media with a size range of 4 mm to 10 mm instead of the regular pebble mill containing +20 mm gravel. The concentrate grade improved to 82.7% carbon at 92.9% carbon recovery because of this secondary regrinding and cleaning circuit.

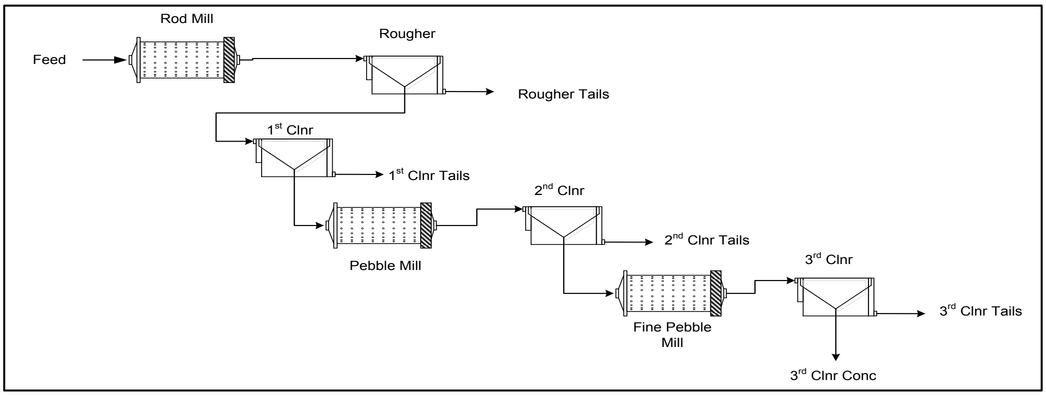


Figure 21: Flowsheet for Molo Mineralization – Mintek

It should be noted that the Mintek test work program focused on carbon recovery optimization and did not focus on flake size preservation.

13.3 Flowsheet Development – SGS

A full flowsheet development program was subsequently initiated at the SGS Lakefield site in June 2013 using a high grade and a low grade composite from the Molo deposit. These composites were generated by collecting a sub-sample from the 200t bulk trench sample that was generated for pilot plant testing. For every 2m of the total trench length of 160m, approximately 2.5t of material were extracted for piloting. A 5 kg sub-sample was then removed from each 2.5t sample to form the high grade and low grade composite for laboratory scale testing.

A series of eight rougher and cleaner flotation kinetics tests evaluated the flotation performance of each of the two composites, as well as a 50:50 blend of the two composites.

The flotation approach employed a flash flotation stage on test charges that were stage crushed to minus 6 mesh followed by primary grinding and rougher flotation. The cleaner tests employed a primary polishing grind followed by three stages of cleaner flotation.

Since the blended composite produced good metallurgical results, a decision was made to use this composite for all further development work as this maximizes the mineral resource of the deposit and simplifies mining.

The following eight rougher kinetics tests evaluated primary grinding with conventional steel rods and ceramic media:

- The energy input created by the ceramic media proved insufficient in a primary grinding application and, therefore, steel media was chosen with grind times targeting a mill discharge of P_{80} = 400 to 500 microns.
- Four proceeding cleaner tests investigated the conditions of the primary cleaning circuit.
- The flash and rougher concentrates were combined and subjected to a polishing grind using ceramic rods.
- The polishing times were varied between 7 minutes and 30 minutes.
- The polishing mill discharge was upgraded in three stages of cleaner flotation and the third cleaner concentrate was subjected to a size fraction analysis.
- A primary polishing time of 22 minutes was identified as the best compromise between maximizing the intermediate concentrate grade and minimizing flake degradation.
- Considering the liberation properties of the different flake sizes, the intermediate concentrate was classified at 80 mesh (177 microns), and 150 mesh (106 microns), and each classification product was then subjected to a secondary polishing grind and cleaner flotation.
- The remaining four tests F22 to F25 in the flowsheet development program investigated the impact of varying secondary polishing times on the combined concentrate grade. Secondary polishing times of 6 minutes, 8 minutes, and 45 minutes for the +80 mesh, +150 mesh, and -150 mesh size fractions, respectively, achieved the best metallurgical results.

The proposed flowsheet, including classification sizes and polishing grind times used in test F24 is depicted in Figure 22 and the size fraction analysis results of the combined concentrate are presented in Table 16. The conditions of test F24 were selected as they constituted the best compromise between maximizing concentrate grades while minimizing graphite flake degradation. All size fractions yielded concentrate grades of at least 95.2% total carbon including the smallest size fraction of -400 mesh (-37 microns).

The mass recovery into the large flake category of +80 mesh (177 microns), was very good at 47.6% and only 18.6% of the mass reported to the fines smaller than 200 mesh (74 microns).

The carbon recovery into the combined concentrate was 83.4% in this open circuit test. Graphite recovery is expected to increase during closed circuit operation as the intermediate streams are circulated rather than treated as tailings.

The significant differences between the metallurgical results obtained in the Mintek and SGS programs were the result of several factors. Firstly, the two Molo composites were collected from different areas of the deposit. Secondly, the Mintek approach focused on maximizing carbon recovery into the graphite concentrate. In contrast, the SGS approach focused on maximizing flake size preservation and concentrate grade.

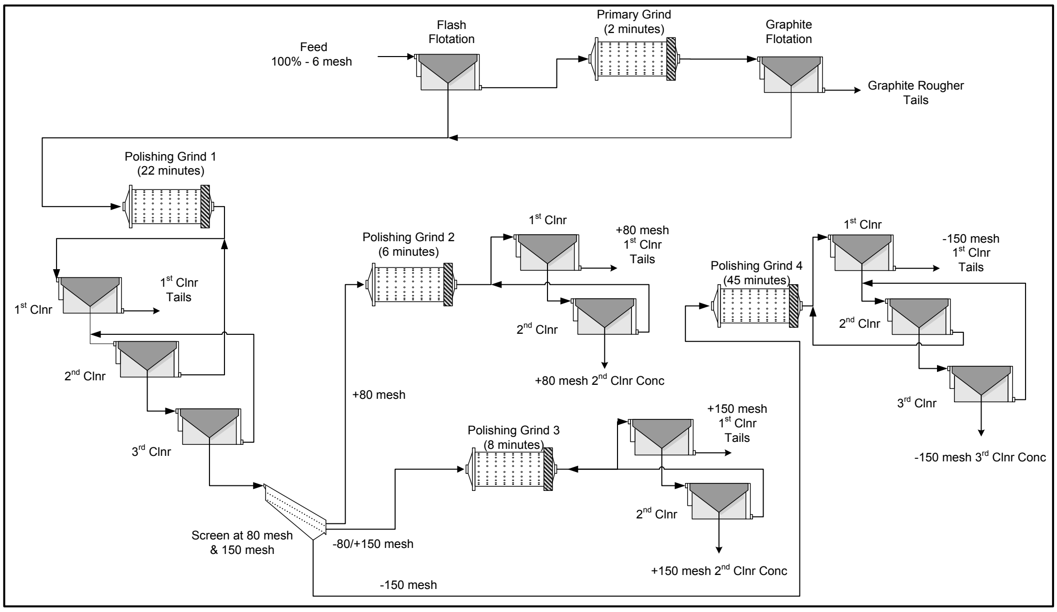


Figure 22: Proposed Molo Mineralization Flowsheet – SGS

Table 16: Size Fractions Analysis Results of Test F24

Product		Mass (%)	Grade (% Total Carbon)
Mesh	**Microns**		
+48	+297	21.7	97.4
-48/+65	-297/+210	17.9	96.7
-65/+80	-210/+177	8.1	95.7
-80/+100	-177/+149	10.9	96.0
-100/+150	-149/+106	12.5	95.2
-150/+200	-106/+74	10.3	95.3
-200/+325	-74/+44	8.5	96.2
-325/+400	-44/+37	3.0	95.9
-400	-37	7.1	95.7

13.4 Pilot Plant Campaign

A pilot plant campaign using a 200 t Molo bulk (trench) sample was conducted at the SGS Lakefield, Canada site in September / October 2013.

The pilot plant campaign was carried out to confirm the robustness of the above SGS proposed flowsheet that was developed in the laboratory program. Further, approximately 10.8t of graphite concentrate were generated for downstream testing, including vendor testing and evaluation of potential off-takers.

The bulk sample that was processed in the pilot plant was collected on site by extracting two samples of approximately 100t each, one of the low grade area and one of the high grade area of the Molo deposit. The aim for the two bulk samples was to be representative of the future plant feed.

The position of the bulk sampling trench is depicted in Figure 23. The first 80m starting from the west were classified as high grade material and the following 80m as low grade material. The trench sections were sub-divided into 2m intervals and approximately 2,500 kg of ore were extracted from each interval after removal of any soil and overburden. At the same time, approximately 5 kg was removed from each interval to generate the material for the SGS laboratory scale flowsheet development program described above.

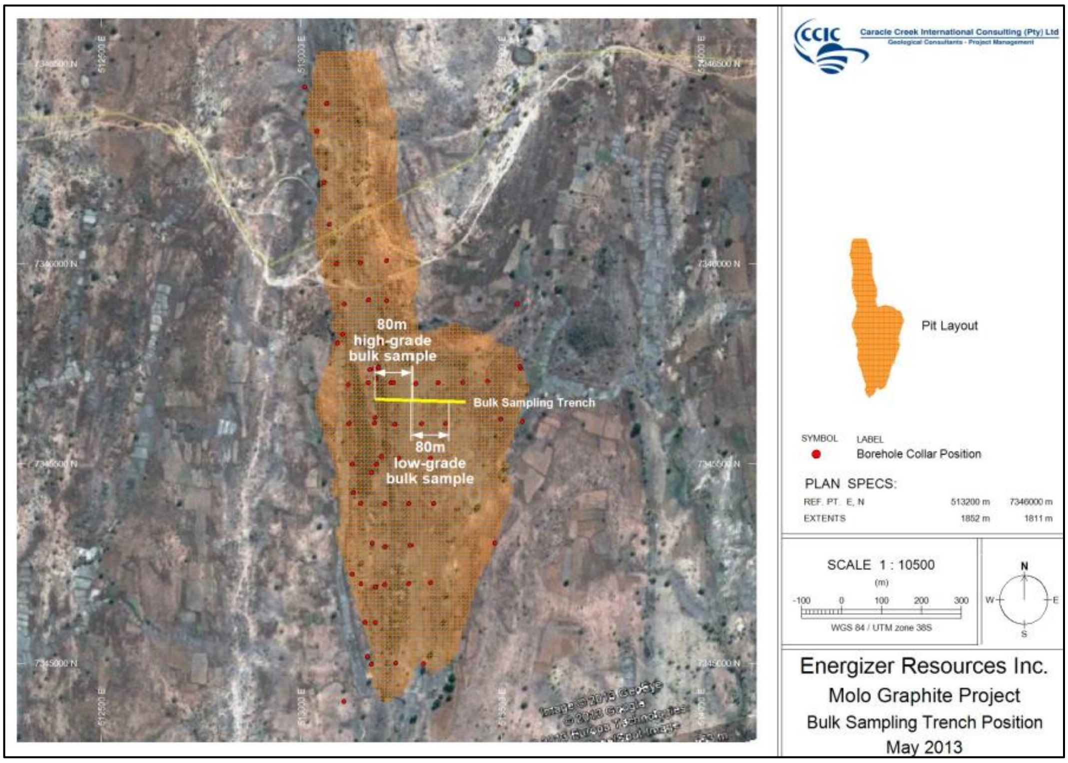


Figure 23: Map Showing Position of Bulk Sampling Trench

A single composite was processed in the pilot plant campaign grading 7.98% total carbon. The pilot plant composite was generated by blending high grade and low-grade ore in a ratio of approximately 46:54.

The results of comminution tests that were carried out on the individual composites, as well as the pilot plant blend are summarized in Table 17.

Table 17: Summary of Comminution Test Results for Pilot Plant Composites

Sample	Relative	JK Parameters			RWI	BWI	AI
Name	Density	A x b[1]	A x b[2]	T_a	(kWh/t)	(kWh/t)	(g)
Blend	2.29	199	0	2.28	8.3	11.2	0.129
High Grade	2.08	-	151	1.88	8.7	11.4	0.125
Low Grade	2.37	-	192	2.09	6.9	9.7	0.106

[1] A x b from DWT

[2] A x b from SMC

A total of nineteen pilot plant runs, PP-01 to PP-19, were completed to process the approximately 200 t of the Molo bulk composite using the flowsheet depicted in Figure 24. The only reagents that were employed were fuel oil number 2 (diesel collector), and methyl isobutyl carbinol (MIBC frother) at average dosages of 117 g/t and 195 g/t, respectively.

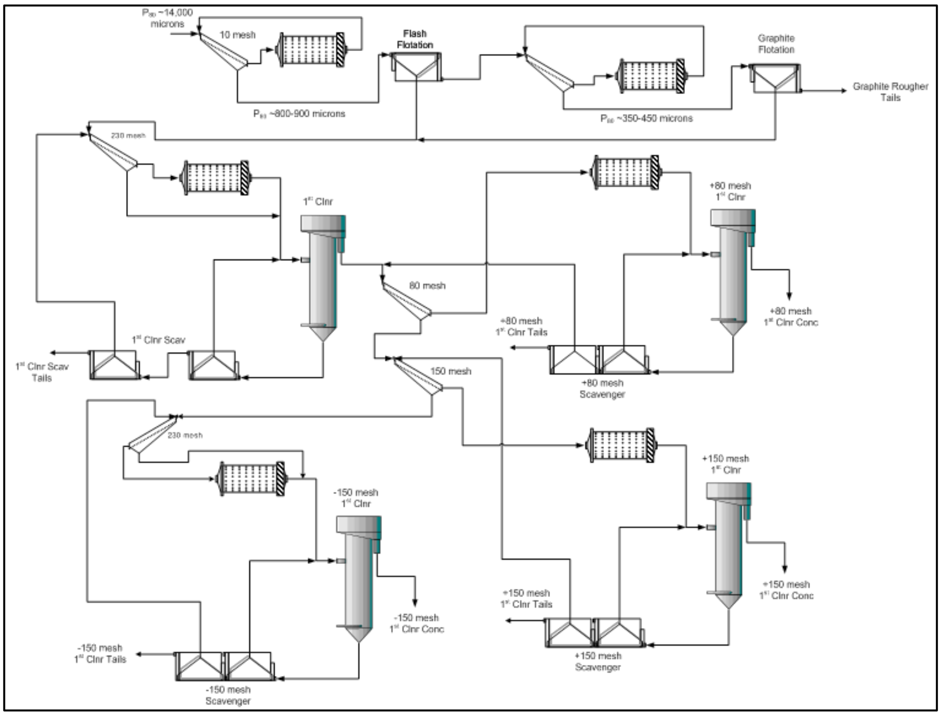


Figure 24: Molo Pilot Plant Flowsheet

A total 15 circuit surveys were completed between PP-04 and PP-18. In each survey, up to 36 streams were sampled 5 times over the course of an hour and then submitted for sizing and/or chemical analysis.

All products submitted for chemical analysis have been assayed for total carbon and the low grade tailings streams, as well as for graphitic carbon.

The results were then used to generate full circuit mass balances using the BILMATTM data reconciliation software.

In addition, hourly grab samples of strategic streams were collected and submitted for chemical analysis, or sizing.

Turn-around times of sizing and assay results were typically less than one hour. This approach was chosen to ensure that metallurgical targets have been met and to facilitate the optimization of the operating conditions.

The average feed sizes to the flash and rougher flotation circuits were P_{80} = 835 microns and P_{80} = 443 microns, respectively.

The results from pilot plant test PP-17B were selected to generate the process design criteria. The combined concentrate grade yielded 93.1% total carbon at a carbon recovery of 90.6%.

A summary of the mass balance is presented in Table 18.

Table 18: Summary of Mass Balance for Pilot Plant Test PP-17B

Product	Mass (%)	Assay % Total Carbon	Distribution % Total Carbon
+80 mesh Concentrate	4.0	95.9	53.1
+150 mesh Concentrate	1.7	92.7	21.9
-150 mesh Concentrate	1.3	85.1	15.7
Combined Concentrate	7.0	93.1	90.6
Combined Tailings	93.0	0.72	9.4
Plant Feed	100	7.17	100

The product size of the final graphite concentrate from the fifteen surveys yielded an average value of P_{80} = 268 microns.

The average size distribution and total carbon grade of each size fraction are presented in Table 19.

Table 19: Average Combined Concentrate from Fifteen Pilot Plant Surveys

Size Mesh	Size Microns	Mass as Percentage of Total Concentrate Mass in %	Grade % Total Carbon
+48	+297	15.7	97.7
+65	+210	17.6	97.4
+80	+177	10.2	96.7
+100	+149	9.7	96.4
+150	+105	15.0	96.1
+200	+74	10.1	95.2
-200	-74	21.6	88.2

The metallurgical results from the laboratory scale flowsheet development program on the 50:50 high-grade / low-grade composite, as well as the results from the laboratory and pilot scale testing of the actual pilot plant composite are summarized in Table 20.

While there were some differences in the results, the data from the three test phases correlated well overall.

The largest difference was the carbon grade of the -150 mesh size fraction in the pilot plant, which was approximately 5% lower compared to the two lab results. It was postulated that this was likely the result of poor polishing efficiency due to dewatering difficulties with the -150 mesh cleaning circuit feed.

Table 20: Comparison of Laboratory and Pilot Plant Metallurgical Results

Product	Laboratory Scale		Pilot Plant
	Master Composite (F24)	Pilot Plant Composite	Pilot Plant Composite
% Mass > 80 mesh	47.6	42.4	43.5
% C(t) > 80 mesh	96.8	96.8	97.4
% Mass -80/+150 mesh	23.5	25.3	24.7
% C(t) -80/+150 mesh	95.6	95.6	96.3
% Mass -150 mesh	28.9	32.3	31.8
% C(t) – 150 mesh	95.7	95.7	90.4

13.5 Optimization and Metallurgical Variability Program

The flowsheet that was employed in the pilot plant campaign provided a fair degree of flexibility to operate the circuit on a relatively ad-hoc basis to meet specific, prevailing market demands in terms of product quality. However, with a focus on reducing capital and operating costs, as well as improving the ease of operability of the plant, an optimization program was initiated to evaluate the possibility of simplifying the process flowsheet while maintaining the graphite concentrate quality.

In May 2014, a review of the metallurgical work completed by Mintek and SGS was accordingly conducted by representatives of DRA and the Company, in conjunction with the author of this chapter.

The first process option identified consisted of the original front end of the original flowsheet with flash and rougher flotation stages. The combined flash and rougher concentrates are subjected to a polishing grind followed by a cleaner flotation circuit.

The intermediate cleaner concentrate is classified on a screen and the screen oversize constitutes a final graphite concentrate. This necessitates that the primary polishing and cleaner flotation conditions can produce a flotation concentrate grading at least 95% total carbon in the larger size fractions.

The screen undersize comprising of below target concentrate is subjected to a secondary cleaning circuit with a polishing mill and cleaner flotation. The cleaner concentrate from this circuit and the screen oversize then constitutes the final combined graphite concentrate.

The second process option was further simplified by eliminating the graphite rougher flotation circuit by incorporating flash flotation only. The flash flotation concentrate is then subjected to a polishing grind and cleaner flotation.

The graphite concentrate generated in this primary cleaning circuit constitutes the final concentrate. This highly simplified process option is based on the postulation that the degree of liberation in the flash concentrate is superior to a rougher concentrate as it contains mostly large graphite flakes that generally are more easily liberated and upgraded in the primary cleaning circuit. By eliminating the rougher circuit, the middlings of graphite and gangue minerals in the primary cleaning circuit feed are reduced significantly, which requires more mechanical manipulation to improve mineral liberation.

The increased graphite losses to the flash tailings associated with the simplified flowsheet may be offset by reduced capital and operating costs and a superior graphite concentrate in terms of flake size distribution.

13.5.1 Sample Selection

Based on the sample locations and logs provided by CCIC, DRA in conjunction with geologists from CCIC selected quarter core samples from various locations and depths within the Molo deposit for optimization and variability test work. Twenty (20) drill core samples were selected based on pit location, depth and indicated grade. The samples that were selected are summarized in Table 21 below.

Table 21: Sample for Optimization and Variability Testing

Description	Drill Core Identification
Year 1-5 High Grade Material	Molo 13, 14, 15, 16, 17, 18, 34, 35 and 36
Year 1-5 Low Grade Material	Molo 45, 46, 47 and 48
Year 5+ North Pit High Grade Material	Molo 37, 38 and 39
Year 5+ South Pit High Grade Material	Molo 29, 30, 31 and 32

As part of the optimization program six comminution composites were generated using drill core from different depths within the Molo 2015 FS 5-year pit layout. Four comminution composites were tested at SGS and 2 additional composites were shipped to Mintek in South Africa for testing. The make-up of the 6 comminution composites is shown in Table 22.

Table 22: Comminution Composites – SGS and Mintek

Lab	Description	Depth (m)	Drill Core Identification
SGS	Comminution Composite #1	14-28	Molo 16, 17, and 18
SGS	Comminution Composite #2	57 – 85	Molo 46
SGS	Comminution Composite #3	14 -28	Molo 34 and 35
SGS	Comminution Composite #4	0-14	Molo 29, 30 and 32
Mintek	Grindmill Shallow Composite	0-14	Molo 15, 18, 35, 36, and 39
Mintek	Grindmill Deep Composite	28-56	Molo 15, 29, 37, and 45

13.5.2 *Comminution Testing*

A summary of the results of the comminution tests conducted at SGS is presented in Table 23 and reveals that the ore is typically very soft, or soft with low abrasivity. Only the Bond ball mill grindability tests at the smaller screen size of 212 microns produced indices that placed the ore into the medium hard category.

Table 23: Summary of Comminution Tests – SGS

Sample Name & Depth of the Drill Hole Intervals	Relative Density	JK Parameters		RWI (kWh/t)	BWI (kWh/t) 500 µm	BWI (kWh/t) 212 µm	AI (g)
		Axb	Ta				
Comminution Composite #1 – Medium	2.25	147.6	1.70	9.3	12.1	14.4	0.097
Comminution Composite #2 – Deep	2.36	157.8	1.73	7.1	8.8	12.9	1.116
Comminution Composite #3 – Medium	2.23	126.0	1.46	9.3	12.1	13.2	0.081
Comminution Composite #4 – Shallow	2.29	299.9	3.38	6.0	8.0	12.1	0.030

Two composites containing drill core from the shallow and the deep areas of the Molo 2015 FS. A 5 year pit layout was shipped to Mintek in South Africa for further Bond ball and rod mill tests, as well as Grindmill tests. The results of these comminution tests are summarized in Table 24. The Bond rod and ball mill test results are consistent with the results obtained for the four composites tested at SGS Lakefield.

Table 24: Summary of Comminution Test - Mintek 2014

Sample Name	Grindmill Feed Top Size -6.7 mm		Grindmill Feed Top Size -9.5 mm		RWI (kWh/t)	BWI (kWh/t) 500 µm	BWI (kWh/t) 212 µm
	% Passing 425 µm at 3 kWh/t	% Passing 150 µm at 10 kWh/t	%Passing 425 µm at 3 kWh/t	% Passing 150 µm at 10 kWh/t			
MOLO 0-14	89.93	69.81	83.95	70.41	7.2	9.3	13.0
MOLO 28-56	77.43	65.38	76.68	68.05	9.2	10.9	13.7

13.5.3 Optimization Flotation Program

A series of six optimization composites were generated to evaluate the two alternative flowsheet options. The primary two composites were 50:50 blends of high grade and low grade mineralization from the shallow section (0m to 14m depth), and the deep section (28m to 56m depth) of the deposit.

Two drill holes of each the high grade (HG) and the low grade (LG) mineralization was selected to generate these two composites. A rougher kinetics test was completed on the two composites and the results revealed that the composite from the shallow section produced an inferior metallurgical response.

Since the process plant must be able to treat all ore within the pit layout, a decision was made to proceed with the majority of the optimization program using the more challenging shallow composite.

A series of rougher kinetics tests on the 50:50 LG:HG shallow and deep (F3 to F12) composites was completed to establish the primary grind size required to achieve a combined flash and rougher carbon recovery of 94% to 95%. This grind size was established at P_{80} = 400 to 450 microns.

A series of four open circuit cleaner flotation tests evaluated the possibility of obtaining a flotation concentrate grading at least 94% total carbon with a single polishing and cleaning circuit.

The four tests, (F13 to F16), included a flash flotation circuit only followed by polishing and cleaning to determine if the improved liberation properties of the flash concentrate would result in reduced polishing and cleaning requirements. Even at the longest polishing time tested, the combined concentrate graded only 85.1% carbon.

Consequently, the simplest proposed flowsheet consisting of flash flotation, polishing, and a single cleaning circuit proved insufficient to produce target concentrate grades even if lower graphite recoveries were accepted by employing flash flotation only.

Three open circuit cleaner tests, (F17 to F19), with flash and rougher flotation, polishing grind and cleaner flotation evaluated the polishing time necessary to achieve satisfactory concentrate grades in the coarser size fractions, which tend to display improved liberation properties. A polishing time of 30 minutes proved sufficient to generate an intermediate flotation concentrate that yielded grades more than 95% total carbon in the size fractions larger than 48 mesh (297 microns).

The intermediate cleaner concentrate was classified on a 48 mesh screen and the screen oversize constituted a final concentrate. The screen undersize was subjected to different polishing times followed by secondary cleaning in two cleaner flotation tests (F20 and F21). Both tests failed to produce a final concentrate grade of at least 95% total carbon.

In order to evaluate whether these metallurgical properties were created by an individual sub-sample of the optimization composite, each of the four drill hole intervals included in the optimization composite was subjected to a batch cleaner test using the conditions of test F21, which employed a secondary polishing time of 60 minutes.

The two low grade composites from drill holes Molo-45 and Molo-47 produced high combined concentrate grades of 97.4% total carbon and 97.7% total carbon, respectively.

In contrast, the two high grade composites from drill holes Molo-16 and Molo-35 yielded lower combined concentrate grades of 83.6% total carbon and 89.0% total carbon, respectively.

To evaluate the variation of metallurgical results for the different composites, a decision was made to proceed with variability flotation testing to develop a better understanding of the metallurgical processing properties.

13.6 Variability Flotation – Phase I

A variability flotation program was carried out in October 2014, using 20 different drill hole composites, which are specified in Table 25.

Table 25: Variability Composites – Phase I

Drill Hole	Depth of Drill Hole Interval		
	0-14 m	14-28 m	28-56 m
MOLO-14	√	√	√
MOLO-16			√
MOLO-17	√	√	√
MOLO-30	√	√	√
MOLO-35			√
MOLO-38	√	√	√
MOLO-46	√	√	√
MOLO-48	√	√	√

The optimized flowsheet and conditions shown in Figure 25 was chosen for the variability flotation program. Each variability composite was subjected to the flowsheet using this flowsheet with minor adjustments to the primary grind time, flotation times and reagent dosages.

These adjustments were required to address the different hardness of the various composite, as well as observations made during the test with regards to the flotation response.

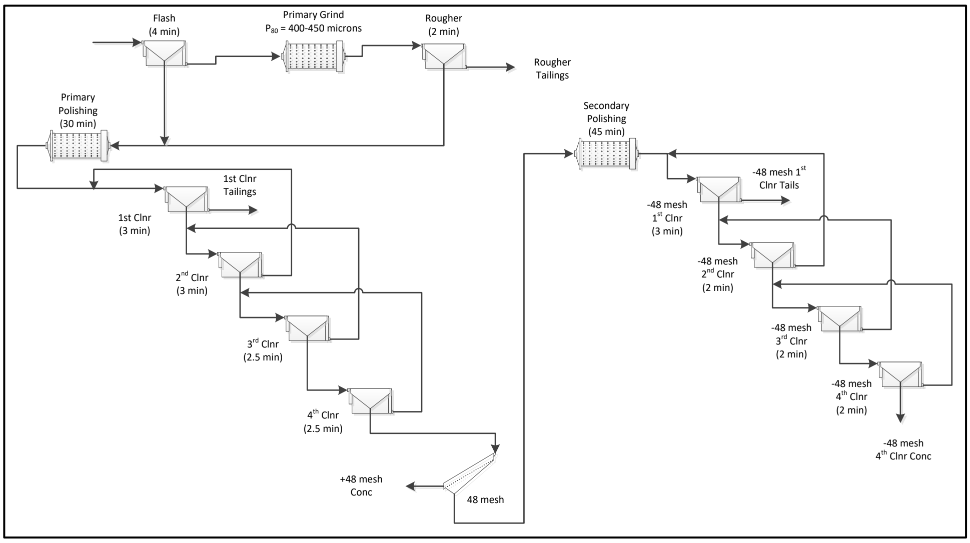


Figure 25: Optimized Flowsheet

Each of the twenty variability composites was subjected to an open circuit flotation test. The mass recovery into the various size fractions is summarized in Graph 9, alongside the average pilot plant results. The chart reveals that the average flake size distribution of the twenty variability composites compared well with the average results of the pilot plant.

The mass recovery into the +48 mesh and +65 mesh size fractions was up to 3.2% better for the variability samples. The range of mass recovery into specific size fractions was significant and the largest for the +48-mesh product.

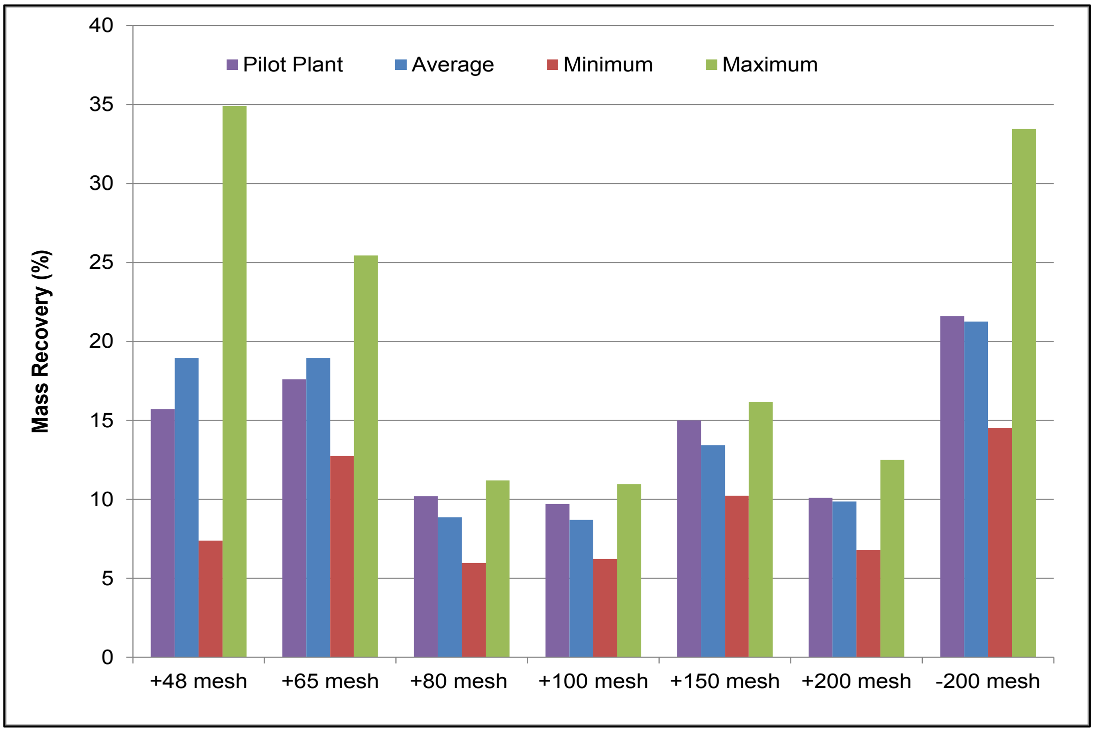
The lowest mass recovery into the jumbo flake category was 7.4% for the Molo-46 (0m to 14m) composite while the highest mass recovery into this product of 34.9% was achieved with the Molo-48 (0m to 14m) composite.

An analysis of different potential factors such as grade and depth of the composite did not produce a strong relationship between the variable and the flake size distribution.

The average mass recovery into the +80 mesh size fractions of concentrates from high grade composites was 47.4% compared to 46.1% for the low-grade composites.

With regards to depth, the average mass recovery into the +80 mesh size fractions of concentrates from shallow (0 metres to 14 metres), and medium (14m to 28m), depth composites was almost identical at 44.6% and 44.4%, respectively.

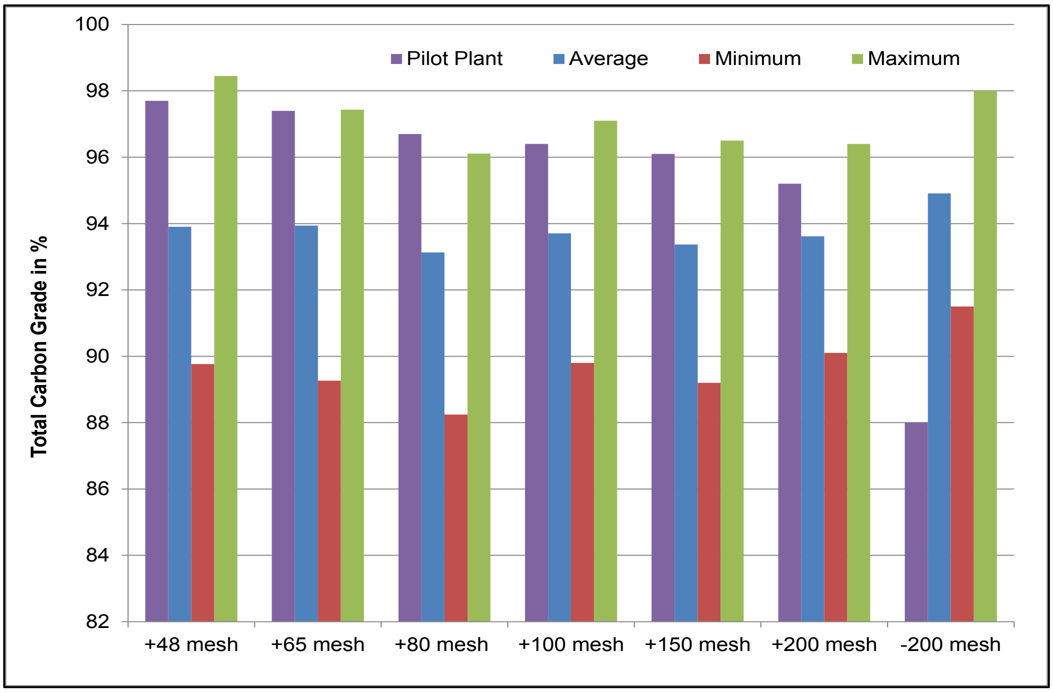
Only the concentrate from deep (28m to 56m), composites contained a slightly higher percentage of +80 mesh material at 50.2% mass recovery.



Graph 9: Range of Mass Recovery into Size Fractions (V1 to V32 and Pilot Plant)

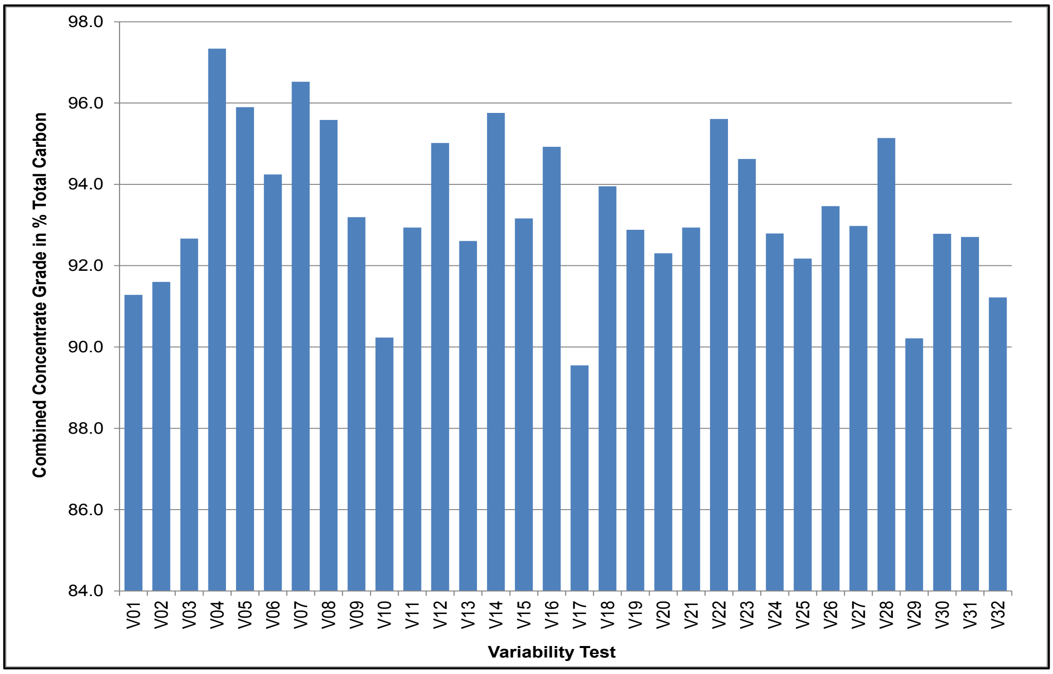
The average, minimum, and maximum total carbon grades of tests V1 to V32 for each size fraction are depicted in Graph 10 together with the average pilot plant results. While the maximum grades matched, or exceeded those of the pilot plant, the average concentrate grade was up to 3.5% total carbon lower compared to the pilot plant for all size fractions greater than 200 mesh.

The average grade of these size fractions was less than 94% total carbon and as low as 93.1% total carbon in the -65/+80 mesh size fraction. The minimum grades were 90% total carbon, or less for all size fractions greater than 200 mesh.



Graph 10: Range of Carbon Grades of Size Fractions (V1 To V32 and Pilot Plant)

The variability in the concentrate grade of the variability composites is further illustrated in Graph 11 which depicts the combined concentrate grade of each of the 32 variability flotation tests.



Graph 11: Combined Concentrate Grade of Tests on Variability Composites (V1 To V32)

The location and the metallurgical response of the drill hole intersections that were evaluated in the variability program within the Molo 2015 FS 5 year pit layout of the original 53,000 tpa Molo 2015 FS are depicted in Figure 26. The pit outline is demarked by the blue line. The three intersections for each drill hole represent the depth intervals 0m to 14m, 14mm to 28m, and 28m to 56m starting from top to bottom.

The colour coding of each depth interval was conducted based on the legend in Figure 26. The results show that only seven drill hole intersections within the Molo 2015 FS the 5 year pit layout achieved a concentrate grade of 94% total carbon, while five composites produced grades of greater than 92% and less than 94% total carbon and two composites graded less than 92% total carbon.

The remaining six composites that were tested fell outside the 5 year pit layout, but the metallurgical response was consistent with the other composites.

The analysis reveals that origin of the composites within the depth interval of the drill hole does not appear to have an impact on the metallurgical response as the three grade ranges were identified in each the three depth intervals.

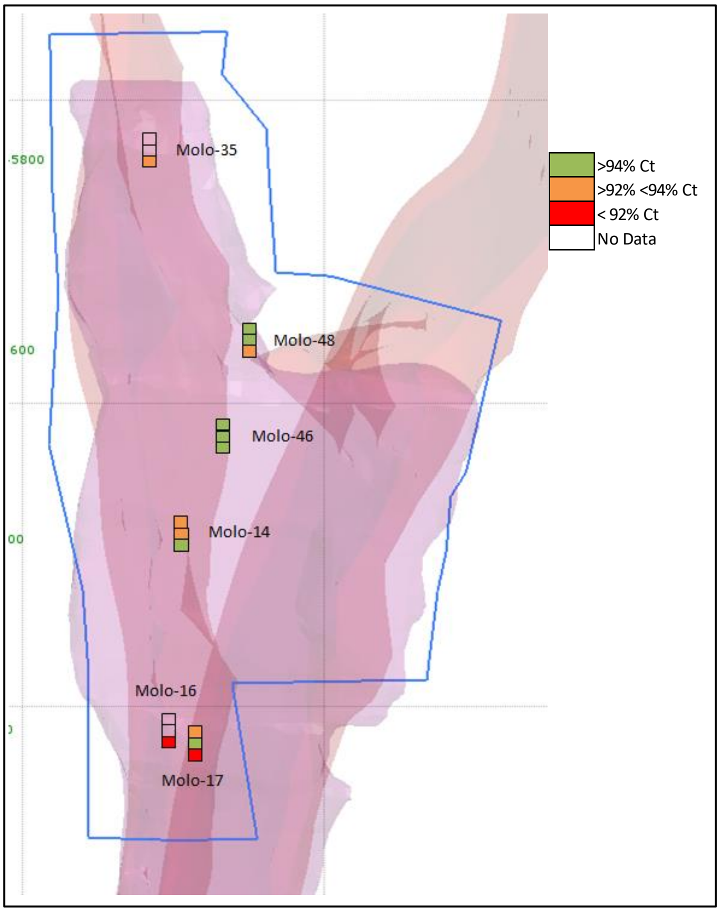


Figure 26: Location and Performance of Phase I Variability Drill Holes

The results obtained in this variability program revealed that the metallurgical response was inconsistent and appears independent of the location of the drill hole and sampling depth.

One limitation of the variability program was the fact that the six drill holes were located on a north-south axis through the deposit and did not cover the east-west extent. In discussions with the Project geologist, it was postulated that most of these drill holes originated from a transition zone between high grade and low grade ore and that this transition zone may be responsible for the inferior flotation properties.

It was paramount to determine if the large variability was linked to this potential transition zone, or if it is encountered throughout the entire mineral resource. If the inferior flotation response is only encountered in a limited and relatively small area, material from this area can be blended with other ore to achieve the flotation concentrate grade target. However, if the substantial variability is encountered throughout the entire mineral resource, upgrading strategies for the concentrate will have to be explored, which would likely result in the addition of an upgrading circuit at the tail end of the proposed process flowsheet. Hence, a decision was made to proceed with a second phase of variability flotation testing.

13.6.1 Optical Mineralogy

In order to develop a better understanding of the lower concentrate grade achieved on some of the variability composites, samples of concentrates grading only 90% to 92% total carbon was submitted for basic optical mineralogy examination.

The optical mineralogy revealed that graphite occurred less than 50 microns to 1,000 microns in size. The graphite was generally free, however, contained impurities of non-sulphide gangue (NSG) minerals.

The NSG minerals were fine-grained between 10 microns and 200 microns, but were inter-layered with graphite across its entire length.

One important observation was that less than 1% of the NSG minerals occurred as liberated grains. An example of inter-layered graphite particles is displayed in Figure 27. The non-sulphide minerals indicated by the green arrows were inter-layered with graphite (red arrows).

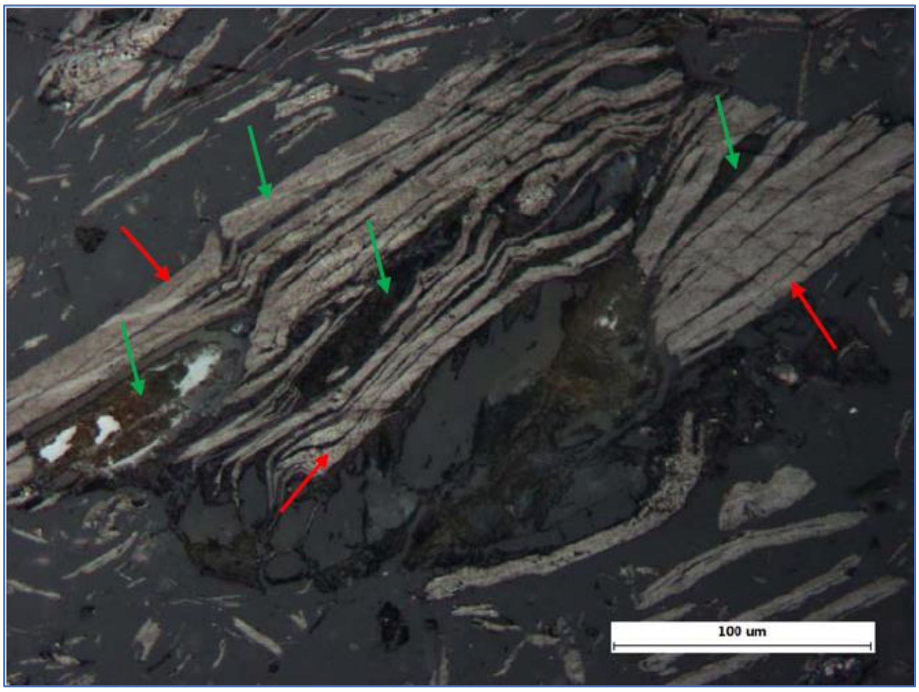


Figure 27: Intercalated Graphite

13.7 Variability Flotation – Phase II

A second variability flotation program was completed on the Molo deposit on a number of drill holes and area composites due to inferior metallurgical results obtained during an optimization and initial variability program. The primary reason for the inferior concentrate grade that was encountered in the flotation program was inter-layering of graphite and non-sulphide gangue minerals. It was postulated that this may be linked to a transition zone between high-grade and low-grade ore that was identified by the Project geologists.

A second phase of variability testing was conducted in the program documented in this report using drill core that originated within the 5 year mine pit boundaries.

A total of 3.5t of core was received at the SGS Lakefield site in December of 2014 and included all drill holes from the 2012 and 2014 campaign that fell within the 5 year mine pit perimeter. The drill core was prepared to generate 21 drill hole composites and 15 area composites. The average grade of all composites was 7.04% total carbon and 6.54% graphitic carbon.

13.7.1 Drill Hole Composites

A total of 7 drill holes with 3 depth intervals were subjected to open circuit cleaner tests to evaluate the metallurgical response of the Molo mineralization in the eastern and western areas of the Molo 2015 FS - 5 year pit layout.

The drill holes used in this phase of testing were generated in a drilling campaign that was conducted in 2012. Although the core was stored for more than 2 years and exposed to potential oxidation. It was concluded that any degradation of the core would likely not have an impact on the metallurgical response of the graphite.

This assumption was made based on the results from the first phase of variability flotation testing, which did not identify a statistically significant difference in the metallurgical response of weathered shallow material and fresh deep core.

A list of composites that were subjected to the variability flotation tests is shown in Table 26.

Table 26: Variability Composites – Phase II

Drill Hole	Depth of Drill Hole Interval		
	0-14 m	14-28 m	28-56 m
MOLO 12-05	√	√	√
MOLO 12-09	√	√	√
MOLO 12-21	√	√	√
MOLO 12-26	√	√	√
MOLO 12-33	√	√	√
MOLO 12-37	√	√	√
MOLO 12-38	√	√	√

The same flowsheet and conditions as in the Phase I variability program were employed in the 21 open circuit flotation tests.

The location and metallurgical performance of the Phase II drill hole composites are depicted Figure 28 and confirms Phase I results.

Good and poor performing composites were frequently encountered in the same drill hole in adjacent depth intervals. Based on the metallurgical results of the individual drill hole composites, no specific area could be characterized as consistently good, average, or inferior performing.

The results suggest that further treatment of the concentrate is required. Repeat tests with longer polishing times using the existing flowsheet failed to produce improved concentrate grades in most cases.

Optical mineralogy that was conducted on concentrates from the Phase II variability program confirmed the metallurgical challenge of intercalated graphite, which will require a different more aggressive upgrading approach than the standard polishing mill grinding.

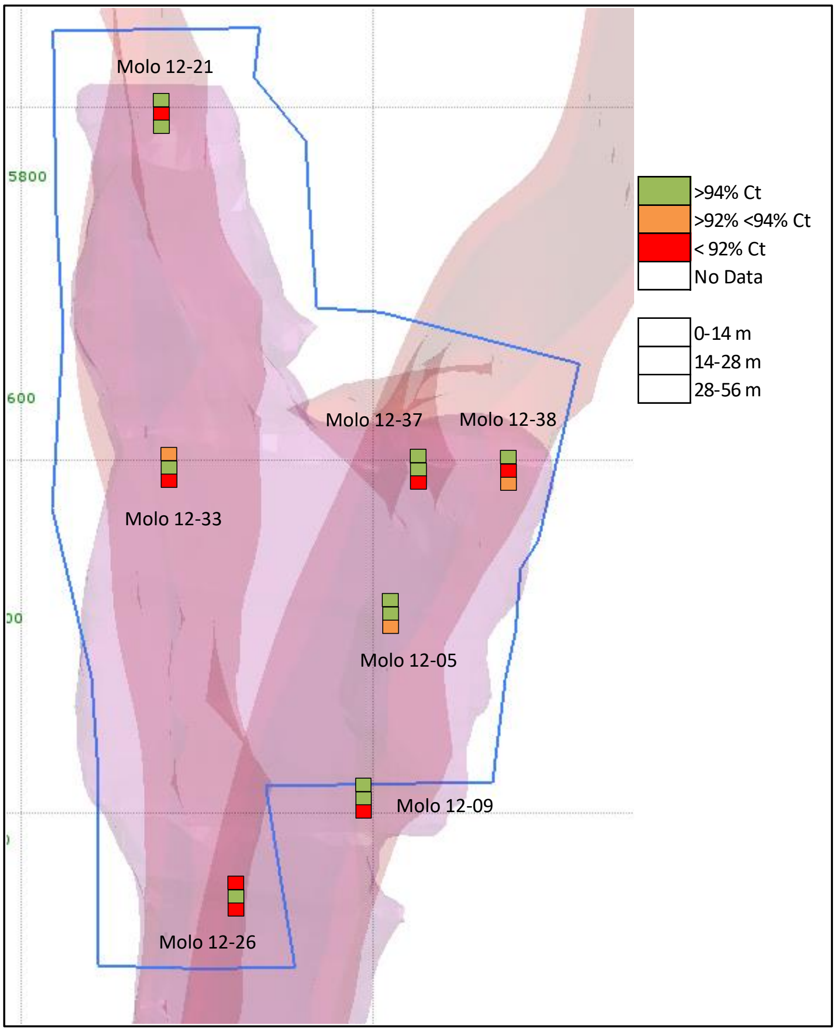


Figure 28: Location and Performance of Phase I Variability Drill Holes

13.7.2 Area Composites

In an attempt to develop a better understanding of the average metallurgical response of the Molo mineralization, area composites were generated, which included samples from all drill holes that were available.

For this purpose, the Molo 2015 FS 5 year mine pit layout was split into five areas and each area composite was then generated by combining sub-samples from all drill holes that fell into a specific area.

The three depth intervals 0m to 14m, 14m to 28m and 28m to 56m were maintained for the area composites ie. a total of 15 area composites were generated. A summary of the drill holes that were included in each area composite is provided in Table 27.

Table 27: Drill Holes included in Area Composites

Composite ID	Drill Hole ID's
Area Composite 1	MOLO 12-20, 12-21, 12-28, 12-29
Area Composite 2	MOLO 12-01, 12-02, 12-03, 12-18, 12-19, 12-27, 12-30, 12-32, 12-33, 12-34, MOLO-01
Area Composite 3	MOLO 12-04, 12-05, 12-35, 12-37, 12-38
Area Composite 4	MOLO 12-23, 12-24, 12-31, 14-15
Area Composite 5	MOLO 12-07,12-08, 12-09, 12-26, 12-40, 14-17, MOLO-22

The average flake size distribution of the fifteen area composites is shown in Table 28. For comparison purposes, the average flake size distribution of the pilot plant campaign and the two variability programs are presented in the same table.

The data reveals good agreement between the results, which attests to the robustness of the flake size distribution across the Molo mineralization.

Table 28: Comparison of Flake Size Distribution

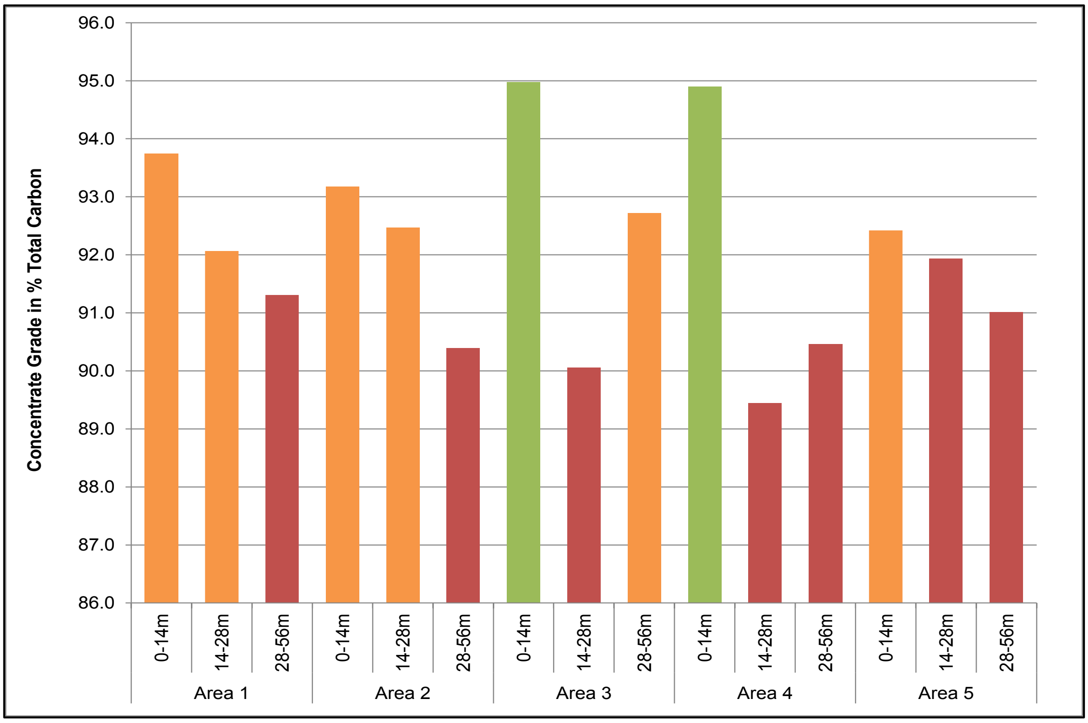
Screen Size	Area Composites	Variability Phase I	Variability Phase II	Pilot Plant Campaign
+48 mesh	26.5	19.0	26.6	15.7
+65 mesh	17.0	19.0	18.3	17.6
+80 mesh	8.1	8.9	8.3	10.2
+ 100 mesh	6.6	8.7	6.6	9.7
+150 mesh	12.2	13.4	12.0	15.0
+ 200 mesh	8.4	9.9	8.2	10.1
- 200 mesh	21.2	21.3	19.9	21.6

The grades of the combined concentrates of the fifteen area composites are presented in Graph 12 and Graph 13. Only two of the composites produced concentrate grades of greater than 94% total carbon.

Six composites graded between 92% and 94% total carbon and the remaining seven composites produced concentrates of less than 92% total carbon.

The best performing composite was that of Area 3 (0m to 14m) with a concentrate grade of 95.0% total carbon, while the worst performing composite was Area 4 (14m to 28m) which produced a concentrate grading of only 89.4% total carbon.

The average concentrate of all fifteen composites was 92.1% total carbon.



Graph 12: Combined Concentrate Grades of Twenty-One Area Composites

The location of the 5 areas and the metallurgical response of the 15 area composites are shown in Figure 29. Only the top 14m of the Area 3 and Area 4 composites produced good concentrate grades, which is consistent with the results of the individual drill holes that originated from that area and depth interval.

The postulation that further upgrading of the graphite flotation concentrate is required was confirmed for the area composites.

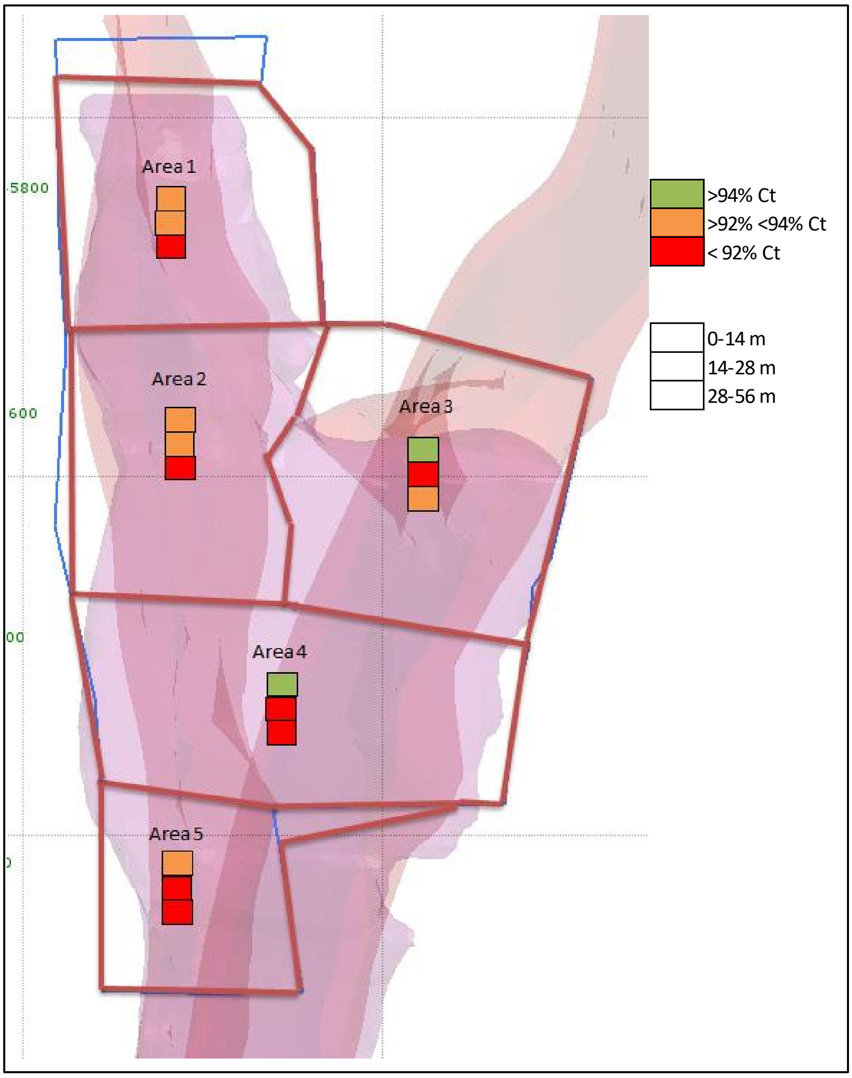


Figure 29: Location and Performance of Area Composites

The average open circuit carbon recovery for the fifteen area composite tests was 87.6% and ranged between 73.2% for the Area 1, (14m to 28m) composite and 93.6% for the Area 3 (28m to 56m) composite.

Since open circuit tests treat the intermediate cleaner tails as final tails, the recoveries are lower compared to closed circuit operation. To determine the closed-circuit performance, four locked cycle flotation tests were completed on the Molo mineralization.

The analysis of the results determined that 66% of the carbon units that reported to the intermediate cleaner tails report to the final concentrate in closed circuit operation.

This factor was applied to the 15 tests using the area composites to arrive at a closed-circuit carbon recovery Projection of 90.5%.

13.8 Concentrate Upgrading Tests

The average concentrate grade of 92.1% of the area composites has a significant impact on the economics of the Project. Concentrates grading between 94% and 97% total carbon are in higher demand and achieve substantially higher prices compared to a concentrate grading only 90% total carbon. To increase the graphite flotation, concentrate from approximately 92% total carbon to at least 94% total carbon, two upgrading strategies were evaluated.

The first approach applied high temperature drying at 400°C for one hour followed by classification of the dried concentrate on a standard set of sieves. It was postulated that the high temperature drying for an extended period of time could possibly weaken, or break the bonds between the graphite layers and non-sulphide gangue minerals within the inter-calated graphite flakes.

Screening the product could then result in upgrading of the coarser graphite flakes if the gangue minerals are liberated in the drying process and report to the smaller size fractions.

This upgrading approach failed to improve concentrate grades and a cleaner flotation test conducted on the dried concentrate did not produce further grade improvements.

Optical mineralogy that was conducted on the dried concentrate confirmed the existence of intercalated graphite, which led to the rejection of this upgrading strategy.

The second upgrading strategy evaluated a series of different sized grinding media and grinding mills, as well as sodium silicate as a gangue depressant. A combined concentrate from the Phase II variability program was homogenized and split into equal test charges that were then subjected to five different upgrading conditions.

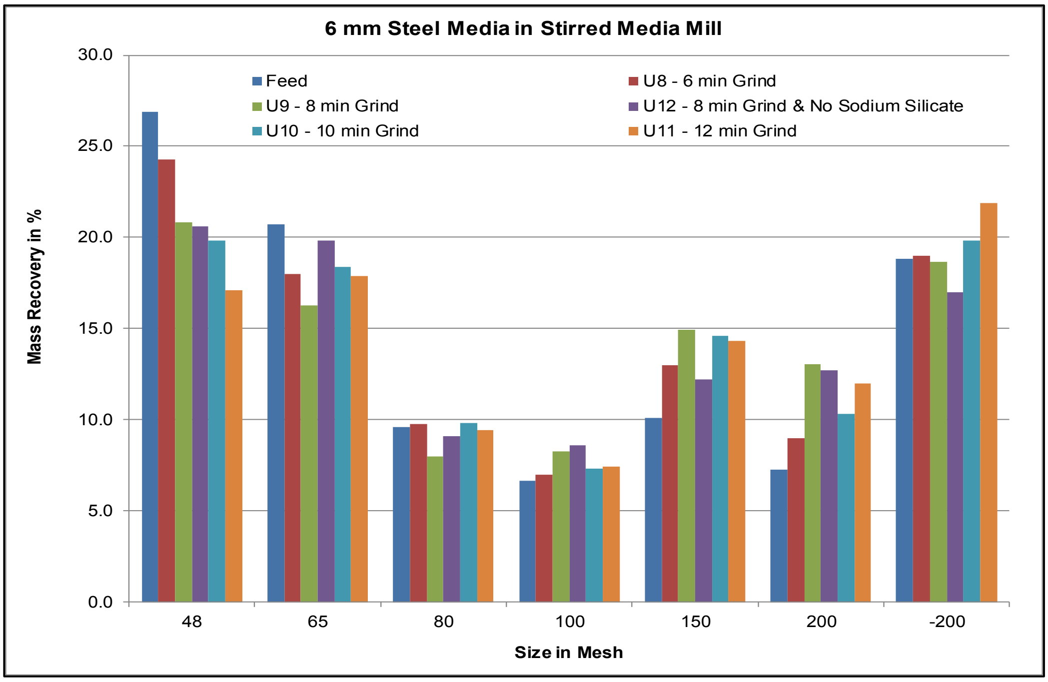
The most promising results were achieved using an attrition scrubber with a 1 mm ceramic media and a stirred media mill with 6 mm steel media.

Ten additional tests were carried out using the attrition mill and attrition scrubber. A weighted combined concentrate of all fifteen area composite tests was generated for those tests, which was a good representation of the average mineral resource.

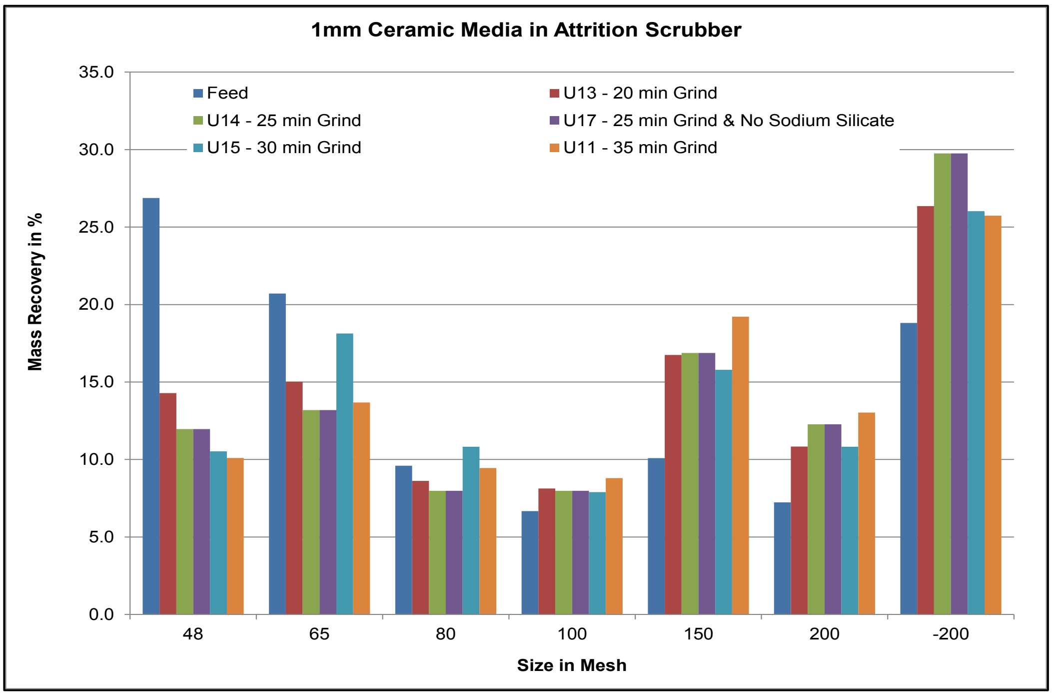
The variables that were modified in the 10 tests were the grinding times and the use of sodium silicate.

While the test using the 1 mm ceramic media in an attrition scrubber produced good and combined concentrates grades of more than 96% total carbon, the flake size degradation was significantly higher compared to the tests using the stirred media mill. This is evidenced in Graph 13 and Graph 14 which also depicts the mass recovery into the size fractions of the final concentrates of tests conducted with the stirred media mill and attrition scrubber, respectively. To quantify the degree of flake degradation, the charts also include the data for the feed sample prior to milling or scrubbing.

The degradation of the flakes larger than 65 mesh was less pronounced for the stirred media mill and even at the longest grind time the mass recovery into the +48-mesh size fraction was still 17.1%. In contrast, the shortest grind time in the attrition scrubber reduced the mass recovery into the +48 mesh concentrate to 14.3%. The shortest grind time in the stirred media mill reduced the mass of the +48 mesh and -48/+65 mesh concentrate by only 2.6% and 2.7%, respectively.



Graph 13: Stirred Media Mill Size Fraction Analysis – Mass Recovery



Graph 14: Attrition Scrubber Size Fraction Analysis – Mass Recovery

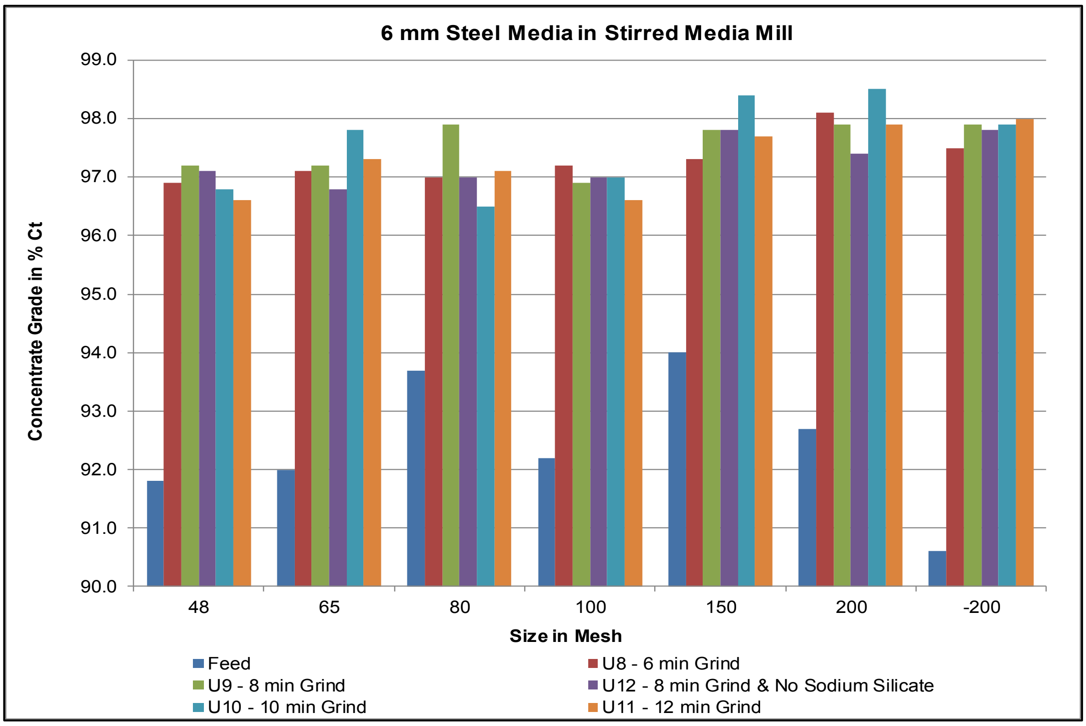
The total carbon grades into the size fractions of the combined concentrate for the 5 tests using the stirred media mill are presented in Graph 15. The shortest grind time of 6 minutes produced concentrate grades of 96.9% to 98.1% total carbon for the various size fractions. Even if the worst-case scenario of the relative measurement error of 1.4% associated with the total carbon analysis by LECO SC_632 is applied, the results are consistently above the minimum grade target of 95% total carbon.

Because the flake degradation increased with longer grinding times without a clear improvement in the concentrate grades, the test with the shortest grind time was deemed the most successful one.

It should be noted that all size fractions of the concentrates of the 5 upgrading tests using the stirred media mill yielded at least 96.5% total carbon, which attests the robustness and repeatability of the upgrading approach.

It may be possible to reduce the amount of flake degradation in the large and jumbo flake categories with the addition of a classification stage of the intermediate concentrate at 80 mesh prior to stirred media milling followed by separate cleaning circuits for the screen oversize and undersize fractions. This approach allows to tailor the final graphite concentrate grade distribution to specific market demands.

A comparison of the results of tests U9, (sodium silicate at 40 kg/t of concentrate), and U12 (no sodium silicate), reveals that the gangue depressant only increased the combined concentrate grade by 0.2% from 97.1% to 97.3% total carbon. Since this grade improvement is statistically insignificant, the addition of sodium silicate cannot be justified for inclusion in the upgrading circuit.



Graph 15: Concentrate Grades of Size Fractions – Stirred Media Mill

The complete Molo flowsheet with the attrition circuit is depicted in Figure 30.

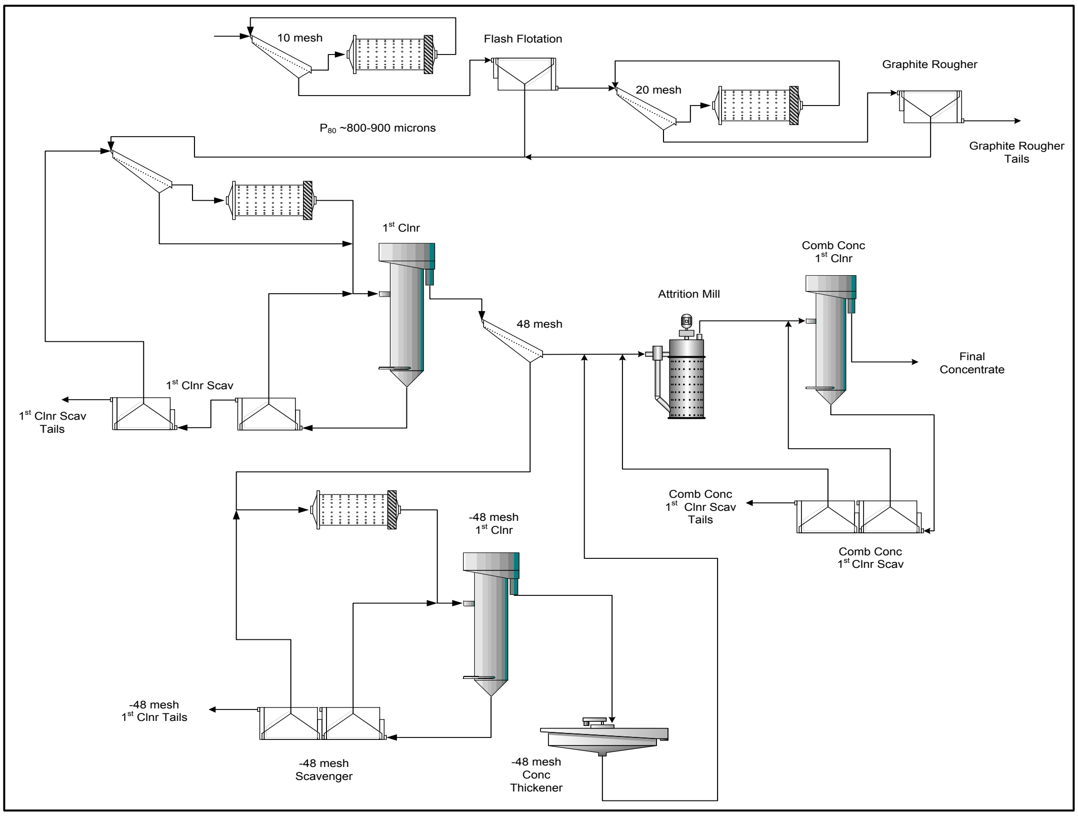


Figure 30: Molo Flowsheet Including Stirred Media Mill Circuit

Since the concentrate used in the upgrading tests was a weighted composite of all area's composite concentrates, the flake size distribution and concentrate grade of test U8 was considered most representative of the average plant product in the first several years of mining operation.

The mass recovery into the various size fractions and associated concentrate grades are presented in Table 29.

The open circuit carbon recovery into the final concentrate of test U8 was 97%. Although carbon recovery will likely increase in closed circuit operation, this conservative number was applied to the circuit carbon recovery of 90.5% prior to upgrading.

As a result, the combined carbon recovery for the main process flowsheet and the upgrading circuit is 87.8%.

Table 29: Mass Recovery and Total Carbon Grades of Size Fractions of Final Concentrate

Product Mesh	Microns	Mass (%)	Grade (% Total Carbon)
+48	+297	24.3	96.9
-48/+65	-297/+210	18.0	97.1
-65/+80	-210/+177	9.8	97.0
-80/+100	-177/+149	7.0	97.2
-100/+150	-149/+106	13.0	97.3
-150/+200	-106+74	9.0	98.1
-200	74	19.0	97.5
Total		**100.0**	**97.2**

13.8.1 Optical Mineralogy of Upgraded Concentrate

The 7 size fractions of the most successful upgrading test U8 using the stirred media mill were submitted for optical mineralogy. All samples displayed similar mineralogical characteristics.

Non-sulphide gangue (NSG) minerals were generally fine grained (<20 microns to 500 microns). NSG occurred as minor liberated grains only in the +48 mesh, and sporadically in some of the other fractions.

The bulk of the NSG occurred inter-layered with graphite grains. They were developed along the long axis of the graphite particles and are of varied width.

A photomicrograph of graphite flakes in the +48-mesh size fraction of the U-8 third cleaner concentrate is depicted in Figure 31 to illustrate the inter-layering. The image shows graphite (red arrows) that is largely liberated in the sample. However, non-sulphide gangue minerals (green arrows) are mainly inter-layered within graphite.

While the inter-layering has not been eliminated in the upgrading stage, the frequency has been reduced significantly, thus resulting in the mean grade improvement of approximately 5% total carbon.

It is postulated that the coarser intercalation between graphite and NSG minerals is separated efficiently in the stirred media mill, but that the thinner layers between graphite and NSG minerals are more difficult to segregate. However, given the more than adequate concentrate grade of 97.2% total carbon in the U8 test, eliminating all inter-layering is not required.

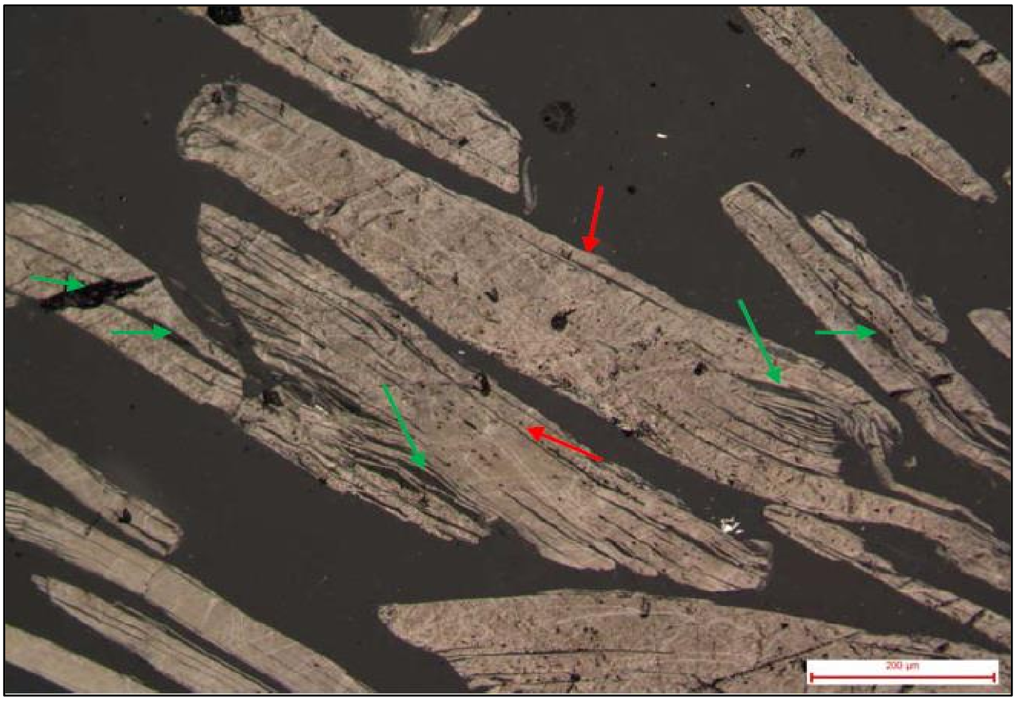


Figure 31: Optical Photomicrographs (Ppl) From The U-8 SFA 3rd ClN CONC +48 Mesh

13.9 Total Carbon and Graphitic Carbon Assay Methods – SGS

Carbon occurs as organic carbon, carbonate carbon and graphitic carbon. The 3 types of carbon combined represent the total carbon content of a sample.

The total carbon content of a sample is determined by combusting a pulverized sample followed by infrared detection on LECO instrumentation. Lower grade samples are analysed with a LECO 844, while samples higher than 30% carbon are analysed for carbon on the SC632 instrument. Both instruments use high temperature combustion followed by infrared detection of CO2.

The graphitic carbon content of a sample is determined in a three step process using couLoMetric analysis. The pulverized sample is roasted in an oven at 500°C for 15 minutes to remove any organic carbon. The carbonate carbon is determined by subjecting one aliquot of the roasted samples to couLoMetric analysis. A second aliquot is used to determine total carbon using a tube furnace. The graphitic carbon is then calculated by the difference between the total carbon and carbonate carbon.

13.10 Additional Testing

The physical properties of graphite are very different to most other commodities because of its particle shape and density. Consequently, it is essential that all unit operations for a proposed graphite processing plant are evaluated in laboratory, or pilot scale trials to obtain robust data for the process design criteria.

Graphite concentrate that was generated in the 2013 pilot plant campaign was shipped to various equipment vendors to evaluate dewatering, drying, and screening applications. These unit operations are required to produce a final saleable product from the initial graphite flotation concentrate.

Further, dewatering tests were carried out on the combined tailings from the pilot plant campaign as poor settling properties of the fine particles in the tailings were observed in laboratory and pilot scale testing.

13.10.1 Thickening

Two equipment Vendors conducted thickening test work on concentrate and tailings samples that were generated in the pilot plant campaign.

Both Vendors conducted static settling and dynamic tests to identify a suitable flocculant and to establish process parameters to achieve a high thickener underflow density and clear overflow:

- **Vendor A** quantified a solids loading rate of 0.25 t/m^2/h for the concentrate thickener yielding an underflow density of 36% solids at a flocculant dosage of 5 ppm Magnafloc 919. The overflow contained 230 ppm solids. The tailings required the addition of a coagulant to achieve satisfactory overflow clarity of less than 100 ppm. The reagent regime consisted of 5 ppm Magnafloc 1011 and 500 ppm Magnafloc 370. An underflow density of 50% solids with an overflow containing 70 ppm solids was achieved at solids loading rate of 0.75 t/m^2/h.
- **Vendor B** quantified the solids loading rate for the concentrate thickener at 0.64 t/m^2/h at a flocculant SNF 905 VHM dosage of 20 g/t. The solids loading rate for the graphite tails was 0.46 t/m^2/h at a flocculant SNF 934 VHM dosage of 140 g/t. The concentrate and tailings thickener underflow solids concentration at 2 hours retention time was 41% and 47% respectively. The overflow clarity for the graphite concentrate was clear, at less than 100 ppm, while the graphite tailings contained a solids concentration of 3,500 ppm.
- **Vendor A** required a large dosage of coagulant Magnafloc 370 to produce an overflow clarity for the graphite tailings thickener application of less than 100 ppm suspended solids, while **Vendor B** failed to generate an acceptable graphite thickener tails overflow clarity. Two reagent suppliers, supplier A and supplier B, were contracted to carry out a more comprehensive reagent screening to evaluate a reagent regime requiring lower dosages.
- **Supplier A** recommended the use of approximately 125 g/t of flocculant Magnafloc 24, 155, 1011, or 919 in conjunction with 100 ppm– 150 ppm of coagulant Magnafloc 1707 to achieve the desired overflow clarity of less than 100 ppm suspended solids in the graphite tailings thickener.
- **Supplier B** did not develop a reagent regime that achieved dosages lower than the ones recommended by equipment Vendor one.

13.10.2 Filtration Tests

Two suppliers conducted filtration test work on concentrate and tailings samples that were generated in the pilot plant campaign.

The first vendor conducted bench scale testing to evaluate filter cloth selection, filter cake thickness, filtration rate, moisture content of the cake, and cake handling characteristics to achieve 15% to 20% w/w moisture in the concentrate cake.

The tests conducted by the first Vendor produced concentrate filter cakes with cake moisture content between 11.0% w/w and 20.5% w/w at filtration rates of 179 kg to 417 kg Ds/m^2/h. Filtration tests on the tailings produced filter a cake moisture content between 12.7% and 17.9% w/w at filtration rates of 92 kgs to 218 kg Ds/m^2/h.

Vacuum filtration tests conducted by the second vendor produced a concentrate cake moisture content of 23% w/w at a filtration rate of 327 kg Ds/m^2/h.

Pressure filtration tests on the concentrate produced a cake moisture content of 23.2% w/w. The vacuum filtration properties of the tailings were poor yielding a cake moisture content of 32% w/w at a filtration rate of 41 kg Ds/m^2/h.

13.10.3 Concentrate Drying

Drying tests were conducted using a rotary dryer and a fluid bed dryer.

While the rotary dryer did not operate well treating the as-received graphite concentrate with a moisture content of 32% to 39% w/w, good performance was obtained when back-mixing some of the dried concentrate to adjust the feed moisture content to 26% w/w.

Since the filtration tests conducted by both vendors produced filter cakes with a lower moisture content than 26% w/w, the rotary dryer is a suitable drying technology to achieve a product moisture content of less than 0.5% w/w.

The fluid bed dryer work failed to produce results.

13.10.4 Wet and Dry Screening Tests

Wet and dry screening tests were carried out at one Vendor to evaluate screening applications on intermediate and final graphite concentrates and to determine if screens could be employed in a dewatering application.

Classification of the dried graphite concentrate was performed at 50 mesh, 80 mesh and 200 mesh using dried concentrate that was generated during the rotary dryer tests.

The dry screening tests suggested that classification can be carried out at a rate of 1.0 tph per metre of screen width, using Vendors screens.

Tests on a wet graphite concentrate to evaluate the classification of the intermediate concentrate at 80 mesh and 140 mesh yielded screening rates of 2.0 tph to 2.2 tph per metre of screen width, using Vendors screens.

Dewatering tests on the -140 mesh material was carried out on a 270 mesh screen deck. The mass recovery into the screen oversize was 63.4% at a moisture content of 49.7% w/w.

13.10.5 Additional Comments

The tails stream from the upgrading, or attrition circuit is being pumped out to final tails, or discard.

The feed to the Attrition circuit is of significantly high grade, (essentially already a product, just not the right final grade), and the attrition circuit serves only to upgrade the product to a higher grade. So, it follows that the tailing stream from this circuit should also still be of high grade enough to warrant that it be recycled back into the circuit.

This stream should not report to final tailings, instead it will be directed to the polishing mill discharge to be treated in the primary cleaning circuit.

14 MINERAL RESOURCE ESTIMATE

This Mineral Resource Estimate was produced by CCIC on September 1, 2014. The approach and methodologies applied in the MRE are in accordance with the definition as defined by NI 43-101 and following the Canadian Institute of Mining, Metallurgy and Petroleum CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines as amended (CIM, 2019). Leapfrog™ software was used to construct volumetric solids for the zones of mineralisation. The three-dimensional resource modelling, as well as the geostatistical techniques for grade estimation was undertaken using Datamine™. The key assumptions and methodologies used for this resource estimate are fully outlined below.

Note that no further work was done for this study in amending the MRE.

14.1 Geological Database

14.1.1 Topography

A three dimensional ("3D") Digital Elevation Model ("DEM") of the topography was supplied by the Company as 0.5m contours in ascii format. These contours were generated from an airborne survey using the SenseFly drones, in 2014. Collar elevations from trenches and drill holes have been resurveyed using a differential GPS and incorporated into the topography. The topography is flat lying, with the highest elevation in the north-western side and dipping gently to the south (Figure 32). Elevations within the area of study range from 570 mamsl to 543 mamsl, with an average gradient of 2°. The higher grade domain on the western side creates a ridge up to 2m in elevation.

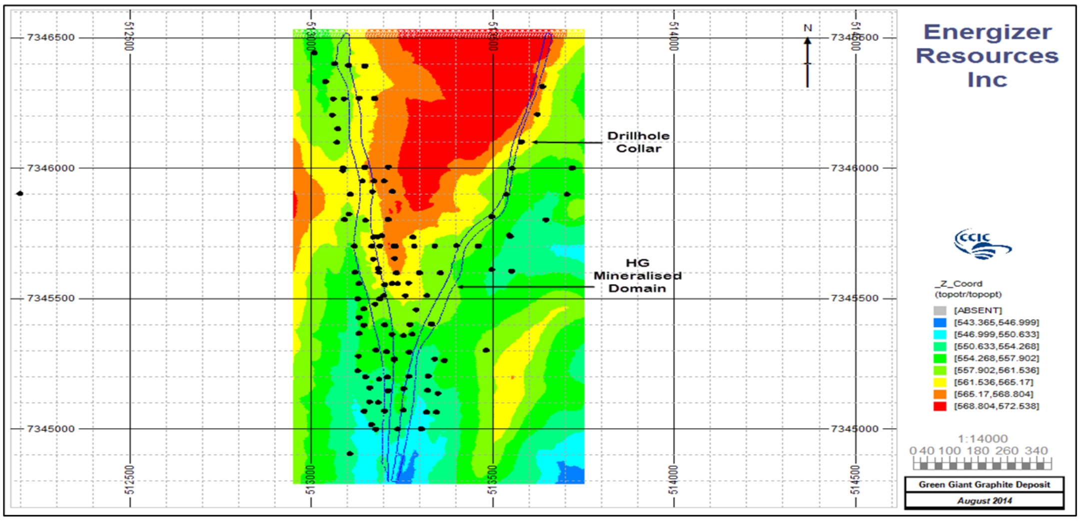


Figure 32: Topographic Contours; Elevations In MAMSL

14.1.2 Drill Holes

This MRE is based on 80 drill holes, (total 11,660m), and 35 trenches (total 8,492m), drilled by the Company. Drill spacing varies from 100m x 100m in poorly informed areas to 50m x 50m in well informed areas. Figure 33 illustrates a plan view of the drill holes and trenches, coloured on % C grades. Drill holes are orientated approximately 45° to the east. The database containing drill hole and trench information was supplied by the Company in a Microsoft Access format. Logging codes used for lithological modelling are summarised in Table 30.

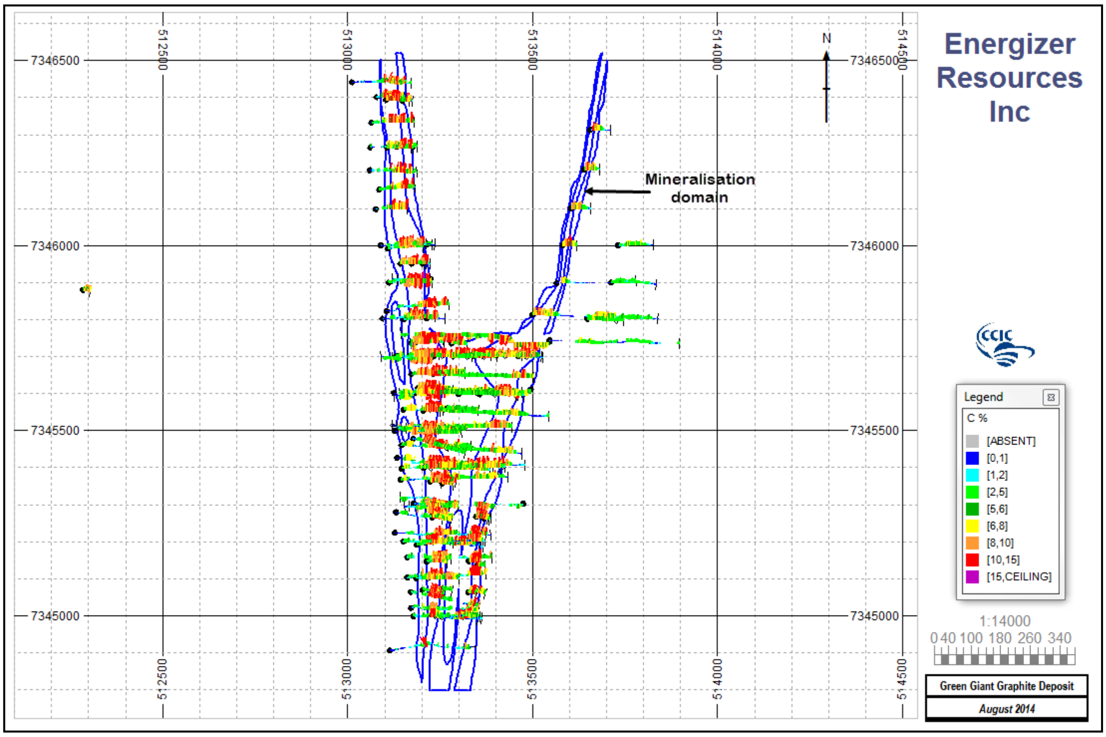


Figure 33: Drill Hole Posting Plan Coloured on C (%) Grades

Table 30: Summary of Lithological Codes Used

Logging Code	Description
Gp Gn	Graphite Gneiss
Gt Gn	Garnet Gneiss
Mb	Marble
SAPR	Saprolite
PEG	Pegmatite
OVBN	Overburden

Spatial and statistical comparisons of % C between drill holes and trenches are presented in Figure 34 and Graph 16. There is good spatial and statistical correlation between the two datasets.

Overall, trenches show a slight positive bias for the Mean. Reason for this may be due to the infilling drilling program focussed on upgrading the higher grade portions of the deposit.

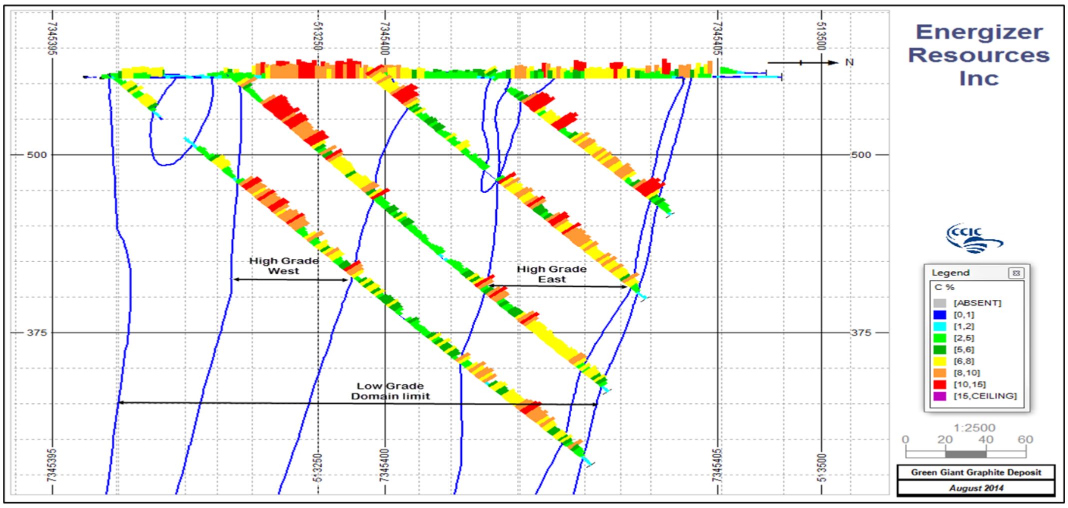
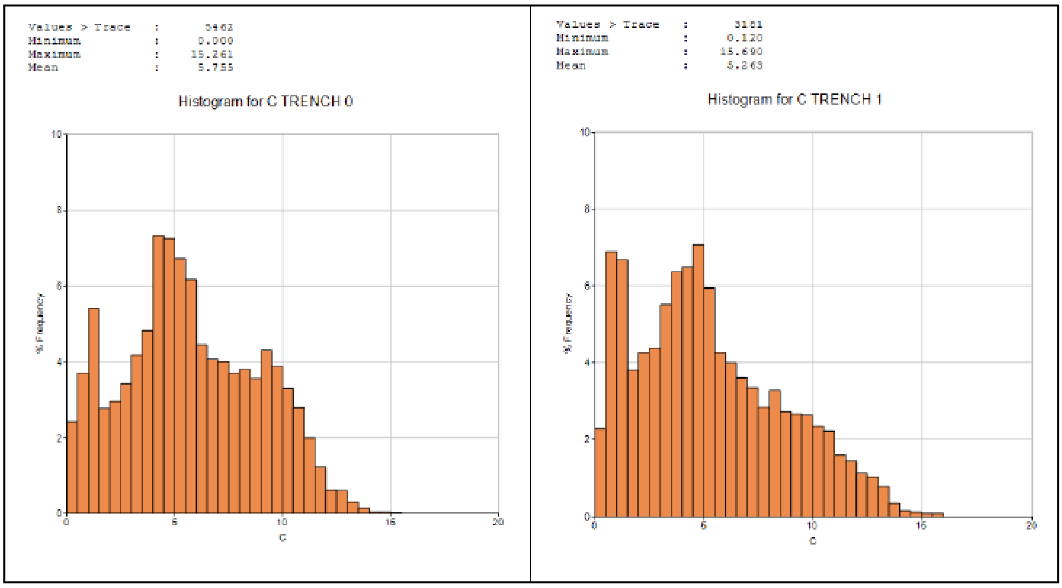


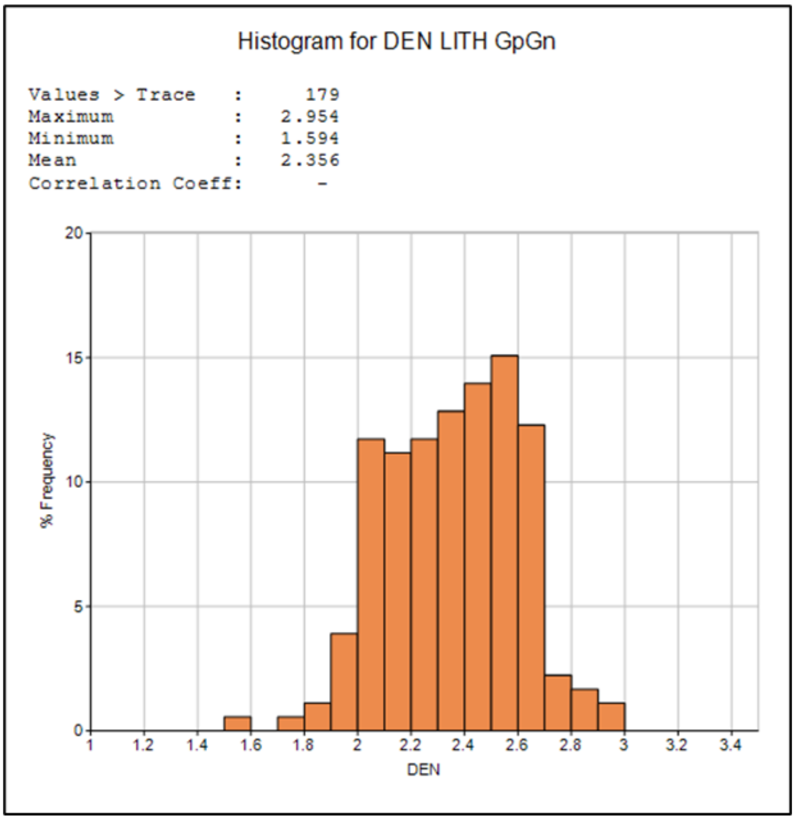
Figure 34: Cross Section Showing the Grade Distribution (C%) Nn the Drill Holes and Trenches



Graph 16: Statistical Comparisons Between Diamond Drill Hole (Dd) and Trench (Th) Samples

14.1.3 Relative Density

A total of 226 RD measurements are contained in the Molo database, 179 of which were for the graphitic gneiss (GpGn). These are presented as a histogram in Graph 17 below. The average RD for all 226 readings is 2.39 t/tm^3. RD values within the GpGn range from 1.59 t/tm^3 to 2.95 t/tm^3, with an average of 2.35 t/tm^3.



Graph 17: Histogram of Relative Density Readings within the Graphitic Gneiss

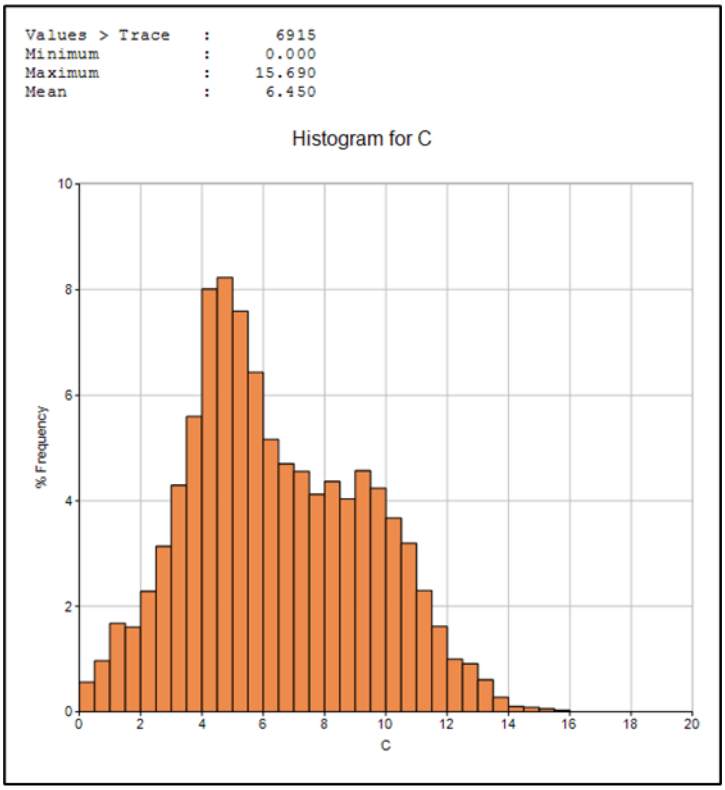
14.2 Geological Model on Which the Grade Estimation is Based

14.2.1 Grade Domaining

The deposit is split into three domains based on mineralisation grade, namely:

- A "barren" or "un-mineralized" domain. This is a lithological boundary that separates the un-mineralised Garnet Gneiss from the mineralized Graphitic Gneiss. This boundary was treated as a "hard" boundary during grade domaining and estimation.
- The mineralized Graphitic Gneiss has been sub-domained into separate "low grade" and "high grade" domains, based on C grade characteristics. Histogram (%) of C grades prior to sub-domaining illustrates a bimodal distribution. A threshold grade of 6% C was, therefore, used as a guideline to sub-domain the "low" and "high" grade zones, while maintaining spatial continuity.
- It is common that lower grade graphitic gneiss tends to produce larger sized graphite flakes than higher grade graphitic gneiss. This is, however, not the case for this deposit. Metallurgical Variability testwork by SGS in October 2014 found that the average mass recovery into the +80 mesh size fractions of concentrates from high grade composites was 47.4% compared to 46.1% for the low grade composites.

Refer to Graph 18 below.



Graph 18: Histogram of C Distribution

14.2.2 *Grade Domaining using Leapfrog*

Leapfrog is an implicit 3D modelling engine that works of a Radial Basis Function. The modelling methodology differs from the traditional way of deterministically digitising out the zones of interest along section lines, then stitching them together to create a 3D wire frame. Leapfrog is instead based on an algorithm that uses all the data points in 3D space, together with geological constraints and parameters to automatically generate volumes of interest. The benefits of using Leapfrog are that:

- Interpretations are not limited to drill holes along a section line. Incorporating drill holes from neighbouring section lines makes the model a full 3D interpretation, ensuring good correlation between section lines.
- The algorithm can generate very complex forms, resulting in more efficient domaining.
- Because the "low" and "high grade" domains are generated concurrently, there are no overlaps, or protrusions between domains.

The contacts for the three domains were flagged using the "Interval Selection Tool" in Leapfrog, which allows the user to interactively determine the intervals that are to be included, or excluded, from the different domains.

The mineralised domain is composed of Graphitic Gneiss, within which occur two lenses of barren Garnet Gneiss. The mineralised domain was further subdivided into a "low grade" domain and two "high grade" lodes. The "high" grade lodes are referred to as "high grade east" and "high grade west" domains. An isometric, and a section view, are illustrated in Figure 35 and Figure 36 respectively.

The mineralisation limits, represented by the graphitic gneiss are shown as pale green, with the barren Garnet Gneiss represented as a white background. Mineralisation strikes approximately north-south, dipping between 75° and 80° to the west. The thickness of the "low grade" zone varies from 60m, where only the "high grade west" domain is developed, to more than 260m in the central portions. "High grade" domains are generally 60m thick, thinning to the south.

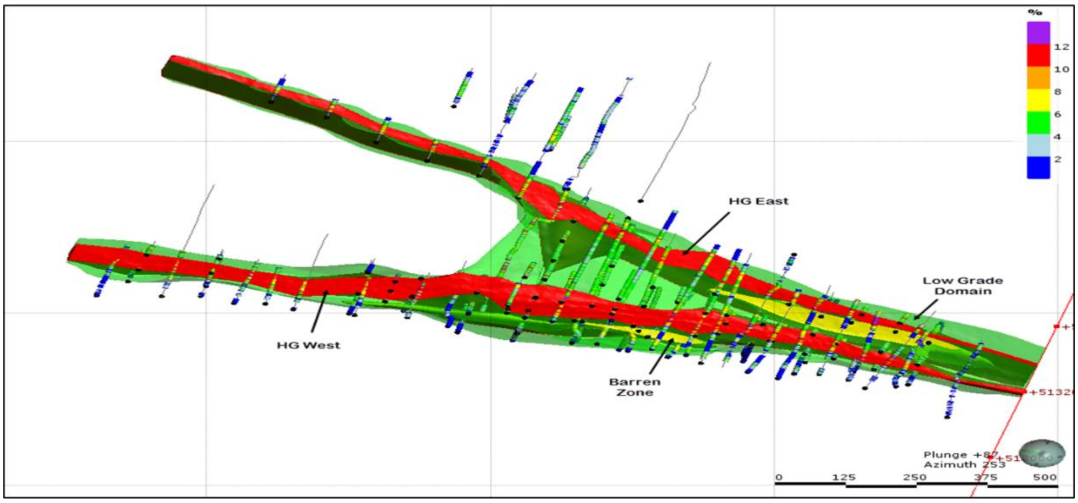


Figure 35: Isometric View Showing "Low" and "High" Grade Domains

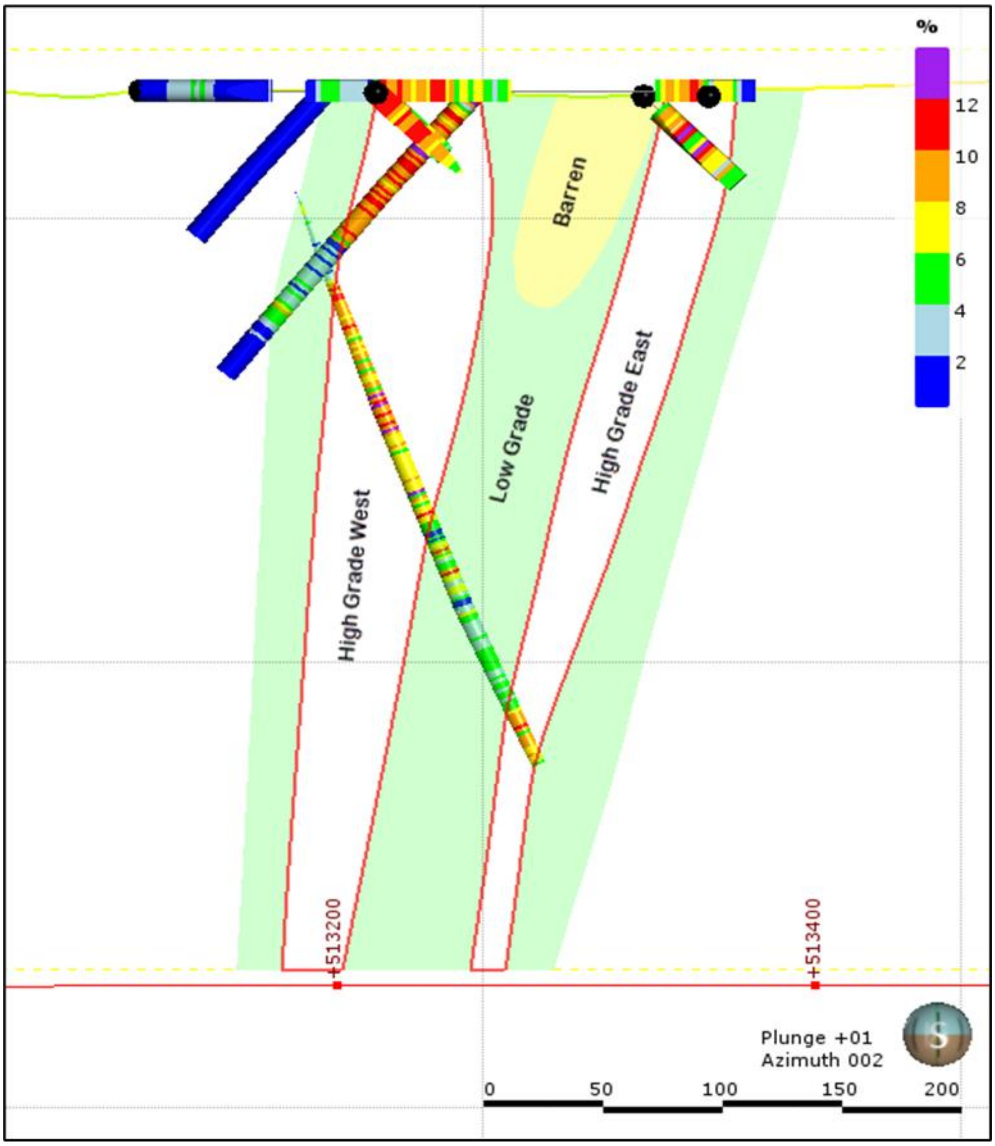
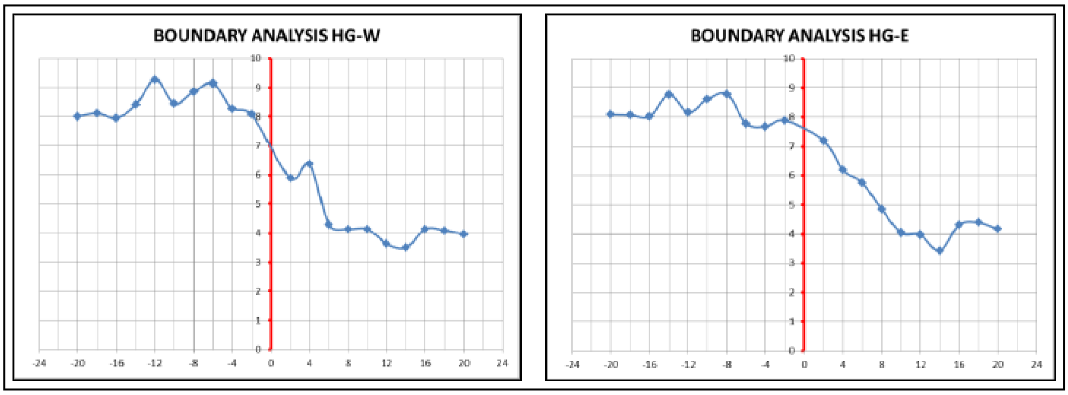


Figure 36: West-East Section Showing the "Low" and "High" Grade Domains

14.2.3 Domaining in Datamine

The envelopes generated in Leapfrog were imported into Datamine for sample and block model flagging. Due to the gradational nature of the boundary between the "low" and "high" grade domains, boundary analysis was undertaken. As illustrated in Graph 19, the contact between the "low" and "high" grade zones was treated as a "soft" boundary. A transition of 5.0m was used to flag samples from the "high" into the "low" grade zones. The boundary was treated as "hard" from "low" into "high" grade.



Graph 19: “High” and “Low” Grade Boundary Analysis

Zonal flagging in Datamine uses a field called Kzone to distinguish the different grade domains during geostatistical analysis and estimation (Figure 37).

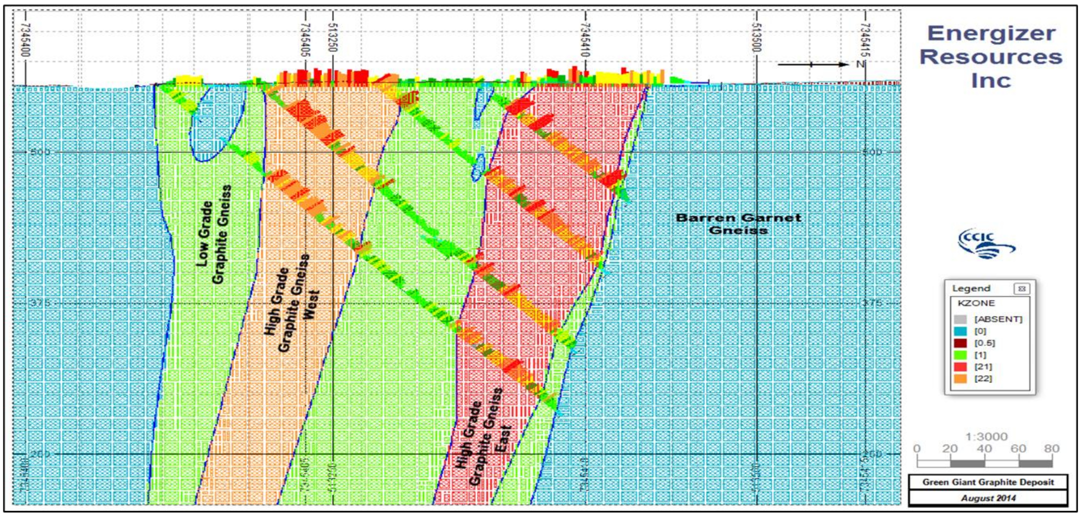


Figure 37: Cross-Section Showing the Kzone Flagging in the Block Model

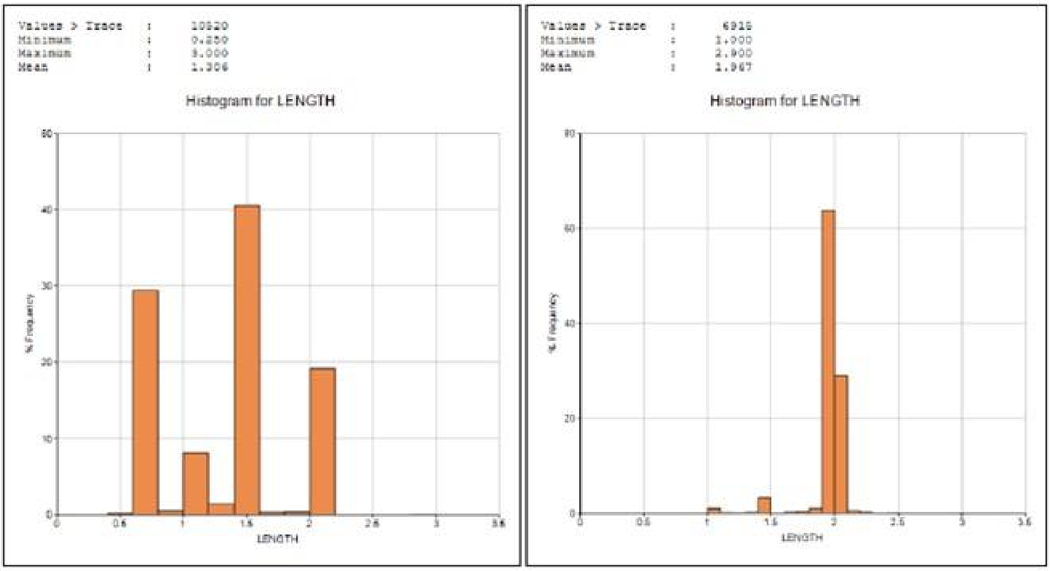
A summary of the Kzone flagging undertaken in Datamine is provided below in Table 31.

Table 31: Summary of Kzone Flagging in Datamine

Zone Description	Kzone Value	Code Name
Garnet Gneiss Barren Zone	0	WST
Overburden	0.5	OVB
Low Grade Mineralised Zone	1	LG
High Grade Mineralised Zone East	21	HG-E
High Grade Mineralised Zone West	22	HG-W

14.3 Compositing

The predominant sampling interval was either 0.5m, 1.0m, 1.5m, or 2m, and hence a composite length of 2m was used to include all the samples, (Graph 20). Compositing used the Kzone to ensure that samples were composited within the different domains. Any samples less than 0.5m after compositing were excluded from the geostatistical analysis and estimations.



Graph 20: Histogram of Sample Length Prior to and After Compositing

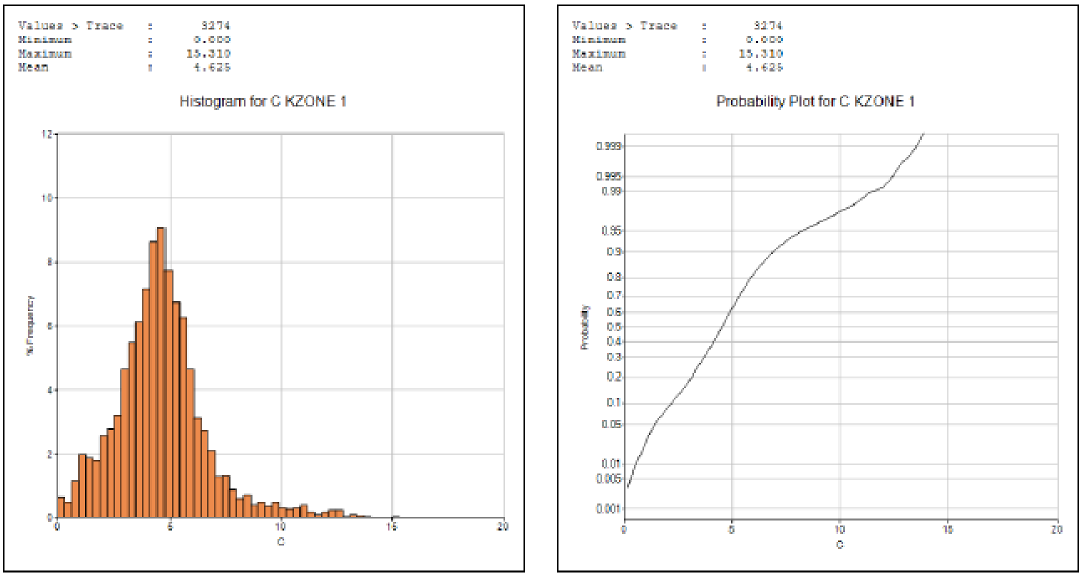
14.3.1 Composited Statistics

The sampling protocol for core intervals deemed to be barren with respect to graphite mineralisation is due to not been submitted for analyses. All un-sampled intervals have, therefore, been set to trace prior to compositing and statistical analysis. A statistical summary for the four grade domains is presented in Table 32 below.

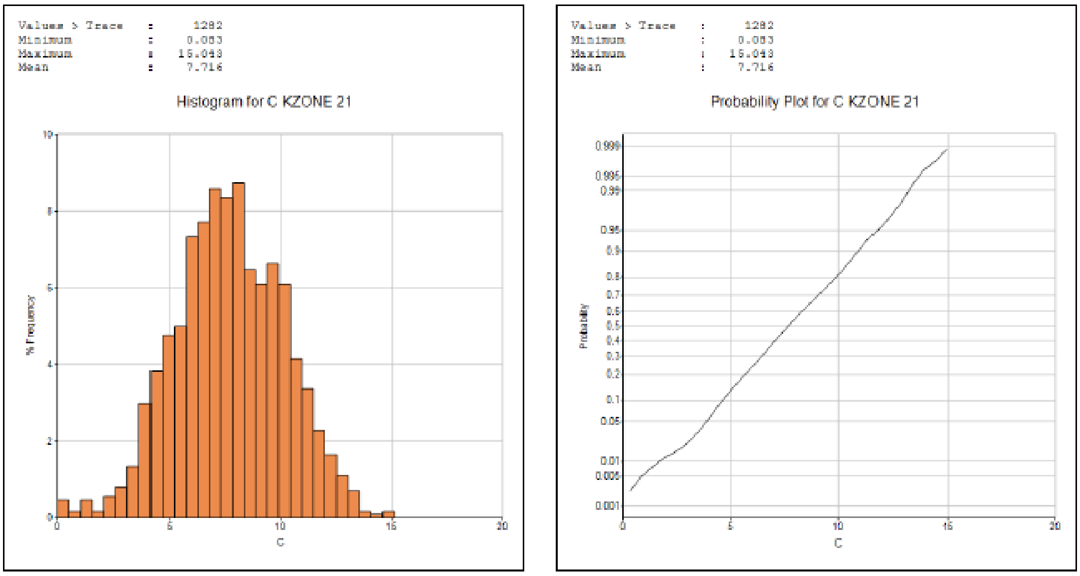
Table 32: Statistical Summary of C% Per Kzone

Kzone	0	1	21	22
Field	C	C	C	C
Numtrace	1,730	3,274	1,282	2,359
Minimum	0.00	0.00	0.08	0.13
Maximum	15.55	15.31	15.04	15.69
Mean	2.07	4.63	7.72	8.30
Variance	3.10	4.15	5.92	6.97
StandDev	1.76	2.04	2.43	2.64
StandErr	0.04	0.04	0.07	0.05
Skewness	1.88	0.91	-0.06	-0.37
Kurtosis	5.47	2.36	-0.07	-0.09
Logestmn	2.26	5.77	7.93	8.51
CoV	0.05	0.44	0.32	0.32
*StandDev = Standard Deviation *CoV = Co-efficient of Variation				

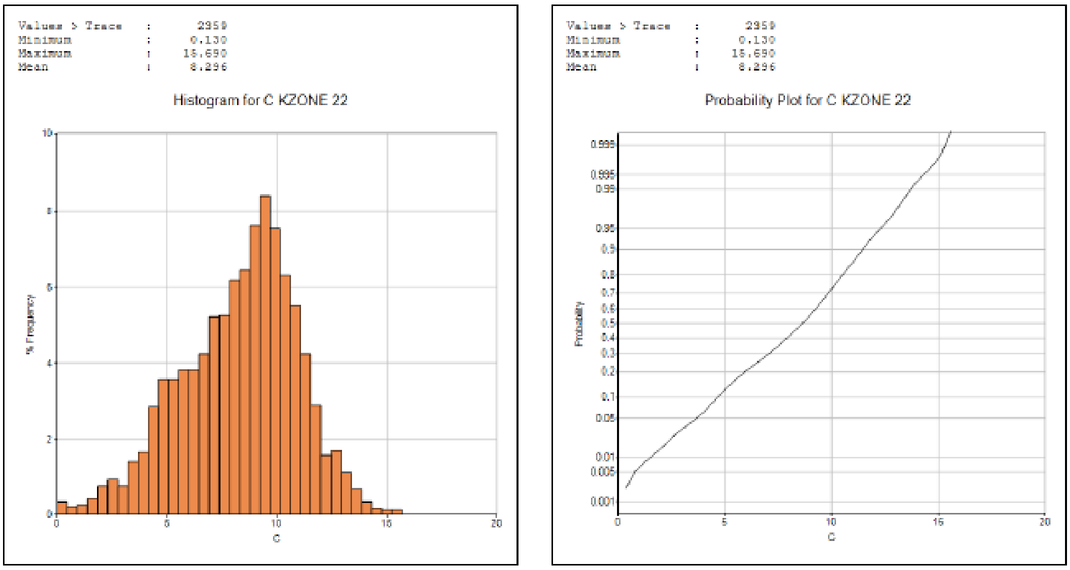
The mean of the samples for “LG” (Kzone 1) domain is 4.63% C, with a positive grade tail up to 15.31% C. The “HGE” (Kzone 21) domain has a mean value of 7.72% C, Co-efficient of Variance (“CoV”) of 0.32 and a maximum of 15.04% C. The “HGW” (Kzone 22) domain has a mean value of 8.30, CoV of 0.32 and maximum value of 15.69% C. Histograms of sample distributions for both low grade and high-grade domains are presented in Graph 21, Graph 22 and Graph 23 below. The sample distributions for both the “high grade” domains exhibit very similar statistical characteristics.



Graph 21: Sample Distributions for the “Low Grade” Domain



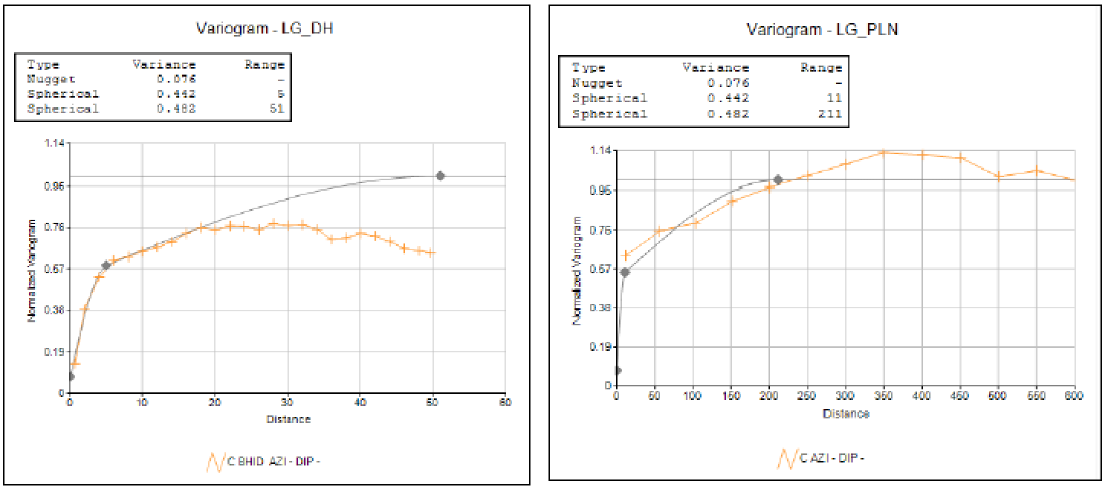
Graph 22: Sample Distributions for the “High Grade East” Domain



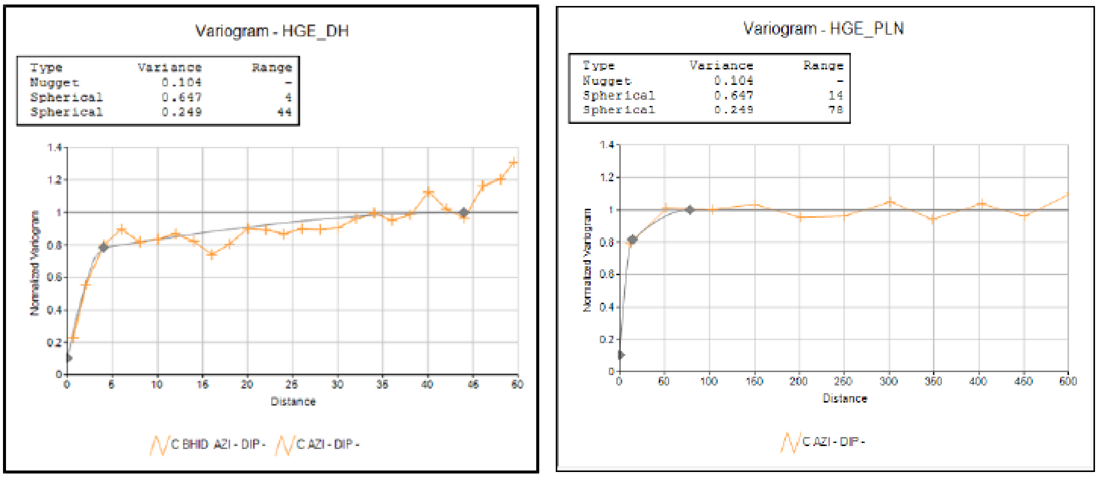
Graph 23: Sample Distributions for the "High Grade West" Domain

14.4 Variography

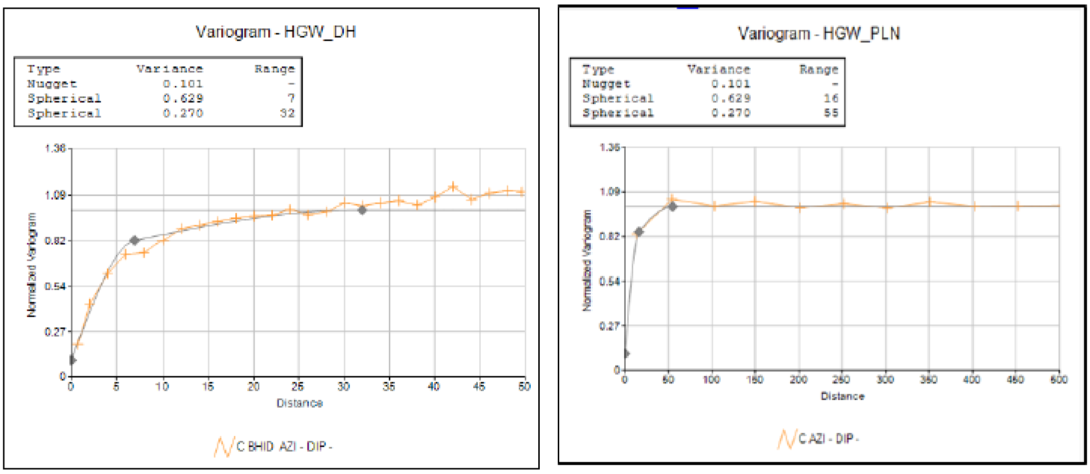
Variogram analysis and modelling was done using Datamine. Variogram models were generated for % C, for the "low grade", "high grade" east, and "high grade" west domains. The down hole semi-variograms together with their respective Isotropic models are illustrated in, Graph 24, Graph 25 and Graph 26. The nugget for the 'low grade" domain is 10%, with more than 90% of the spatial variance occurring within the first 120m. The "high grade" east domain has a nugget of 10% and a range that extends to 80m. The "high grade" west domain has a nugget of 10% and a range of 55m. During the grade estimation the isotropic variogram parameters were used in the strike and dip directions, with the down hole parameters resembling the across strike direction.



Graph 24: Variogram model for "Low Grade" domain – down hole and isotropic



Graph 25: Variogram model for "High Grade East" domain – down hole and isotropic



Graph 26: Variogram Model for "High Grade West" Domain – Down Hole and Isotropic

14.5 Top Capping

The top capping strategy considered various criteria to determine the optimum values. These included:

- Histograms of sample distributions.
- Sample percentiles.
- Spatial locations of "outlier" samples.
- Validation of model estimates against samples.

A summary of the top capping values that were applied are presented in Table 33 below. All samples that were greater than the top capping value were re-set to the top capping value.

Table 33: Summary of Top Capping Values

Domain	Capping -% C	Percentile	Number of Samples
“LG”	14% C	99.9%	4
“HGE”	15% C	99.9%	2
“HGW”	15% C	99.9%	3

14.6 Grade Estimation

14.6.1 Method

The method of estimations for % C was Ordinary Kriging. Estimations were undertaken using the Estima process in Datamine. Parameters for estimations, (i.e. Parent Block size, search distances and the number of samples to be used for an estimate), was optimised using Kriged Neighbourhood Analysis, which is explained in more detail below.

14.6.2 Kriged Neighbourhood Analysis

The aim of Kriged Neighbourhood Analysis is to determine the optimal theoretical search and estimation parameters during Kriging so as to achieve an acceptable Kriging Variance and Slope of Regression (“SOR”), whilst ensuring that none, or a minimal number of samples are assigned negative Kriging Weights. Once this is determined, practicality is taken into account when deciding on the parameters to be used. This optimisation was based on a representative area within the deposit, using the “high grade” west domain, because this is the most prominently mineralised domain. Figure 38 indicates the test location as a block dot with the drill holes coloured on % C values. The following parameters in chronological order were optimised:

- Optimum parent cells size for the block model, in X, Y and Z directions.
- Optimum search distances in the X, Y and Z directions together with determining the appropriate minimum and maximum number of samples required for a reliable estimate.

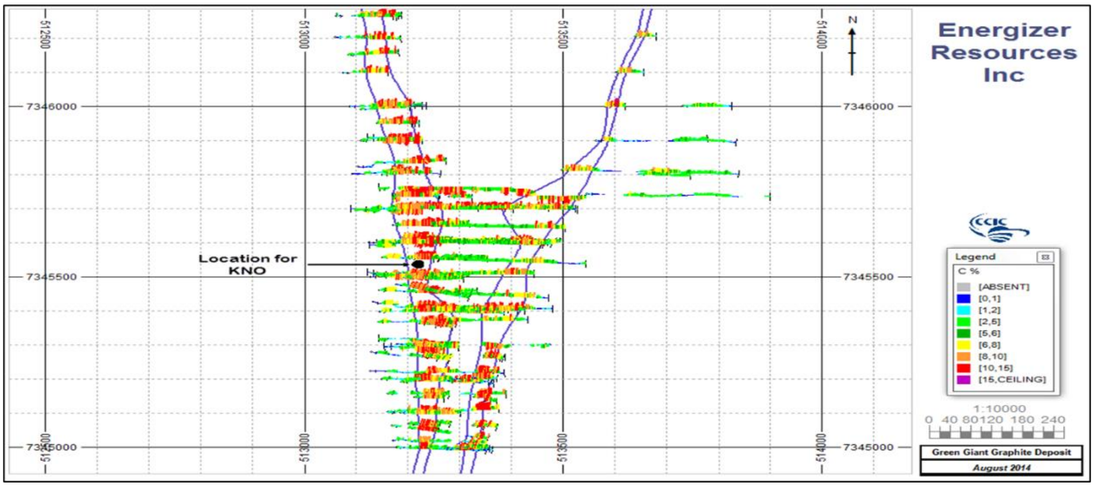
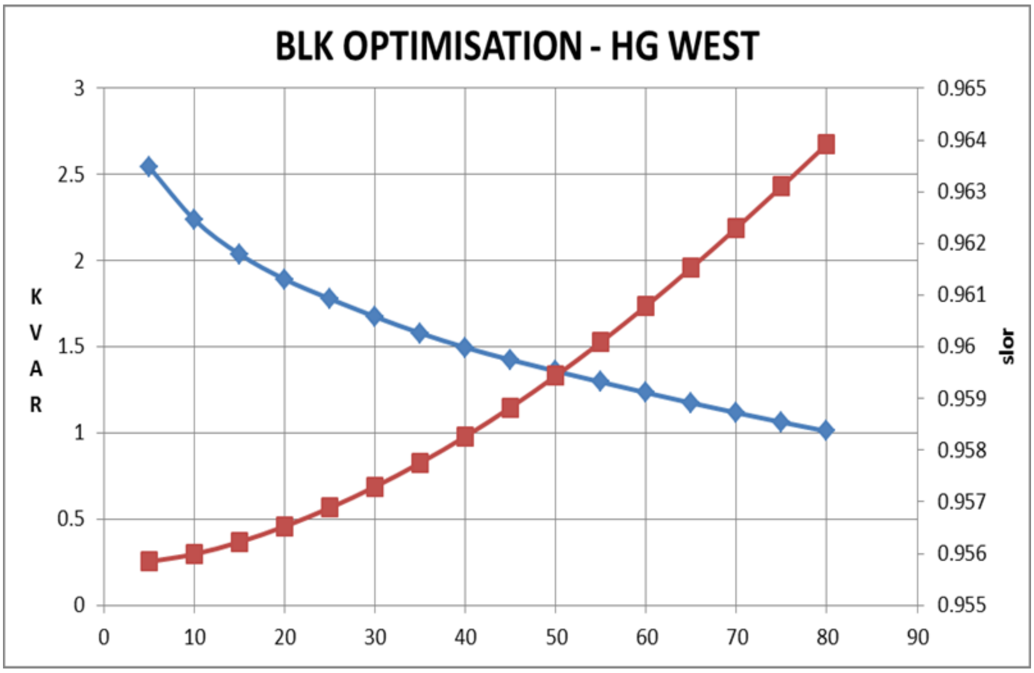


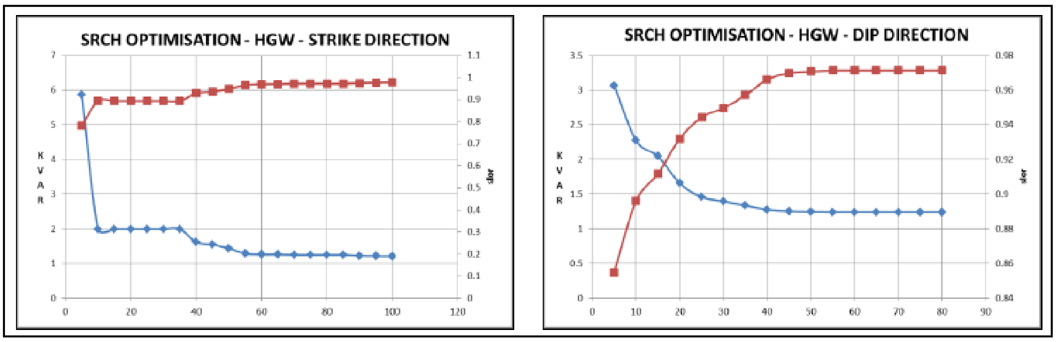
Figure 38: Plan Showing the Location for the Krige Optimizations

During parent cell size optimisation, the approach taken was to use the variogram ranges as a guide to set the search distances, and not to apply any minimum and maximum number of samples to be used. A default parent cell size of 10m x 10m x 10m was selected using the borehole spacing as a guide. Whilst all parameters remain constant, the parent cell size in the Y direction was set to 5m and then incrementally increased for every estimation. The Estimated Grade (% C), Kriging Variance ("KVAR"), Block Variance ("BVAR"), Kriging Efficiency ("KEF"), SLOR, Number of Samples used ("NUMSAM"), and the Percentage of samples with negative Kriging weights ("PCNEGWTH"), are recorded for every estimation. These outputs are then plotted against the incremental cell sizes to determine the optimum cell size in the Y direction. The same process is then repeated for the cell sizes in the X direction. Graph 27 compares the plot of the incremental parent cell sizes against the KVAR for the Y direction. The optimum parent cell size, taking practical mining constraints into consideration, was set to 40m x 10m x 10m in the X, Y and Z directions respectively.



Graph 27: Parent cell size optimizations

During search optimisations, the parent cell size was set to 10m x 40m x 10m and as described above, search distance in the X direction was incrementally increased. The results of the search optimisation are plotted in Graph 28, which compares both KVAR and SLOR against search distances. Attention is also given the number of samples per estimate and the percentage of samples with negative Kriging weights. The optimum search distance, taking borehole spacing into account, was set as 70m x 40m x 10m in the X (strike), Y (dip) and Z (across strike) directions respectively. A minimum of 5 and maximum of 100 samples was used per estimate and was limited to five samples per drill hole.



Graph 28: Search Distance Optimizations

14.6.3 Model Construction and Parameters

A block model was constructed using a parent cell size of 10m x 40m x 10m in the X, Y and Z directions, allowing sub-cell splitting to ensure that the volumes of the graphite mineralised zones were represented. Zonal control was applied during grade estimation with each grade domain in the block model assigned a unique Kzone number, as described under Item 14.2.1 (Grade Domaining) above. A section illustrating the block model colour coded by Kzone is shown in Figure 39. "Waste" domains have been assigned and estimated for indicative purposes only and have not been reported in the mineral resource statement.

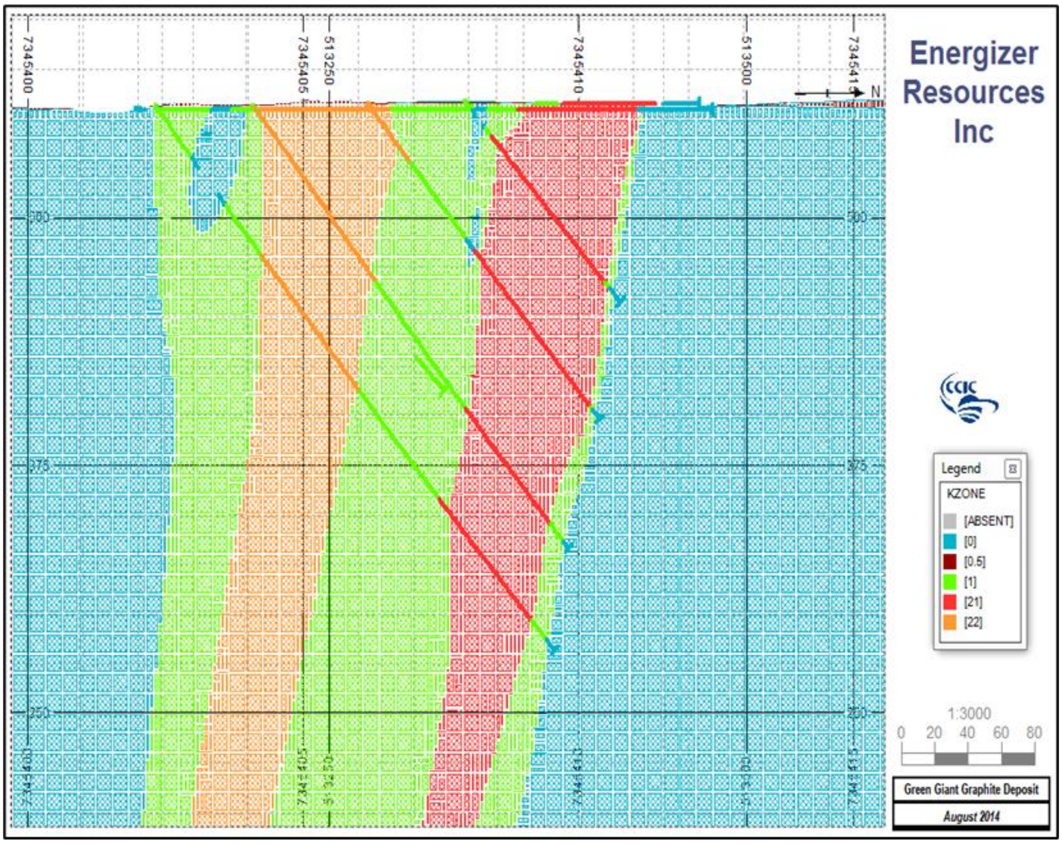


Figure 39: Section Showing Block Model Coding

14.6.4 Kriging Parameters

The method for estimating % C was ordinary Kriging. Zonal control was applied, thereby, ensuring that samples from a particular domain were constrained to estimating grades only into the block model for that particular domain. The boundaries between the waste and low grade domains were treated as hard boundaries because it is a distinct lithological contact. Boundaries between the low grade and high grade domains were treated as a soft boundary due to their gradational nature.

Parental cell estimation was used. An expanding search ellipse allowed for cells that were not estimated with the minimum estimation parameters, to be estimated. A second search ellipse was expanded by two times. The following fields are recorded in the estimated block model:

- C – Ordinary Kriged estimate for % C.
- KVAR – Estimated Kriging Variance for % C.
- SVOL – Flag to identify which of the two search ellipses was used for the % C estimate.
- NUMSAM – Records the number of samples used to estimate % C.

14.7 Model Validation

Model validation included the following:

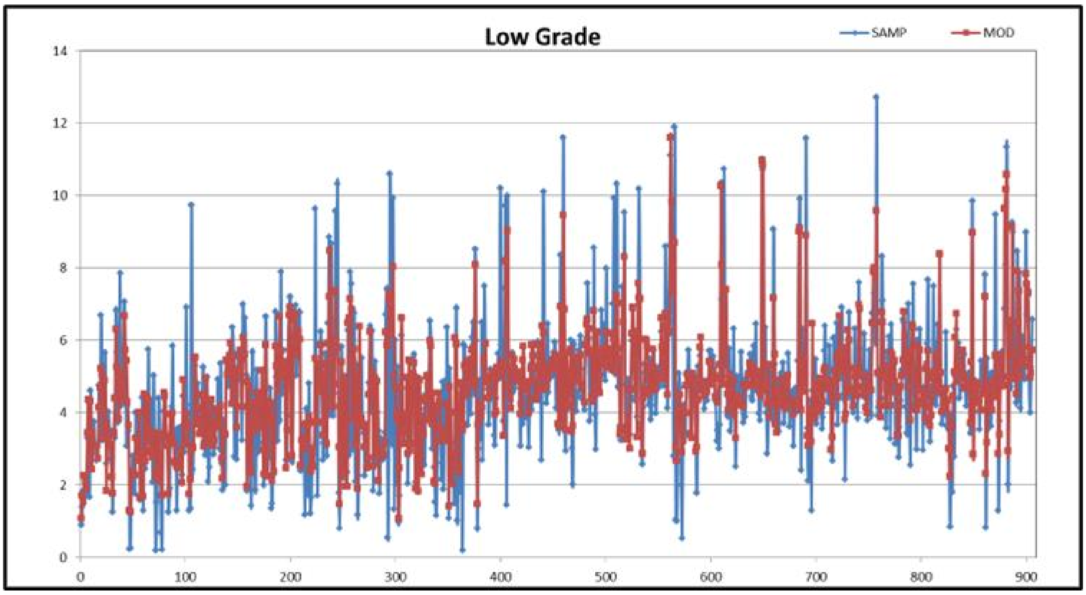
- Visual comparisons of the estimated grades against the composite sample grades.
- Statistical comparisons for the mean of estimated grades against the mean of the composited samples.
- Trends, (or swath analysis checking), to ensure that the regional grade trends from the drill holes are present in the model. To reduce the estimation errors ordinary Kriging tends to have a smoothing effect on the estimates. The objective of this exercise is, therefore, to ensure that both regional and local trends are best preserved.

A statistical comparison between the composited samples and the model estimates is presented in Table 34 below. The mean of the samples is weighted by length, while the model estimates are weighted by volume. The means between sample and model estimates compares favourably.

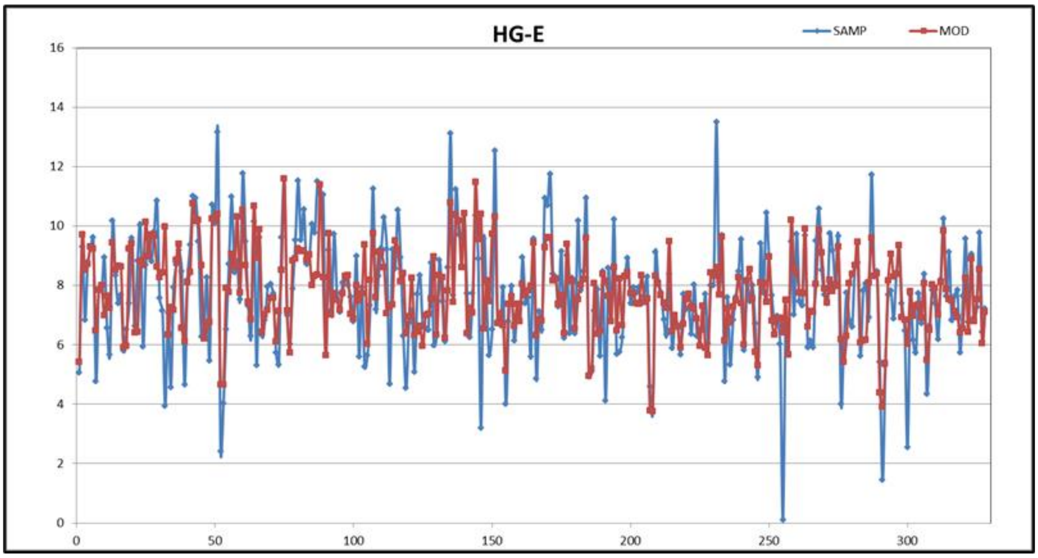
Table 34: Statistical Comparison between Composited Samples and Model Estimates

Sample Composites				Model Estimates		
Kzone	LG	HG-E	HG-W	LG	HG-E	HG-W
Field	C	C	C	C	C	C
N Samples	3,312	1,269	2,359	56,940	21,348	16,534
Minimum	0.00	0.08	0.13	0.20	0.74	3.85
Maximum	14.00	15.00	15.00	11.60	11.92	11.69
Mean	**4.68**	**7.75**	**8.36**	**4.58**	**7.56**	**8.17**
Variance	4.41	5.74	6.73	1.71	1.64	1.55
StandDev	2.10	2.40	2.59	1.31	1.28	1.25
Skewness	0.95	-0.00	-0.36	0.39	-0.24	-0.17
Kurtosis	2.20	-0.09	-0.10	1.51	1.40	-0.19
CoV	0.45	0.31	0.31	0.29	0.17	0.15

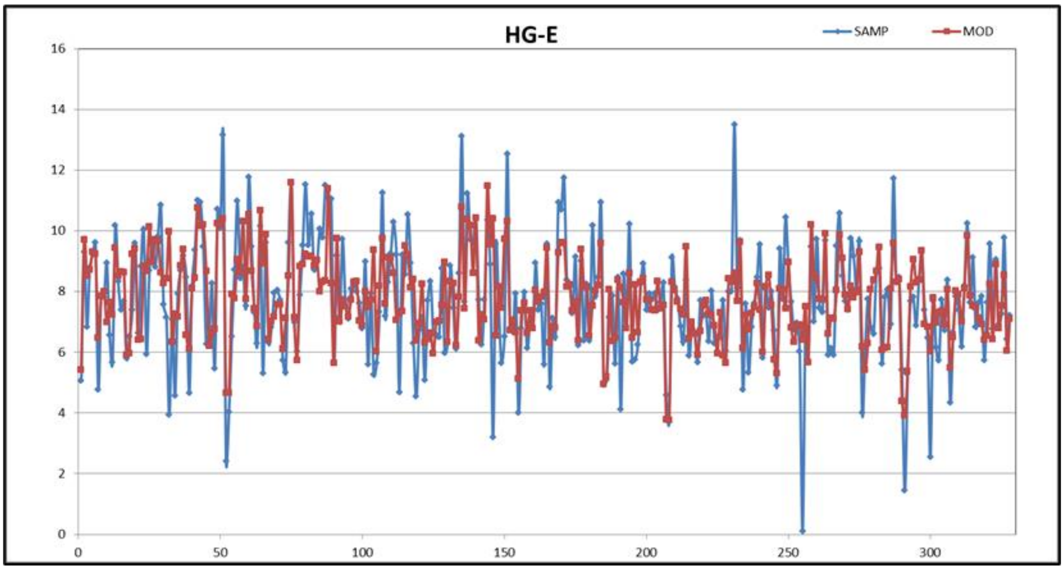
Block on block analysis compares local trends in the samples against model estimates. The approach here was to divide the study area into 10m x 40m x 10m blocks in the X, Y and Z direction respectively and to select samples within each block, and compares their mean against the mean of the model estimates within that same block, per KZONE. Plots comparing the mean of the samples (blue), and mean of the estimates (red), for the high grade domains are illustrated in Graph 29, Graph 30 and Graph 31. The mean of the estimates is smoother and less variable than that of the samples, while preserving the grade trends of the samples.



Graph 29:Trend Analysis plot for %C, “low” grade domain



Graph 30:Trend Analysis plot for %C, “High Grade East” domain



Graph 31: Trend Analysis plot for %C, "High Grade West" domain

14.8 Grade Distribution Plots

Figure 40 is a section illustrating the distribution of the block model super imposed on the drill holes. There is a good correlation between the grades in the drill holes and that of the block model.

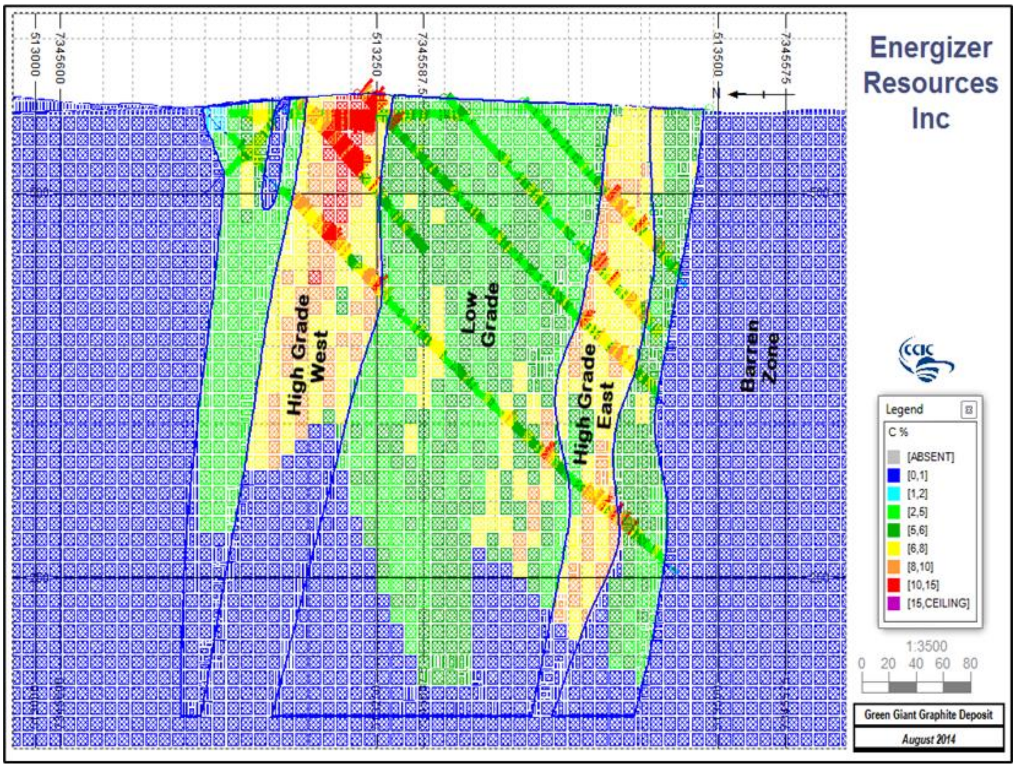
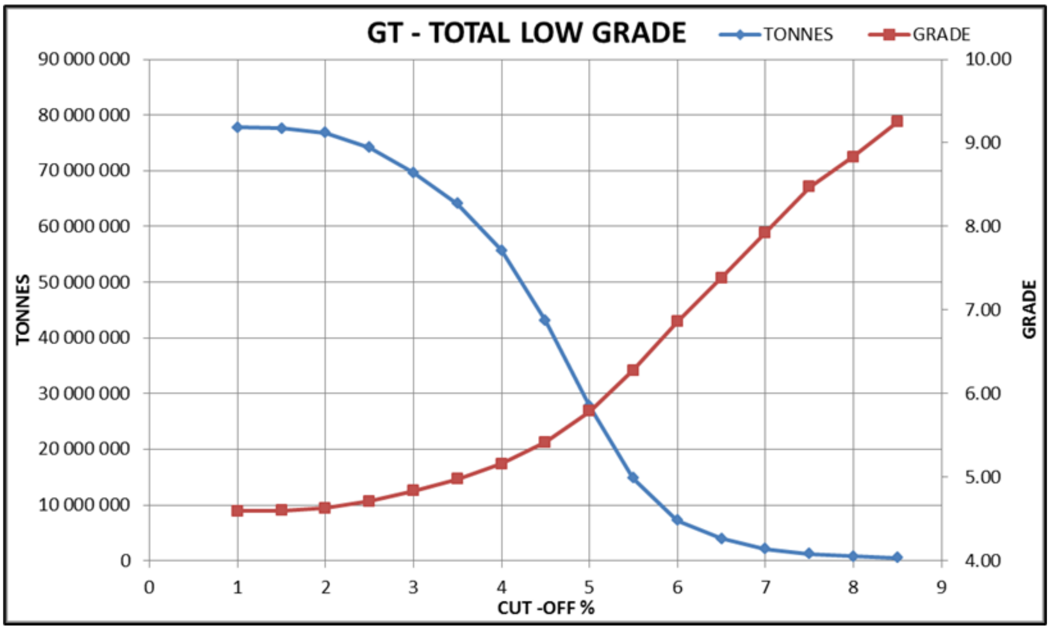
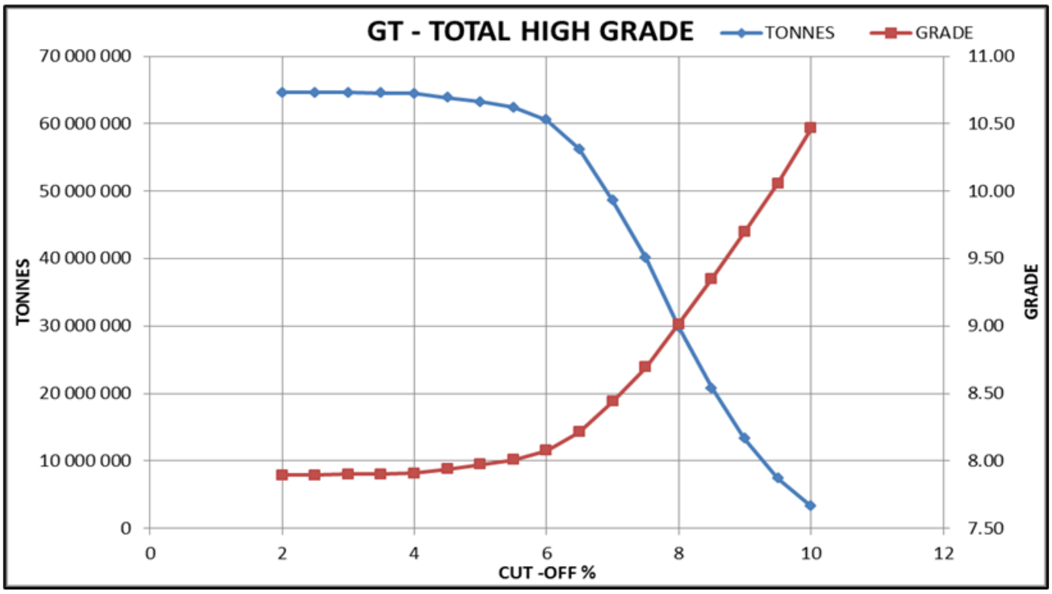


Figure 40: Section Showing C (%) Grade Distribution in the Model

The grade tonnage curves for the "low" and "high" grade domains are presented in Graph 32 and Graph 33. For the "low" grade domain, there is very little variation below a 2% C cut-off, thereafter there is a consistent drop in tonnages and corresponding increase in grade, up to 6.5 % C. For the "high" grade domain, there is very little variation below 4% C, with the majority of this domain occurring between the 6.0% to 10.0% C range.



Graph 32: Grade Tonnage Curve for the "Low" grade domain



Graph 33: Grade Tonnage Curve for the "High" grade domains

14.9 Resource Classification

The definition of a mineral resource according to the CIM reporting code is:

- A Mineral Resource is a concentration, or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including, base and precious metals, coal, and industrial minerals in/or on the earth's crust in such form and quantity and of such a grade, or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics, and continuity of a Mineral Resource are known and estimated, or interpreted from specific geological evidence and knowledge.
- Mineral Resources are sub-divided, and must be so reported, in order to increase confidence in respect of geoscientific evidence, into Inferred, Indicated, or Measured categories.

Based on the geological data and information presented in this report, there is sufficient information about the location, shape, size, geological characteristics, and continuity of the deposit to declare a resource.

QA/QC protocols and results indicate an acceptable level of confidence in the analysis of the samples for these drill holes and trenches. There is also a reasonable correlation between drill hole and trench data.

Drill hole spacing varies from 100m by 100m in some areas, to 50m by 50m in other areas especially within the first five years of mining footprint. The well-informed areas provide adequate geological confidence to place the resources into the Measured Category. Hence, the resource classification methodology (Figure 41 and Figure 42) was based on the following criteria:

- Areas with drill spacing less than 50m by 50m, was considered for Measured Resources.
- Areas with drilling of 50m along dip and 10m along strike, was considered for Indicated Resources.
- Areas within the 100m by 100m drill spacing, was considered for Inferred Resources.

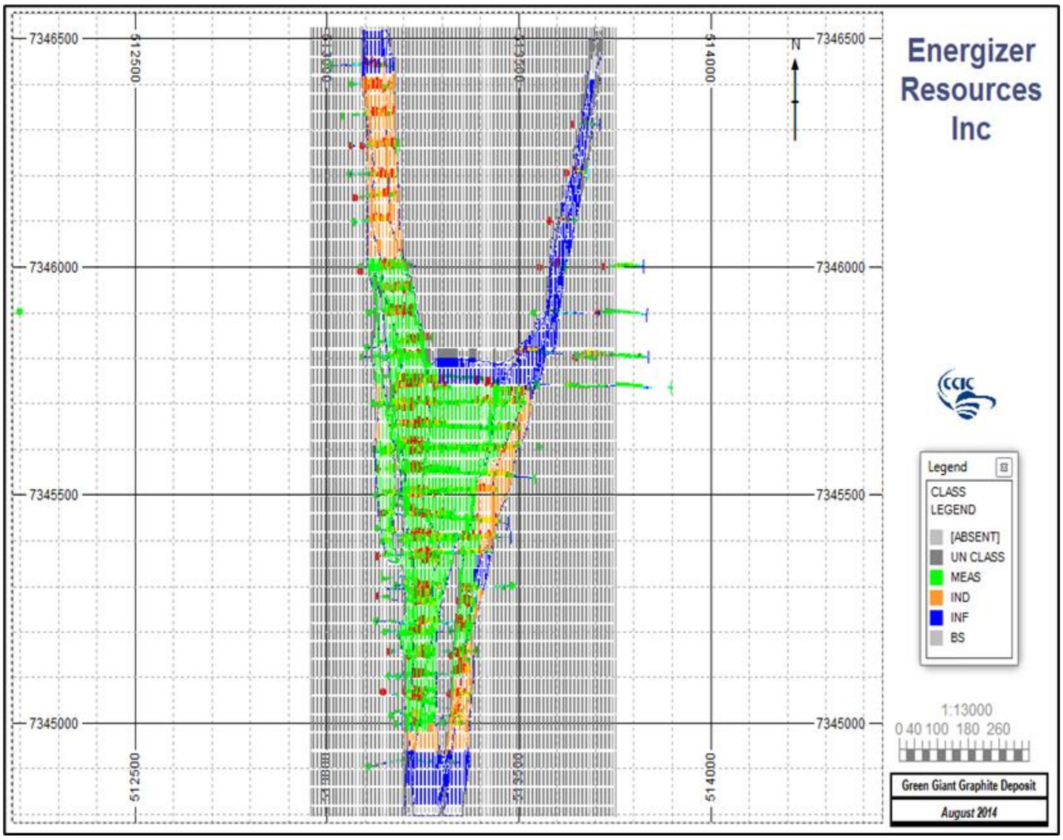


Figure 41: Plan showing Mineral Resource Classification

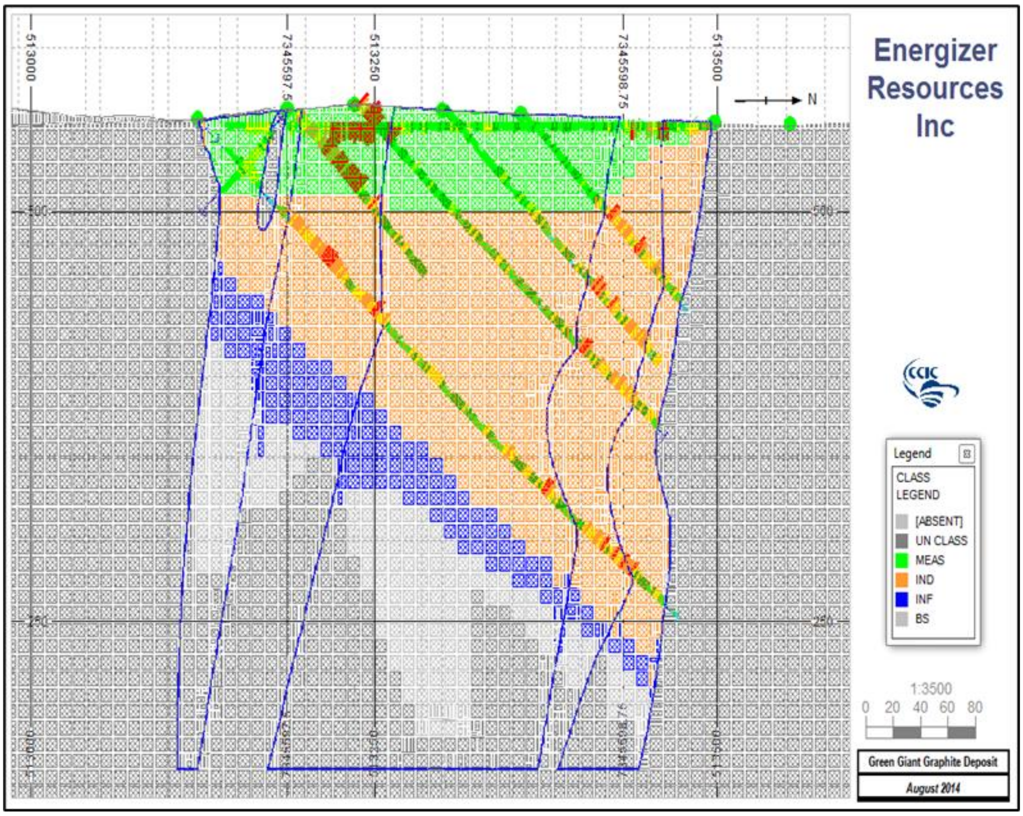


Figure 42: Section showing Mineral Resource Classification

14.10 Mineral Resource Estimate

The current surface mineable mineral resource estimate for Molo is summarised in Table 35 below. A RD of 2.36 t/m^3 was assigned to the mineralised domains. The mineral resources are classified as Measured, Indicated and Inferred categories according to the CIM Definition Standards. The cut-off grade of 2.0 % C was calculated using a graphite price of US$1,400.00/t, mining cost of US$5.75/t and processing cost of US$17.00/t. Resources within the low grade domain are stated at a 2 % C cut-off. The high-grade domains which contain very little material below 4% C (Table 35) is stated at a 4% C cut-off. Whilst the "high grade" resources occur within the "low grade" resources, they are estimated and reported separately. The total Measured and Indicated Resources is estimated at 100.37 Mt, grading at 6.27% C. Inferred Resources is at 40.91 Mt, grading at 5.78% C. The effective date of the mineral resource estimate as at September 1, 2014. Risks which may reasonably affect the mineral resource estimate are related to ongoing exploratory drilling to increase mineral resource confidence of the Inferred Resources (i.e., exploration risk). There has been trail mining of 520t of production between the effective date of the mineral resource estimate (September 1, 2023) and the effective date of the mineral reserve estimate. This production is considered immaterial to the mineral resource estimate.

When compared to the November 2012 resource statement, this shows a 13.7% increase in tonnage 3.4% decrease in grade and 9.8% increase in graphite content. The reason for the increase in tonnage is due to additional exploratory drilling on the north-eastern limb of mineralisation. 23.62 Mt, grading at 6.32% C having been upgraded by infill drilling into the Measure Resource category.

Table 35: Surface Mineable Mineral Resource Statement for the Molo Graphite Deposit – September 1, 2023

Classification	Material Type	Tonnes (t)	Grade - C%	Graphite - T
Measured	"Low-Grade"	13,048 373	4.64	605,082
Measured	"High-Grade"	10573 137	8.4	887,835
Total Measured		**23,621,510**	**6.32**	**1,492,916**
Indicated	"Low-Grade"	39,539,403	4.73	1,871,075
Indicated	"High-Grade"	37,206,550	7.86	2,925,266
Total Indicated		**76,745,953**	**6.25**	**4,796,341**
Measured + Indicated	"Low-Grade"	52,587,776	4.71	2,476,157
Measured + Indicated	"High-Grade"	47,779,687	7.98	3,813,101
Total Measured + Indicated		**100,367,464**	**6.27**	**6,289,257**
Inferred	"Low-Grade"	24,233,267	4.46	1,080,677
Inferred	"High-Grade"	16,681453	7.70	1,285,039
Total Inferred		**40,914,721**	**5.78**	**2,365,716**

C% = carbon percentage; Graphite – T = Tonnes of graphite

Notes:

1. Mineral Resources are classified according to the Canadian Institute of Mining definitions.
2. Mineral Resources are reported Inclusive of Mineral Reserves.
3. “Low Grade” Resources are stated at a cut-off grade of 2% C.
4. “High grade” Resources are stated at a cut-off grade of 4% C.
5. Eastern and western high-grade assays are capped at 15% C.
6. A relative density of 2.36 t per cubic meter (t/m^3) was assigned to the mineralized zones for the resource tonnage estimation.
7. Totals may not represent the sum of the parts due to rounding.
8. Mineral Resource are defined as surface mineable only.
9. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that any mineral resource will be converted into a mineral reserve.
10. % Cg = percentage Carbon Graphite.

15 MINERAL RESERVES ESTIMATE

15.1 Executive Summary

The section outlines the criteria for reporting Proven and Probable Reserves, discusses the application of a 3% cut-off grade for reserves compared to a 4.5% cut-off grade applied in the previous LoM done by Sound Mining. It also provides insight into the block size and Ore Class flags in the resource model.

The total ROM reserves reported are 53,746 kt with a contained graphite content of 3,306 kt.

15.2 Proven and Probable Reserves

Mineral Reserves for the Molo Project are reported as per NI 43-101 requirements. Only measured and indicated resources can be reported as proven, or probable reserves. For technical and economic reasons, only Measured and Indicated resources that are inside the designed final pit and that are equal to/or above the grade cut-off of 3% C are reported as proven and probable reserves. Inferred resources (4.5% of total ROM ore) are not included in the reserve's declaration, even though they are included in the LoM production schedule since NextSource Materials must mine through them to extract the entire pit shell. All other measured and indicated resources that are below the grade cut-off are also not included in the reserve statement as it is planned to be stockpiled separately as waste material.

15.3 Cut-Off Grades

The cut-off grade of 3% is enforced simply to improve the economics of the entire Project. The Inferred resource inside the final designed pit and included in the LoM Production Schedule is 2.52 Mt at an average grade of 5.51% C head grade, this is not included in the reported reserves. This is non-material and is an insignificant amount of the overall ROM ore tonnes (4.5%) and only 4.03% of contained graphite. See section 16 for a graphical illustration on the % of inferred material being mined per year.

It is an iterative process to determine the optimal cut-off grade. A cut-off grade of 4.5% was initially selected and Whittle software was used to determine the optimal pit shell for this scenario with the best DCF.

Extensive work was done by Deswik lately and a more optimal pit shell was developed in the Deswik Pseudoflow software and at a 3% Carbon cut-off grade was selected.

Table 36: Pseudoflow Shell (3% C Cut-off) vs Whittle Shell (4.5% C Cut-off) – Ore / Waste comparison

Reference	Total Tonnes	Waste Tonnes	Ore Tonnes	Grade	Economic Grade cut-off	Graphite Tonnes
PH2 PEA Table37 Pit Optimisation Results Opt 2 (SMS 2022).	109,891,920	45,886,570	64,005,350	6.16%	4.50%	3,575,170
Deswik CAD Final Design.	73,816,140	17,197,640	56,618,500	6.28%	3.00%	3,131,120

It will be expected that more ore will be mined when a lower cut-off grade is selected at a lower average grade, but due to a more practical pit selection there are 11% less ore in this optimized pit shell, but at a slightly higher grade with 62% less waste to be mined, resulting in only 12% less product.

A more practical mineable pit was designed in the Deswik Software (refer to Figure 43, Figure 44 and Figure 45 below and more detail in section 16 of this document.

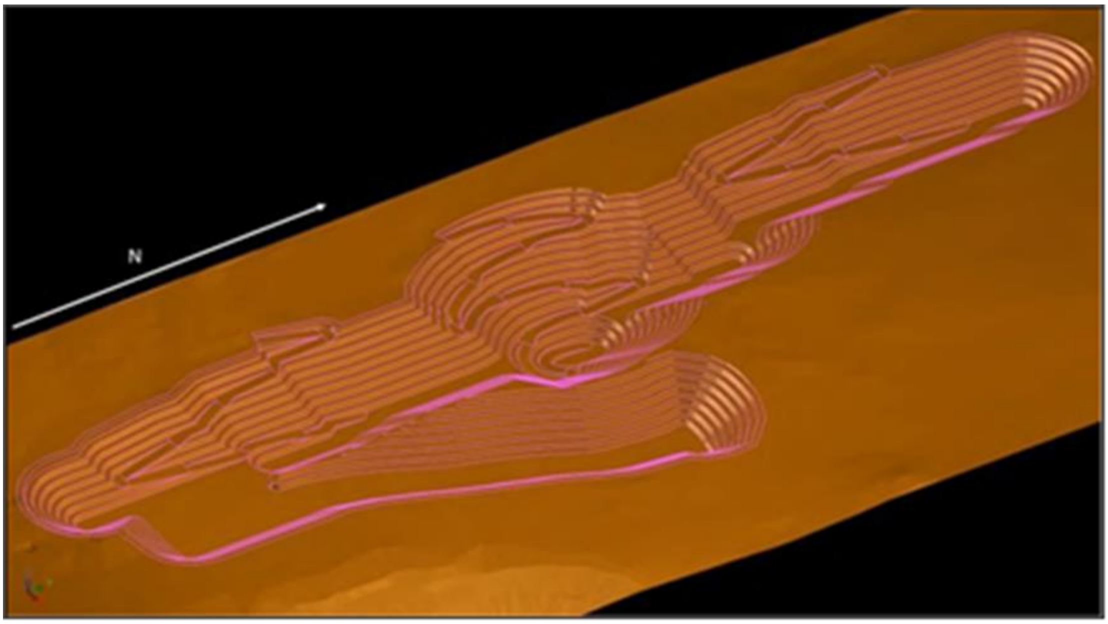


Figure 43: Whittle Final Pit Shell

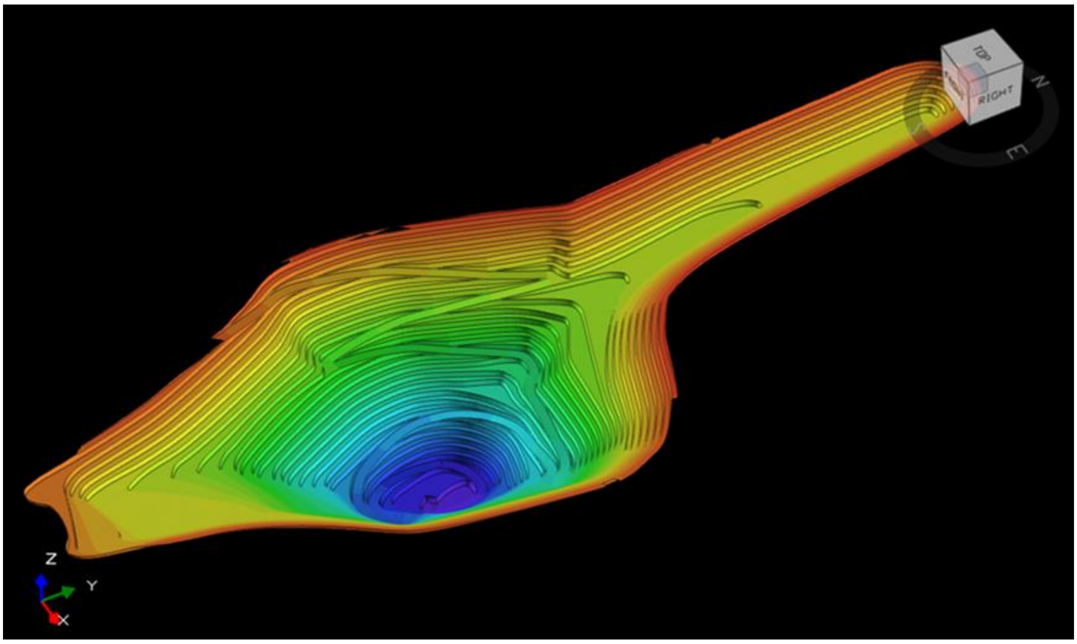


Figure 44: Deswik Pseudoflow Optimised Pit Shell

The reserves are split into high-grade and low-grade reserves. The high grade is >5% C and the low grade ≥3% C and ≤5% C. Refer to Table 37 below. One ROM ore tonne equals one in-situ tonne minus 5% losses plus 3% contamination @ 0% C. The grade reported is the diluted head grade in the LoM production section in section 16.

It must be noted that the 5% losses applied is very conservative since this is a massive orebody. What is lost on one bench will be loaded on the next and the actual losses will therefore be less in reality. Real losses will be incurred on the peripherals of the pit and at the floor of the bottom bench. The 3% contamination / dilution applied has 0% C which will also not be true since the dilution comes from other benches with grade. A very conservative stance has been taken here in the calculation of contained graphite.

Table 37: Reserve Statement

Molo Reserve Statement				
Reserve Statement - 1 September 2023				**Contained Graphite in Plant Feed Tonnes**
Classification	**Material Type**	**ROM Ore Tonnes (kt)**	**Grade – Cg% Head Grade**	**Graphite Tonnes (kt)**
Proven Reserves	High Grade	15,489	7.00	1,085
	Low Grade	5,845	4.25	248
	Total	**21,334**	**6.25**	**1,333**
Probable Reserves	High Grade	24,734	6.64	1,642
	Low Grade	7,677	4.32	331
	Total	**32,412**	**6.09**	**1,973**
Grand Total		**53,746**	**6.15**	**3,306**

Notes:

1. Mineral Resources are classified according to the Canadian Institute of Mining definitions.
2. Mineral Resources are reported Inclusive of Mineral Reserves.
3. "Low Grade" Resources are stated at a cut-off grade of 2% C.
4. "High grade" Resources are stated at a cut-off grade of 4% C.
5. Eastern and western high-grade assays are capped at 15% C.
6. A relative density of 2.36 t per cubic meter (t/m^3) was assigned to the mineralized zones for the resource tonnage estimation.
7. Totals may not represent the sum of the parts due to rounding.
8. Mineral Resource are defined as surface mineable only.
9. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that any mineral resource will be converted into a mineral reserve.
10. % Cg = percentage Carbon Graphite.

15.4 Reserve Statement Clarification

The Geological Model (Resource Model) is based on the "finmod_v4 data file" and was set up in 10m x 10m x 10m blocks. For each block, an Ore Class flag was allocated to the block, i.e. Ore Class = 1 is a "measured resource", Ore Class = 2 is an "indicated resource" and Ore Class 3 is an "inferred resource".

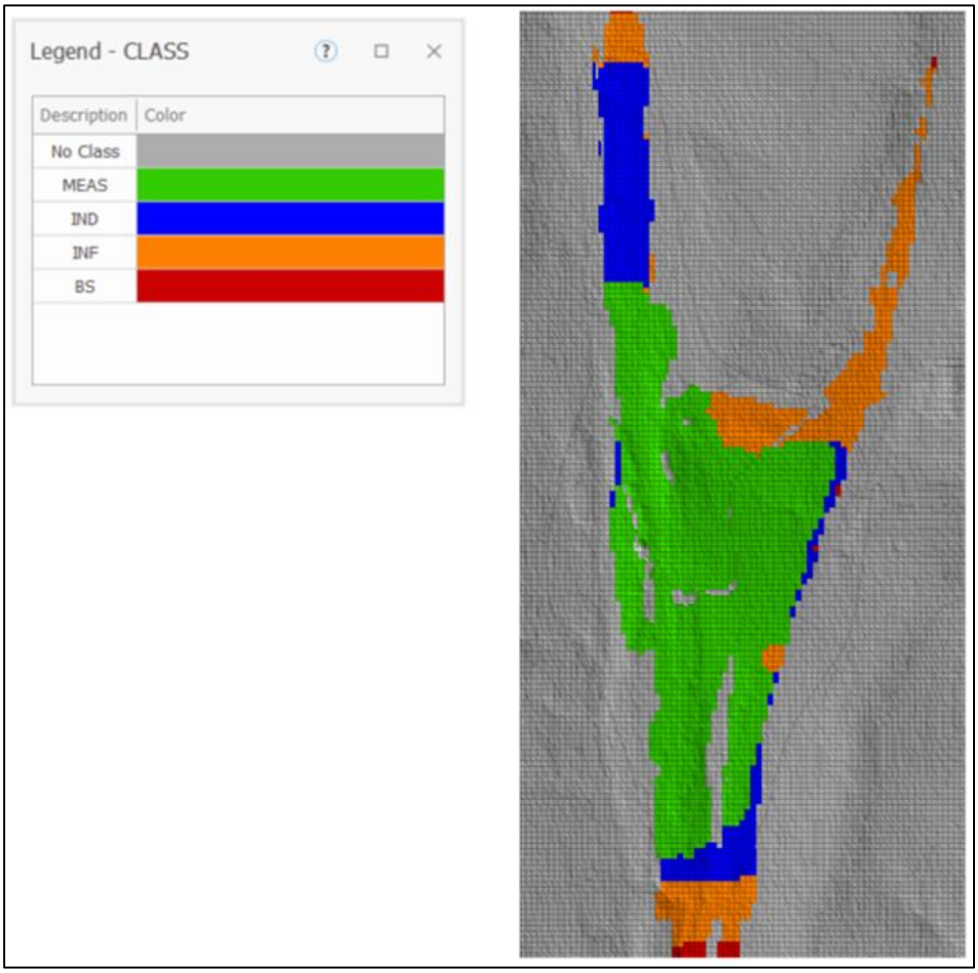


Figure 45: Resource Model Ore Classification

A practical mining block of 30m x 30m x 8m (2 x 4m lifts) high was selected to suit the operation of the selected mining equipment on site and the mining block model is therefore not similar to the 10m x 10m x 10m block size of the resource model. This results that one mining block may have measured, indicated and sometimes a small portion of inferred resources in it, (almost 9 resource blocks per mining block). Careful interrogation was done in Deswik Software to report each class or category of ore extracted correctly. A mining block is, therefore, not categorized as a certain ore class, but rather the tonnes in a mining block are reported per ore class to ensure the reserves are reported correctly.

16 MINING METHODS

16.1 Introduction

The expansion projects includes the mining expansion, increased processing facility and associated infrastructure capable of processing 2,640,000 tpa of ore producing approximately 150,000 tpa of graphite concentrate over the LoM.

The Company has every intention to develop the Expansion Project in close succession to the completion of Phase 1 and has the mineral resources to support further increases of its mining and beneficiation capacity as the inevitable increase in demand is realised.

16.2 Geological Setting

The Molo deposit is situated 160 km south-east of the city of Toliara, in the Tulear region of south-western Madagascar. The deposit occurs in a sparsely populated, dry savannah grassland region, which has easy access via a network of seasonal secondary roads radiating outward from the village of Fotadrevo. Fotadrevo in turn has an all-weather airstrip and access to a road system that leads to the regional capital, (and port city) of Toliara and the Port of Ehoala at Fort Dauphin via the RN10, or RN13. (Refer Figure 46) below.

Figure 46: Molo Graphite Mine Location

16.3 Logic and Description of why Open Pit Mining is the Preferred Mining Method

The surficial, lateral expanse and the massive nature of the Molo deposit make it suitable for open pit mining methods. It is a typical pipe shaped and steeply dipping ore body, with an extended mineral outcrop along the strike (north-south direction) of the deposit.

Refer to Figure 47 below.

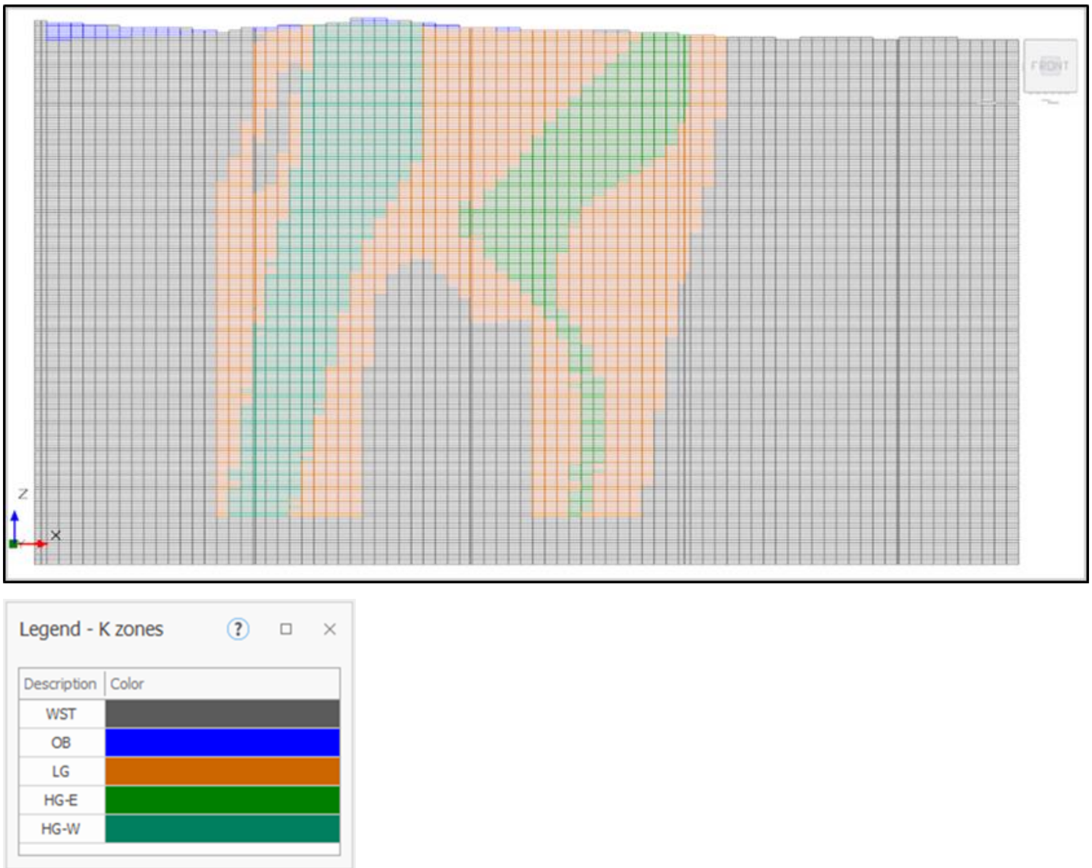


Figure 47: Typical Section E-W through the Block Model showing the Different Mineralised Zones (Source: Deswik, 2022)

16.4 Pit Geotechnical Design Criteria

16.4.1 Summary of Geotechnical Work Completed

Open House Management Solutions (OHMS) was tasked to perform the geotechnical investigation and stability assessment of pit slopes for the Molo Graphite Project.

In order to perform mine design studies, a reasonable understanding of the geotechnical environment is required and for this purpose, the following work was conducted:

- Rock mass classification from core, recovered from existing expiration boreholes and purpose located orientated diamond drilled boreholes.
- Quantification of discontinuity orientations from orientated core.
- Laboratory testing of core samples to quantify mechanical properties of the relevant rock types.

The recorded data and properties were processed and analysed to yield the following information:

- Rock Mass Rating (RMR), Mining Rock Mass Rating (MRMR) values for the relevant rock types.
- Stereo nets with defined joint sets.
- Intact and in-situ mechanical properties of the relevant rocks.
- Mohr CouLoMb and Hoek Brown failure envelopes for the relevant rock types.
- Adjusted joint shear strength parameters.

Using the geotechnical database and basic slope geometries the following designs were conducted:

- A wire frame surface was constructed to depict the contact between saprolitic and fresh rock.
- Basic assumptions of the pit depth were used to assess appropriate slope angles from design charts.
- Basic design angles and slope geometries were used to perform kinematic assessment to quantify the effect of discontinuity sets on slope stability.
- Slope geometry specifications were proposed for Whittle optimisation.
- Geometries from Whittle pit shells were used to perform numerical modelling to quantify final pit slope stability.
- Factor of Safety and Probability of Failure was quantified from slope geometries and alterations were suggested where deemed necessary.

A detailed geotechnical study was undertaken, consisting of core logging from geotechnical boreholes.

MOLO-1 to MOLO-6 can be seen in Figure 48 below.

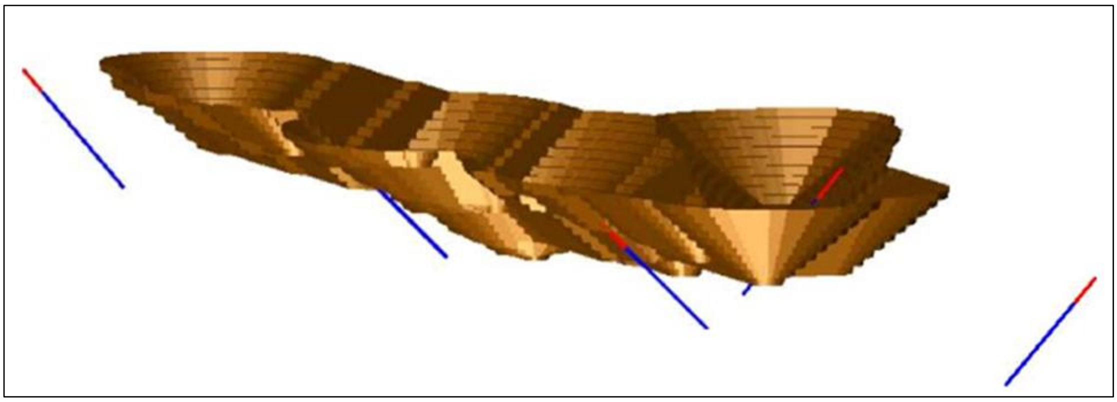


Figure 48: Location of geotechnical boreholes

Samples were collected from these boreholes for rock strength testing.

The data collected through the investigation was used for the design of slope angles and to identify potential modes and scale of instability.

This report describes the data that was recorded, details the findings of the investigation that was conducted and provides recommendations regarding the pit slope geometry.

16.4.2 Residual Soil and Rock Strength Properties

16.4.2.1 Residual Soil

The Molo mining area is covered with a thin layer of topsoil and residual soil. Residual soil can be described as completely decomposed rock and is generally mechanically weak. As the soil layer will be strapped and not exposed in the slopes, it was not considered necessary to obtain soil tests to estimate material properties. It will, therefore, not be considered in the slope stability assessment.

16.4.2.2 Saprolitic and Fresh Rock

To quantify the likely behaviour of the saprolitic and fresh rock mass, when exposed by mining operations, representative samples had Io be collected and tests were conducted at an appropriate and approved geotechnical laboratory. Samples were selected such that a representative understanding may be gained of the mechanical properties of the rock.

A series of uniaxial and triaxial strength tests were conducted. The purpose of these tests was to quantify the intact strength of the relevant rock. All tests were conducted strictly according to the prescribed ISRM procedures.

The uniaxial compressive strength tells (UCS), of core samples collected from fresh rock, we performed with strain gauges. Strain gauges were used to quantify the elastic constants, Young's modulus and Poisson's ratio.

This testing methodology allows for the full pre-failure stress strain curve to be recorded. This also allows reliable quality control of results.

The UCS values obtained from the laboratory tests were evaluated using the Modulus ratio method proposed by Dr D.U. Deere (Consultant Engineering Geology and Applied Rock Mechanics). It requires that the ratio be less than 500 to accept the test as representative.

Using this method, the average projected UCS values for samples can be seen in Table 38 below.

Table 38: Average Projected UCS Values for Samples

Rock Type	UCS
High Grade	14.9 MPa
Low Grade	15.8 MPa
Waste	32.0 MPa

16.4.3 Saprolitic and Fresh Rock Contact Surfaces

In order to understand the influence of weathering on the rock mass, the transition from saprolitic rock to fresh rock was analysed. The locations of the transition, in a selection of representative geotechnical boreholes were quantified.

Typically, geological logs of the core recovered from exploration drilling are used to establish the locations of the various transitions. As no information regarding the weathering was available on the geology logs, it was assumed that weathering would correspond to the profile constructed from the geotechnical boreholes.

The locations obtained from the borehole data were used to create point clouds. Regular grids of data points were obtained from the point clouds, using the Kriging methodology. Three-dimensional digital terrain models were constructed from the regular data grids. Six geotechnical boreholes were used in this analysis. An image of the saprolite to fresh surface, constructed from these boreholes, can be seen in Figure 49 below.

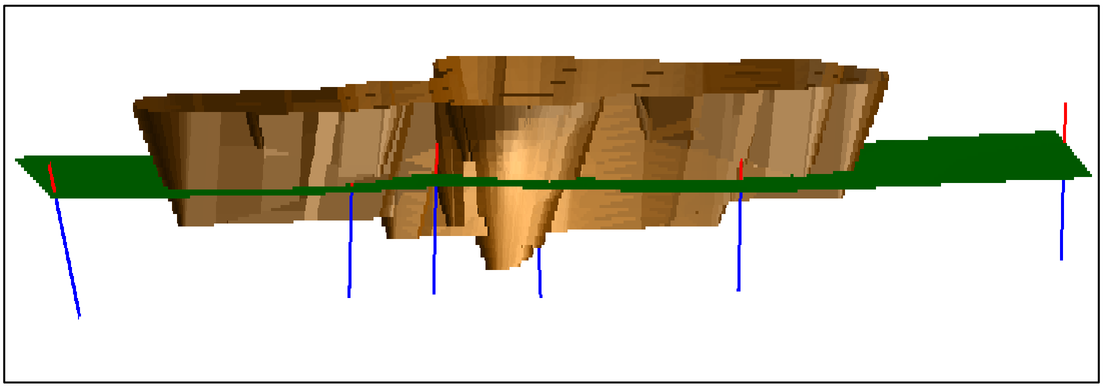


Figure 49: Location of geotechnical boreholes Slope Angle Design

16.4.3.1 *Empirical Design Chart*

For indicative purposes the Haines and Terbrugge empirical design chart was used to assess the probable safe slope angles. This analysis should, therefore, not be seen as definitive. As the chart is more relevant to fresh rock slopes, it will not be used to estimate safe slope angles for the saprolitic rock.

The minus one standard deviation values, as determined by the adjusted MRMR method, were used for the estimation (Table 39). An average of 41 was used.

Table 39: Minus One Standard Deviation Values, as determined by Adjusted MRMR Method

Ref	RMR	Standard Deviation	Weathering	Orientation	Blasting	Mean MRMR	Mean -1 Standard Deviation
MOLO-1	68	8	0.9	0.8	1	49	43
MOLO-2	72	5	0.9	0.8	1	52	52
MOLO-3	52	6	0.9	0.8	1	37	33
MOLO-4	61	8	0.9	0.8	1	44	38
MOLO-5	54	2	0.9	0.8	1	39	37
MOLO-6	63	4	0.9	0.8	1	45	43
Average MRMR							**41**

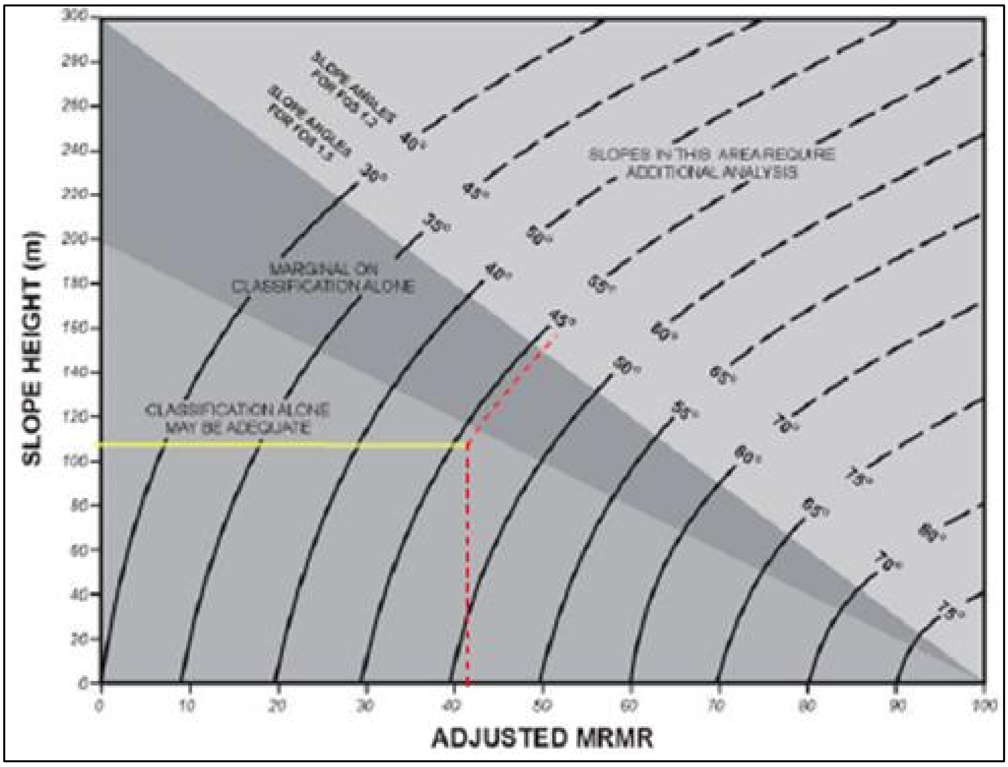


Figure 50: Haines and Tebrugge design chart for overall slope angles

The Haines and Terbrugge design chart in Figure 50, suggests that an overall slope angle of approximately 46° at a depth of 112 m, which will have a Factor of Safety (FoS) of 1.5.

Overall slope angles of 46° in fresh rock are assumed to be stable and further analysis and modelling will be conducted considering these estimates. As these angles were determined using the MRMR minus one standard deviation, it may be reasonably conservative.

16.4.4 Conclusion

Geological structures in various rock types were quantified by means of measurements from orientated core. Two joint sets, with their mean dip and orientations were identified.

The rock mass quality has been quantified to a reasonable extent through geotechnical logging of core. The rock mass quality was quantified using the RMR methodology proposed by Bieniawski. RMR values for saprolitic and mesh rock were determined in Table 40 below.

Table 40: Residual Soil and Rock Strength Properties

Ref	RMR	Standard Deviation	Weathering	Orientation	Blasting	Mean MRMR	Mean -1 Standard Deviation
Saprolite	37	10	09	OB	1	27	19
Fresh	62	13		OB	1	45	35

Adjustments to Rock Mass Rating (RMR) were performed to estimate Mining Rock Mass Rating (MRMR) values for design purposes.

A representative sample of all the rock types that are likely to be encountered was collected and tested in an accredited geomechanics laboratory. The tests were appropriate for the quantification of the material to allow for reasonable design of pit slope angles.

Laboratory tests results were downgraded using RMR values to quantify the in-situ mechanical properties of the rock. The mechanical properties were defined in terms of Mohr CouLoMb and Hoek-Brown failure criteria.

Slope stability was assessed using Phase2D Finite Element Model. Factor of Safety and Probability of Failure were determined for the given pit geometries. Models were run, and it was found that the FoS is satisfactory. From all these models it is clear that circular failure in the weathered rock is the most likely failure mechanism. The likelihood of failure occurring is, however, remote given the high FoS and low Probability of Failure. The modelling does suggest that there is considerable scope to steepen the ultimate slope angles. The low overall slope angles of the pit geometry resulted in very 37 high FoS and low Probability of failure, facilitating the optimization of the pit shell.

A transitional surface between saprolite and fresh rock was constructed from borehole information. DXF files of the transitional surface are provided and can be incorporated into mine plans.

Slope angles determined from the Haines and Terbrugge design chart suggest overall slope angles of 46° with a Factor of Safety of 1.5 in fresh rock.

Various kinematic failure modes were investigated, and it was found that the rock mass reflects a low probability of failure. The potential for toppling type failure may, however, occur on the east dipping slope, and should be considered during mining.

Catch berms with 8m widths were determined for slopes at 46° in weathered rock and 6m wide for fresh rock.

Rock fall hazard analysis was performed, and it was found that the hazard is reasonably low, and the width of catch berms will be sufficient to contain most potential falls.

No seismic activity is anticipated during the mining process.

The quantification of critical input parameters and level of detail considered in the design is sufficient for bankable feasibility purposes. It remains the responsibility of the operator to continuously verify critical rock mass parameters during the mining operations. Continuous verification of the design is strongly recommended during the operational phase.

16.5 Pit Optimization Methodology

16.5.1 Recognition of Revenue

The Carbon Concentrate produced by the process plant is seen as accrued revenue and is recognized as an asset on the balance sheet, because it represents revenue that has been earned, but not yet received. Since the company has provided goods, or services associated with the revenue, its obligation is met, which means it can count the revenue as an asset, rather than a liability.

Block Ore Tonnes x Block Ore Grade x Plant Efficiency = Carbon Concentrate tons.

Carbon Concentrate Tonnes x Basket sales price per ton of concentrate = Revenue recognized @ Mine Gate.

16.5.2 Basis of Cut-off Derivation

Utilizing the Deswik Model, the ore tonnes for Ore C>= 3% to <4.5% were determined to be 7.29 Mt, 38% of which is classified as Large Flake (Low Grade) and 62% classified as Fine Flake (High Grade) as per the geological flags in the "finmod" geological database.

If the Fine Flake product is combined with the Large Flake product, which has a market price of US$1,400 - US$2,300 ref: (North American FS Level Graphite Projects), the profit margin is positive, as indicated in Table 41 below.

Table 41: Profit Margin

Description	Unit	Unit Cost	Large Flake @ 38% of ROM	Fine Flake @ 62% of ROM	Combined LF+FF
Plant Recovery	%	88.30%	88.30%	88.30%	88.30%
ROM ore (3%> = to <4.5%)	t		2,771,928	4,522,619	7,294,546
Product	t		98,751	161,120	259,871
Waste & Ore Mining Cost	US$	6.78	18,793,669	30,663,355	49,457,025
Rehandle of Ore at Plant	US$	0.56	1,552,279	2,532,667	4,084,946
Processing Cost per Ore Tonne	US$	19.11	52,971,537	86,427,245	139,398,783
G&A Cost	US$	12.64	35,037,166	57,165,902	92,203,067
G&A Cost / Ore Tonne	US$				
Product Selling Cost	US$	190.55	18,816,983.33	30,701,393.85	49,518,377.18
Product Basket Price (Revenue) Ton	US$/t		1,750.00	1,230.50	1,427.91
Product Revenue	US$		172,814,069	198,258,017	371,072,086
SR t waste / t Ore	t/t	0.25	0.25	0.25	0.25
Mining Cost	US$		127,171,635	207,490,563	334,662,198
Profit/Loss/t Ore Mined and Processed			**45,642,434**	**-9,232,546**	**36,409,888**

The Graph in Figure 51 indicates the price that has to be obtained for the Large Flake product, in order for the combined Profit / Costs to break even, for the 7.29 Mt.

As it was considered that the market price of US$1,381.30 will be easily achieved for the Large Flake Graphite, an Ore Grade C of 3% cut-off was selected for the Pit Optimisation.

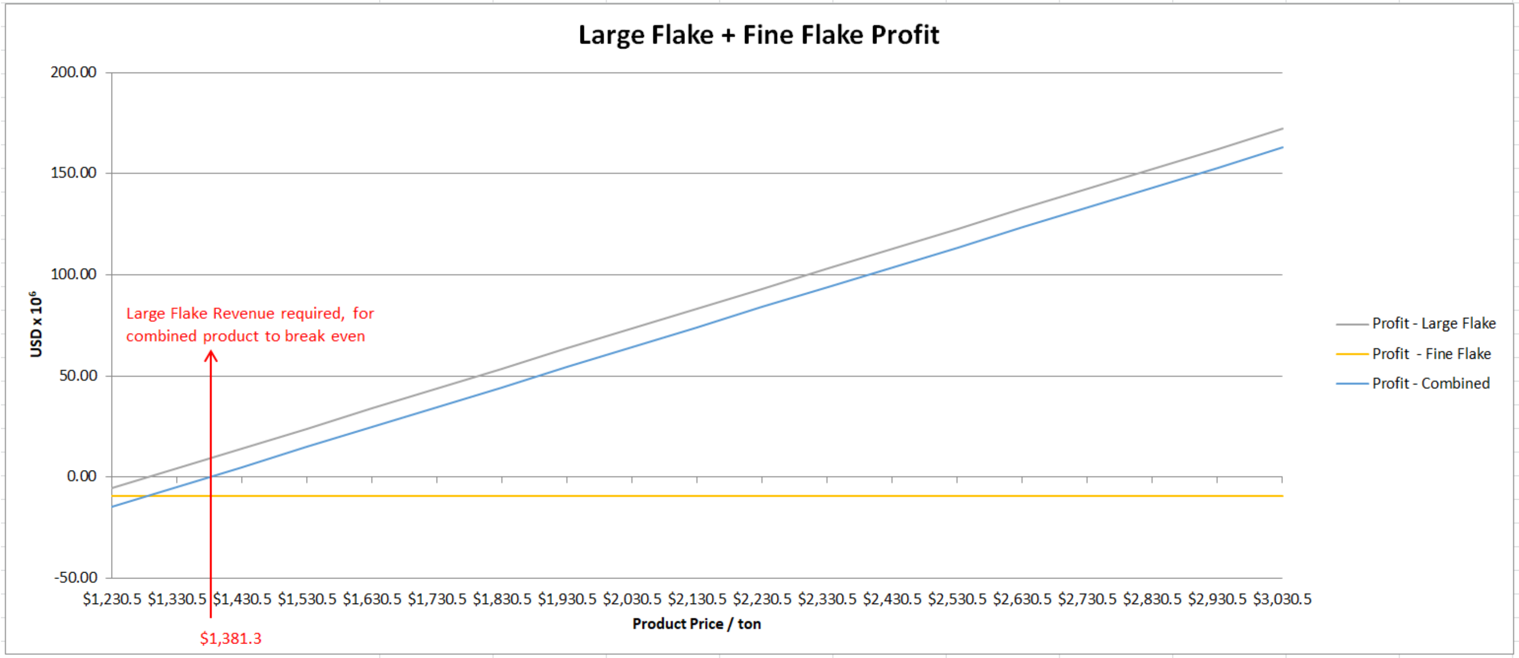


Figure 51: Large Flake and Fine Flake Profit

16.5.3 Detailed Description of Application of Psuedoflow Algorith

Pseudoflow is a network flow algorithm that determines pit shells at varying revenue factors for a deposit, using specific input parameters including slope dependencies, costs, and revenues. Firstly, you need to prepare your block model for optimization and then configure and run Pseudoflow passes to determine the maximum economic mining shell for the deposit, which utilizes the Pseudoflow algorithm. The Pseudoflow algorithm is a computationally more efficient algorithm developed some 35 years after the original Lerchs-Grossman algorithm (1965). Pseudo – Input Parameters shown in Table 42 below.

Table 42: Pseudoflow – Input Parameters

Category	Item	Value	Unit	Data Source
Geology and Survey Inputs	Geological Model initial			finmod_v4.dm
	Geological Model regularized			finmod_v4_reg10X10X2_V02_Detail_proven.gmdlb
Financial Parameters	Graphite cut-off grade	3.0	%	Meeting with Next Source Materials – 02/08/2022
	Basket sales price of product (Revenue)	1230.50	USD/prod t	Meeting Next Source Materials – 02/08/2022; Molo Phase2 PEA_27April2022 – Table45
	Waste and Ore mining (Cost)	6.78	USD/romt	Molo Phase2 PEA_27April2022 – Table34
	Rehandle of Ore at plant (Cost)	0.56	USD/romt	Molo Phase2 PEA_27April2022 – Table34
	Processing (Cost)	19.11	USD/romt	Molo Phase2 PEA_27April2022 – Table34
	General and Administration (Cost)	12.64	USD/romt	Molo Phase2 PEA_27April2022 – Table34
	Product selling (Cost)	190.55	USD/prod t	Molo Phase2 PEA_27April2022 – Table34
Geotechnical Design	Overall batter angle	46	Degrees	Molo Geotechnical Report; Molo Phase2 PEA_27April2022
	Bench height	8	meters	Molo Phase2 PEA_27April2022 – Table34
Mining Inputs	Ore mining recovery	95	%	Molo Phase2 PEA_27April2022 – Table34
Plant Inputs	Plant Capacity Phase1 from Nov2022	17,000	Prod tpa	Molo Phase2 PEA_27April2022
	Plant Capacity Phase2 from Apr2024	150,000	Prod tpa	Molo Phase2 PEA_27April2022
	Plant recovery	88.3	%	Molo Phase2 PEA_27April2022 – Table34

16.6 Pit Design

16.6.1 Basis of Design including Widths, Geotechnical Considerations & Gradients

According to the study report done by Open House Management Solutions (OHMS) the recommended catchment berm widths are as follows:

- Weathered Rock: 8m width.
- Fresh Rock: 6m width.

The average depth of the Weathered rock according to the study is at about 40m and then the Fresh rock follows further down.

As there is no rock classification in the Geological Model, the Deswik Model was based on the 40m average.

Refer to Figure 52 below.

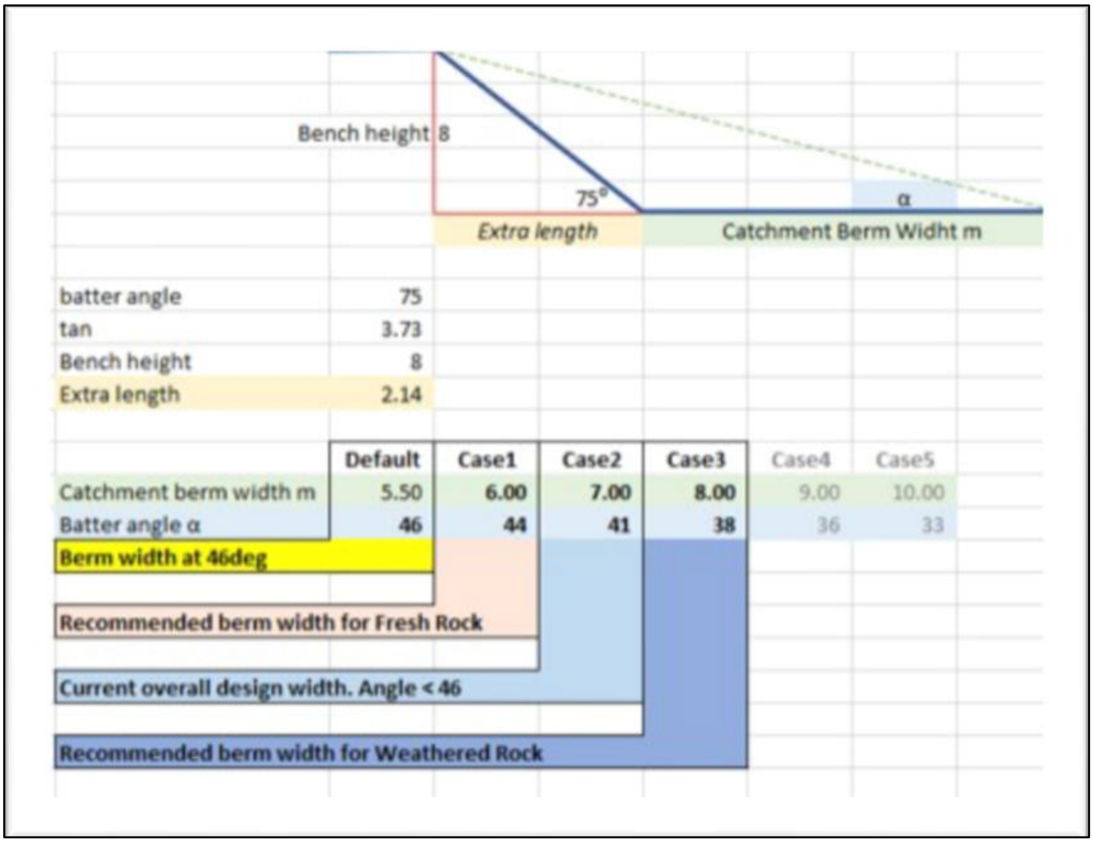


Figure 52: Pit Shell Design Criteria

16.6.2 *Open Pit Final Design*

Using the minimum mining width of 30m suited for the selected mining equipment and the recommended 46° slope angle, the final pit design at Molo Graphite was drafted including the following key design parameters.

A single ramp was designed in the centre zone of the pit and is designed at a width of 15m. The single ramp is sufficient access to the pit due to the low mining volumes and total mineable volume within the designed pit shell. The ramp width is designed to allow dual way traffic and satisfies the industry norm of the ramp width at least 3.5 times the width of the largest vehicle. The (38 tonne) tipper trucks at a width of 2.5m are more than suited for these ramps, which are designed at a 10% gradient and ramp switchbacks, or turns are designed in a 180° turn with a radius of 15m.

The Bench design includes a bench height of 8m, a berm width of 8m in weathered material, or 6m in fresh material and a bench slope or batter angle of 75°, as recommended from the detailed geotechnical study.

Refer to Table 43, Figure 53 and Figure 54 below.

Table 43: Pseudoflow Shell vs Whittle Shell – Ore / Waste Comparison

Reference	Total Tonnes	Waste Tonnes	Ore Tonnes	Grade	Economic Grade Cu-off	Graphite Tonnes
PH2 PEA Table 37 Pit Optimisation Results Opt 2. (SMS2022)	109,891,920	45,886,570	64,005,350	6.6%	4.50%	3,575,170
Deswik CAD Final Design	73,816,140	17,197,640	56,618,500	6.28%	3.00%	3,131,120

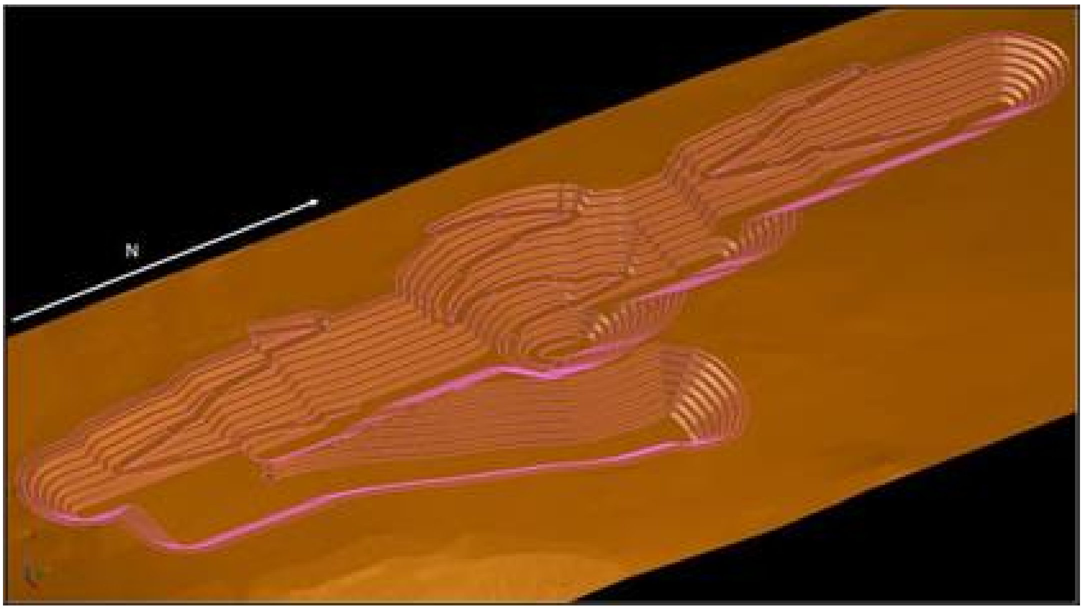


Figure 53: Whittle – Final Pit Shell

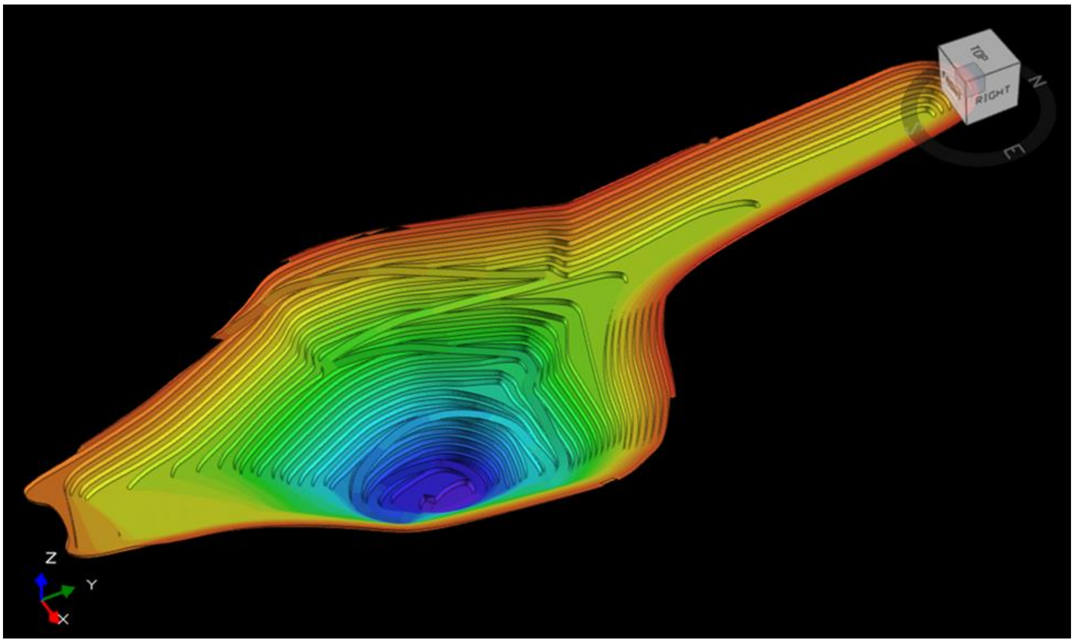


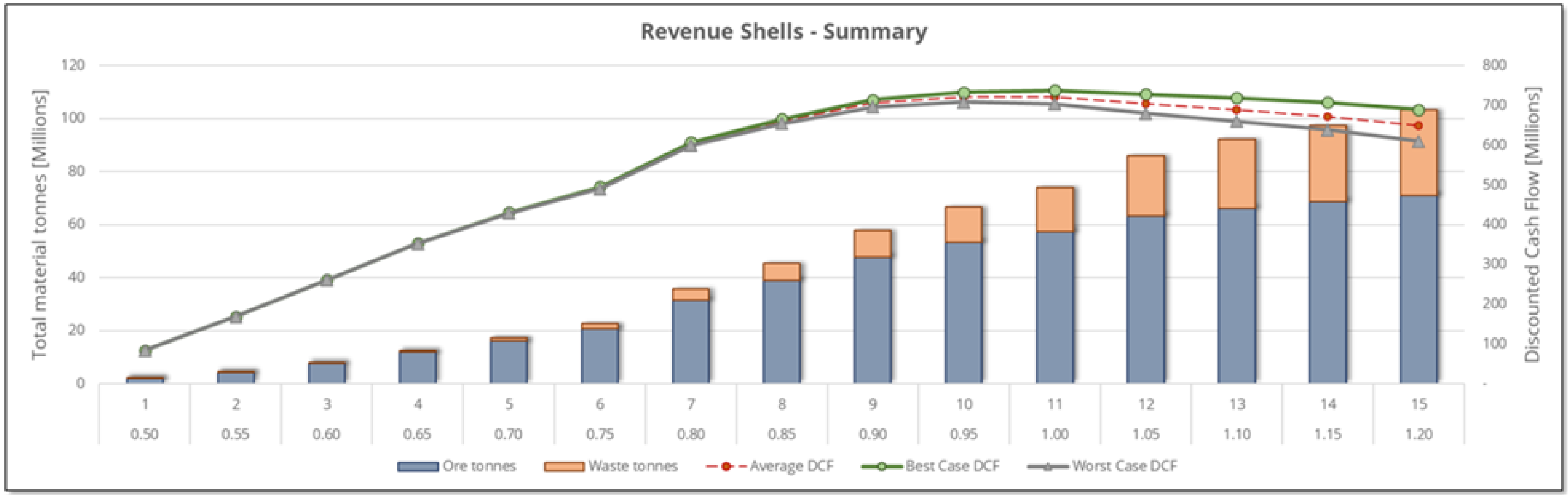
Figure 54: Pseudoflow – Final Pit Shell

16.6.3 Pit Optimisation Results

The scenario is constrained and as a result all ore materials, including ore from the Measured, Indicated and Inferred Mineral Resource categories, are sent to the treatment plant, however, a cut-off grade of 3.0% carbon has been applied.

All material below this cut-off grade was treated as waste. Graph 34 presents the derived pit optimisation results. Pit Shell No:10 was selected and considered to represent a practical compromise for the final pit shell selection.

Revenue Factor	**0.50**	**0.55**	**0.60**	**0.65**	**0.70**	**0.75**	**0.80**	**0.85**	**0.90**	**0.95**	**1.00**	**1.05**	**1.10**	**1.15**	**1.20**
Shell Number	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**
Ore Tonnes	1,922,327	4,448,356	7,790,022	11,986,960	16,263,561	20,672,549	31,594,611	39,080,803	47,840,357	53,449,299	57,384,290	63,400,314	66,314,998	68,621,424	71,069,710
Waste Tonnes	49,897	112,156	243,877	583,708	1,167,295	2,009,078	4,136,763	6,348,174	10,214,120	13,394,372	16,755,451	22,705,722	26,053,953	28,935,013	32,291,277
Best Case DCF	84,010,848	169,392,626	261,197,109	354,148,099	431,694,414	495,296,885	607,296,013	666,303,707	714,657,619	733,259,669	737,607,449	728,662,058	719,170,423	707,193,441	689,498,894
Worst Case DCF	84,010,848	169,287,441	260,816,774	353,130,136	429,504,658	491,407,451	599,572,899	654,882,227	696,748,787	708,217,009	704,266,739	680,187,006	660,500,663	638,990,134	610,383,719
Ave DCF Change		85.33M	91.67M	92.63M	76.96M	62.75M	110.0M	57.16M	45.11M	15.04M	0.2M	-16.51M	-14.59M	-16.74M	-23.15M



Graph 34: Conceptual LoM Production Schedule based on Pit Shell 10 (Source: Deswik, 2022)

16.6.4 Assumptions

The massive nature of the Molo deposit, the absence of major geotechnical anomalies and the flexibility benefits of using small equipment indicates that the mining recovery can comfortably be set at 95%. By applying a 3% waste dilution factor, a more realistic and reliable estimate of the recoverable graphite can be obtained, minimizing the risk of overestimating ore grades and optimizing operational planning for sustainable and efficient mining practices.

Refer to Table 44 below.

Table 44: Assumptions Table

Item	Value	Description
Dilution	3%	Inclusion of non-economic material in the extracted ore.
Mining Recovery	95%	Mining and Geological losses.
Plant Recovery	88.3%	Plant efficiency.

16.7 Mining Equipment

16.7.1 Basis of Type Selection

At the start of Phase 1 of the “Molo Graphite Mine” project, the decision was made to purchase the following mining operational machines:

- Three of Sany SYL956H Front End Loaders with 3m³ Bucket.
- Sany SY500H Excavators with 2.2 – 3.1m³ Bucket.
- Shantui SD22W Dozer.
- Sany SSR100AC Single Drum Roller.
- TAIYE X45DTH Hydraulic Drill Rig (115 mm Diameter).
- Three of Sinotruk Howo 7.18m³ Tipper truck.
- Sinotruk Howo 64290 Water Tanker, 20 kL.

For economic and logistical reasons, it was decided to purchase Chinese manufactured equipment as pricing is far more competitive than US/European, or Japanese equivalents. Purchasing all equipment from a single country had the added advantage that logistics for importing equipment to Madagascar was made a lot simpler and less costly.

Due to access road bridge weight limitations of 25 t route from Fort Dauphin to Fotadrevo, it was decided to purchase two Sany SY245H Excavators (Operating Weight = 25,500 kg) instead of the Sany SY500H Excavator (Operating Weight = 49,500 kg), with a much smaller bucket capacity of 1.3m³ each.

The Sinotruk 18m³ tipper trucks were not available, thus had to be replaced by the Sinotruk 26m³ tipper trucks, creating a mismatch in loader / haul truck configuration. The general "rule of thumb" within the mining industry, is that the loader should do 4 to 6 passes to load a truck if the loading cycle is to be productive and cost effective.

For "Molo Graphite Mine" Expansion Project Feasibility Study, it was decided to include 4m³ buckets for the Sany SYL956H Front End Loaders and include two Sany SY750H Excavators with 4.5m³ buckets, aligning the mining operation with the 4 to 6 pass "rule of thumb". In addition, another 11 Sinotruk 26m³ tipper trucks are added, to achieve the required production targets.

The "make and type" of mining equipment is kept the same as in Phase 1 of the mining operation, as the stores need to keep spares for all equipment, which is a lot more diverse and costly if a wider variety of equipment is used.

Another factor that affects "Molo Graphite Mine" spares holding is the mine location, which complicates the logistics of transporting spares to and from the mine.

The Sany SY750H Excavator, has an Operating Weight of 76,200 kg, which will be problematic for bridge crossings when transporting the equipment from port to mine site. The excavators will have to be disassembles at port for transport purposes and reassembled on the mine premises.

16.8 Equipment Matching Philosophy, Haul Profiles, Cycle Times and Calculations

16.8.1 Mining Sequence

Refer to Figure 55 below.

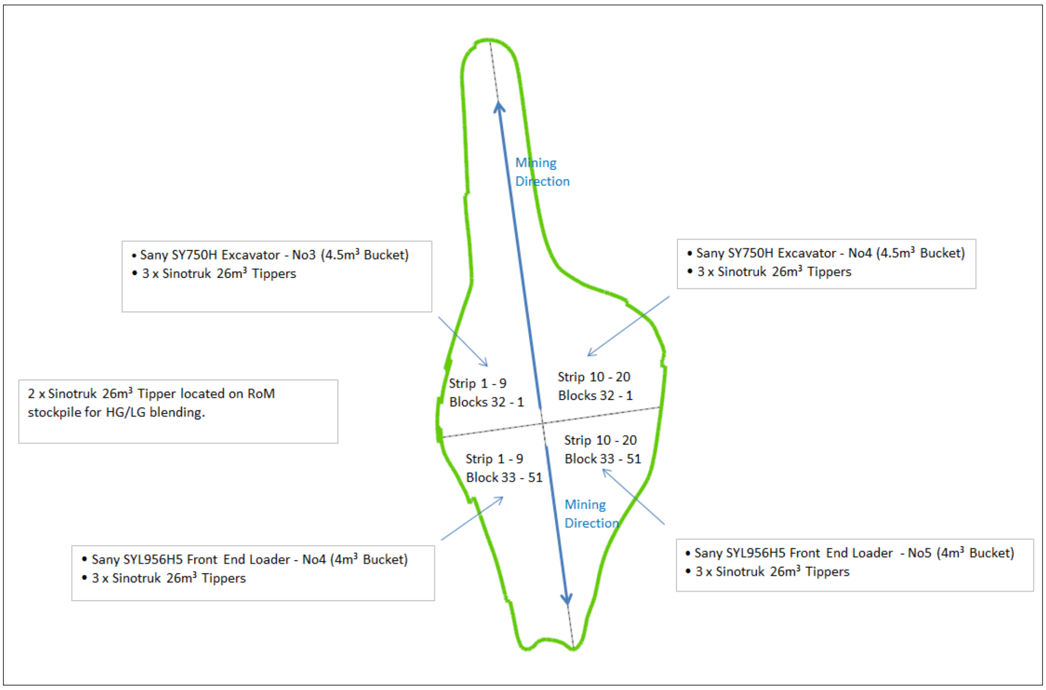


Figure 55: Snapshot A

16.8.2 Loading and Hauling Machine Cycle Times

Ore properties	UoM	Value
Ore		Graphite
Density in-situ waste	t/m³	2.62
Density in-situ ore	t/m³	2.36
Broken bulk	t/m³	1.56
Fill factor	%	90%

Description	Bucket capacity m³	Bucket capacity ton	Passes/RDT 18m³ / 25 t	26m³ / 37 t
Sany FEL - SYL956H - 3m³	3	4.2	6	9
Sany FEL - SYL956H - 4m³	4	5.6	5	7
Sany Excavator - SY245H	1.3	1.8	14	20
Sany Excavator - SY750H	4.5	6.3	4	6

Loading time - 26m³ RDT				Reference
Sany SYL956H 3m³ x	9 passes @	50 s/pass =	7.5 minutes	Lebedev - Dynamic Simulation
Sany SYL956H 4m³ x	7 passes @	50 s/pass =	5.4 minutes	Caterpillar perfomance Handbook 48
Sany Exc SY245H x	20 passes @	30 s/pass =	10 minutes	Methvin - Excavator Production Estimation Table
Sany Exc SY750H x	6 passes @	40 s/pass =	4 minutes	Methvin - Excavator Production Estimation Table

Table 45: FEL & Excavator Productivity

Description	Avail	Util	Hrs/Day	Bucket Capacity (m^3)	Bucket Capacity (t)	Cycle/Hr	Ton/Hr	Tons/Day
Sany FEL-SYL956H 3m³	85%	39%	8.0	3.0	4.2	72	303	2,413
Sany FEL-SYL956H 4m³	85%	58%	11.8	4.0	5.6	72	404	4,784
Sany Excavator -SY245H	85%	58%	11.8	1.3	1.8	120	219	2,591
Sany Excavator -SY2750H	85%	58%	11.8	4.5	6.3	90	569	6,728

16.8.3 Rear Dump Truck Tipping Time

Tipper 26m³ – Tipping Time = 0.7 minutes.

16.8.3.1 Truck Speeds

Table 46: Reference – Lebedev Dynamic Simulation

Speed Empty	UoM	Value
Flat on surface	km/h	30
Down ramp	km/h	27
In Pit	km/h	18
Speed Laden		
In Pit	km/h	15
Up the ramp	km/h	12
Flat on surface	km/h	30

16.8.4 Rear Dump Truck Productivity

For the purpose of calculating RDT productivities to be included in the Deswik Schedule, 4 time frames evenly spread across the LoM schedule where selected, and productivities calculated over the measured haul distances.

The first period June 2026 to July 2027 is pictorially represented by snapshot B.

Refer to Figure 56 below.

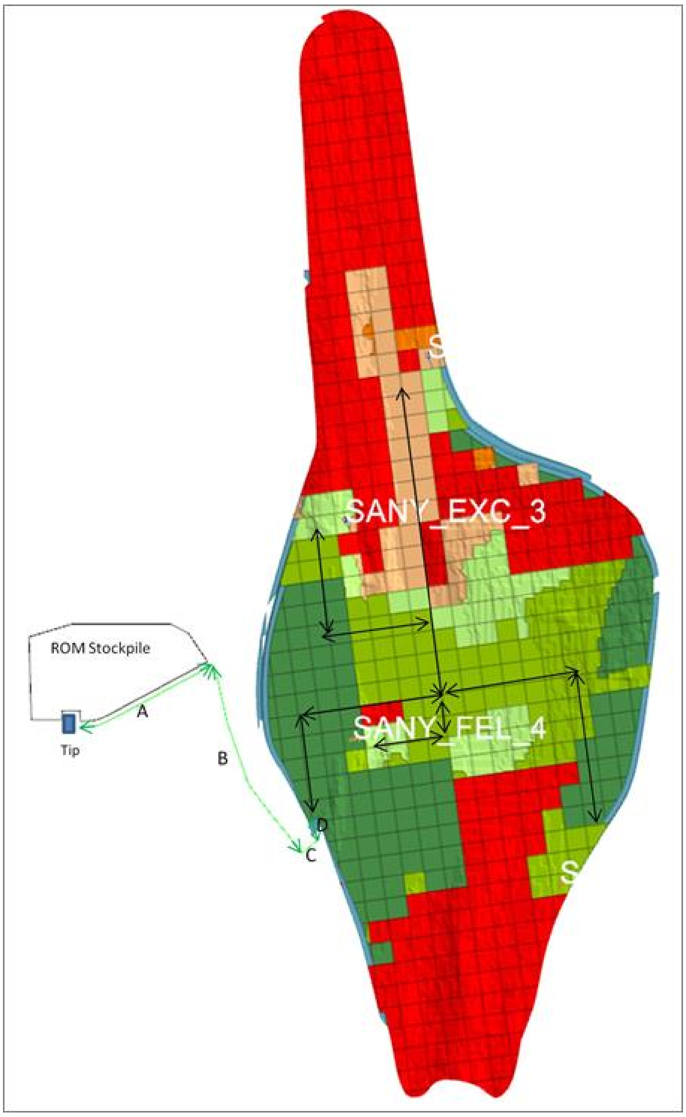


Figure 56: Snapshot B

Haul Distance – Start – Jul 27	
Truck Haul	**Distance (m)**
Flat in pit	726
Up/down ramp	40
Flat on bank	542
Up/down ramp S/P, Tip	12
Flat on pad, S/P and Tip	340

Table 47: Rear Dump Truck Haul Distance & Productivity for Snapshot B

RDT PRODUCTIVITY																
Description	**% Avail**	**% Util**	**Hours/ Day**	**Average Speed Empty (km/h)**	**Average Speed Full (km/h)**	**Haul Distance Pit to Tip (km)**	**Cycle Time Empty (min)**	**Cycle Time Full (min)**	**Loading Time (min)**	**Tipping Time SP (min)**	**Total Cycle Time SP (min)**	**Load Body Capacity (m^3)**	**Load Body Capacity (t)**	**Tonne per Hour (tph)**	**Tonnes per Day (tpd)**	
RDT $26m^3$	85%	58%	11.8	24.7	22.9	1.66	4.0	4.4	5.4	0.7	14.5	26	37	152	1,784	
Loading Unit = SANY FEL 956H $4m^3$ Bucket																
RDT $26m^3$	85%	58%	11.8	24.7	22.9	1.66	4.0	4.4	10.0	0.7	19.1	26	37	115	730	
Loading Unit = SANY excluding SY245H $1.3m^3$ Bucket																
RDT $26m^3$	85%	58%	11.8	24.7	22.9	1.66	4.0	4.44	4	0.7	13.1	26	37	167	1,084	
Loading Unit = SANY excluding SY750H $4.3m^3$ Bucket																

The second period August 2027 to December 2030 is pictorially represented by snapshot C. Refer to Figure 57 below.

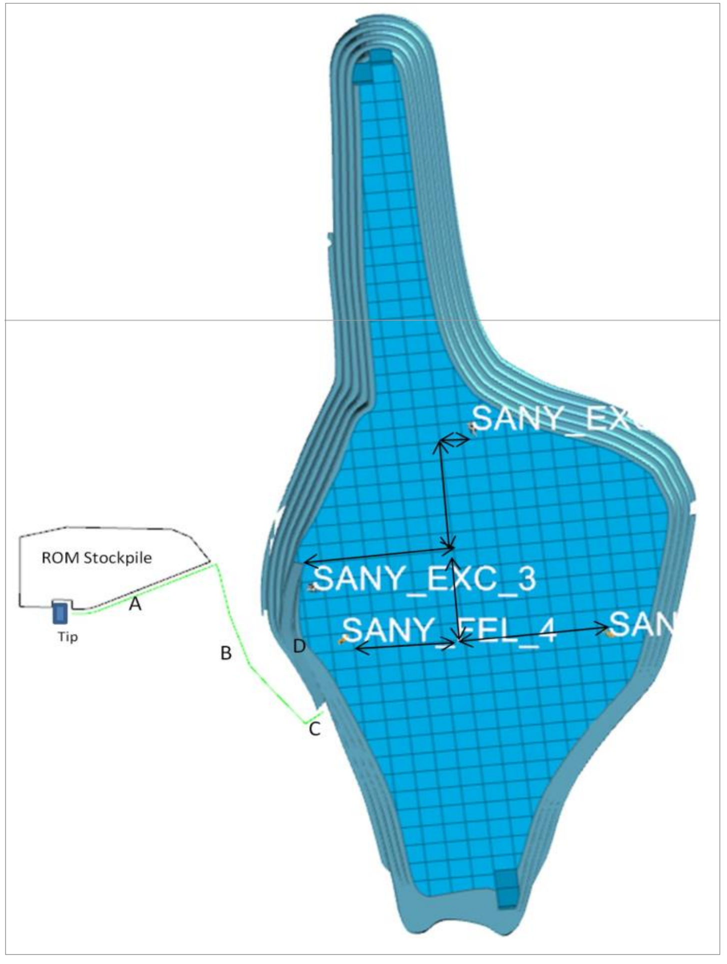


Figure 57: Snapshot C

Table 48: Rear Dump Truck Haul Distance & Productivity for Snapshot C

Haul Distance – August 27 – December 30	
Truck Haul	**Distance (m)**
Flat in pit	579
Up/down ramp	262
Flat on bank	542
Up/down ramp S/P, Tip	12
Flat on pad, S/P and Tip	340

Table 49: Rear Dump Truck Haul Distance & Productivity for Snapshot C

RDT PRODUCTIVITY																
Description	**% Avail**	**% Util**	**Hours/ Day**	**Average Speed Empty (km/h)**	**Average Speed Full (km/h)**	**Haul Distance Pit to Tip (km)**	**Cycle Time Empty (min)**	**Cycle Time Full (min)**	**Loading Time (min)**	**Tipping Time SP (min)**	**Total Cycle Time SP (min)**	**Load Body Capacity (m^3)**	**Load Body Capacity (t)**	**Tonn es per Hour (tph)**	**Tonnes per Day (tpd)**	
RDT 26m^3	85%	58%	11.8	25.5	22.2	1.74	4.1	4.7	5.4	0.7	14.9	26	37	147	1,738	
Loading Unit = SANY FEL 956H 4m^3 Bucket																
RDT 26m^3	85%	58%	11.8	25.5	22.2	1.74	4.1	4.7	10.0	0.7	19.5	26	37	112	1,329	
Loading Unit = SANY excluding SY245H 1.3m^3 Bucket																
RDT 26m^3	85%	58%	11.8	25.5	22.2	1.74	4.1	4.7	4	0.7	13.4	26	37	162	1,920	
Loading Unit = SANY excluding SY750H 4.3m^3 Bucket																

The third period January 2031 to December 2036 is pictorially represented by snapshot D. (Refer to Figure 58) below.

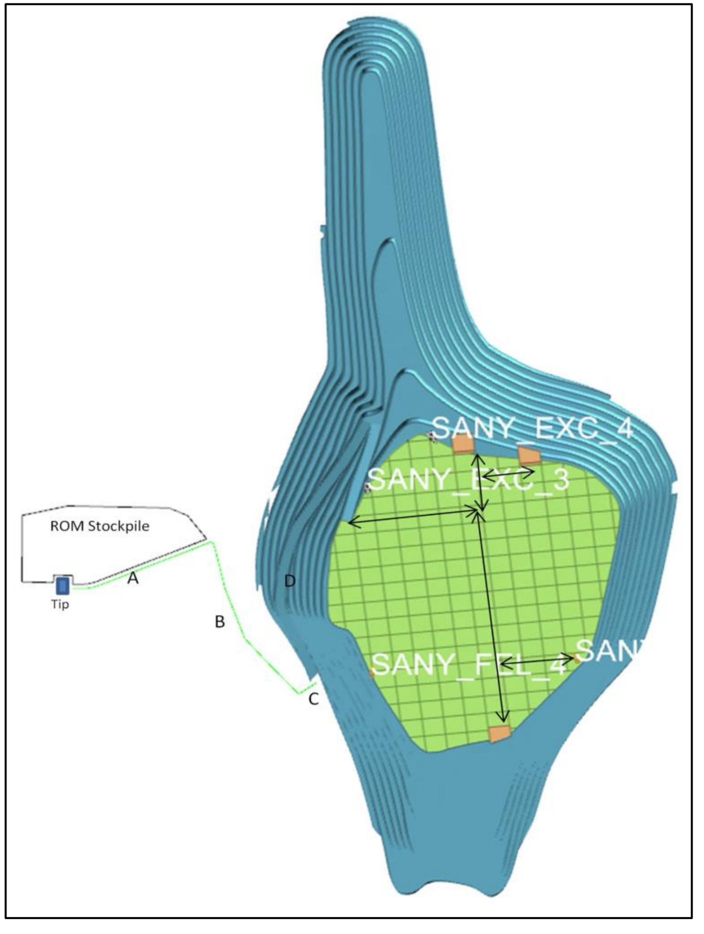


Figure 58: Snapshot D

Table 50: Rear Dump Truck Haul Distance & Productivity for Snapshot D

Haul Distance – Jan-31 to Dec-36	
Truck Haul	**Distance (m)**
Lat in pit	432
Up/down ramp	706
Flat on bank	542
Up/down ramp S/P, Tip	12
Flat on pad, S/P and Tip	340

Table 51: Rear Dump Truck Haul Distance & Productivity for Snapshot D

RDT PRODUCTIVITY																
Description	**% Avail**	**% Util**	**Hours/ Day**	**Average Speed Empty (km/h)**	**Average Speed Full (km/h)**	**Haul Distance Pit to Tip (km)**	**Cycle Time Empty (min)**	**Cycle Time Full (min)**	**Loading Time (min)**	**Tipping Time SP (min)**	**Total Cycle Time SP (min)**	**Load Body Capacity (m^3)**	**Load Body Capacity (t)**	**Tonn es per Hour (tph)**	**Tonnes per Day (tpd)**	
RDT 26m³	85%	58%	11.8	26.4	20.5	2.03	4.6	6.0	5.4	0.7	16.7	26	37	131	1,550	
Loading Unit = SANY FEL 956H 4m³ Bucket																
RDT 26m³	85%	58%	11.8	26.4	20.5	2.03	4.60	6.0	10.0	0.7	21.3	26	37	103	1,217	
Loading Unit = SANY excluding SY245H 1.3m³ Bucket																
RDT 26m³	85%	58%	11.8	26.4	20.5	2.03	4.6	6.0	4	0.7	15.3	26	37	143	1,694	
Loading Unit = SANY excluding SY750H 4.3m³ Bucket																

The fourth period January 2037 to June 2048 is pictorially represented by snapshot E. (Refer Figure 59) below.

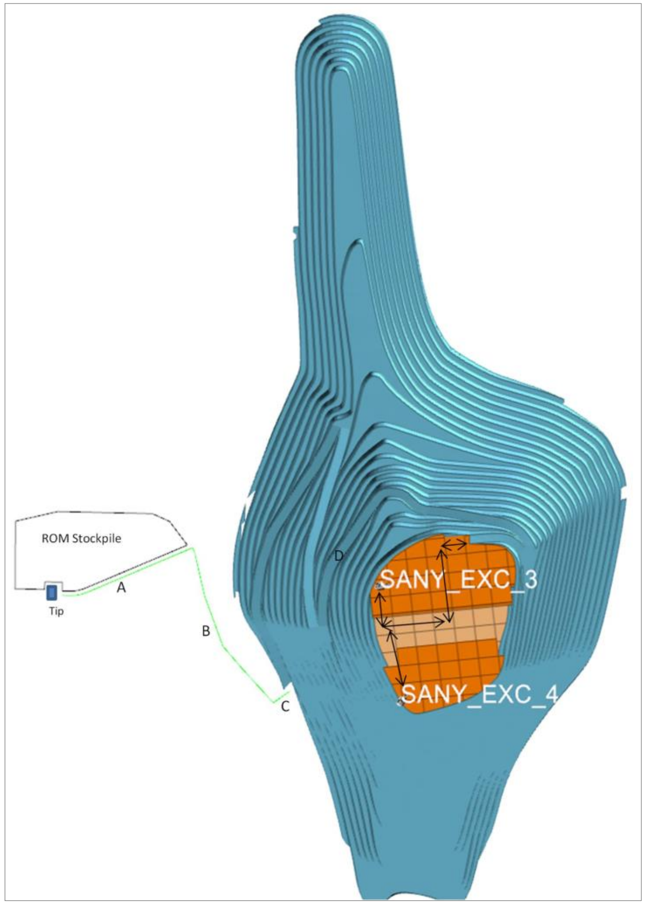


Figure 59: Snapshot E

Table 52: Rear Dump Truck Haul Distance & Productivity for Snapshot E

Haul Distance – January 37 – June 48	
Truck Haul	**Distance (m)**
Flat in pit	385
Up/down ramp	1,555
Flat on bank	542
Up/down ramp S/P, Tip	12
Flat on pad, S/P and Tip	340

RDT PRODUCTIVITY																
Description	**% Avail**	**% Util**	**Hours/ Day**	**Average Speed Empty (km/h)**	**Average Speed Full (km/h)**	**Haul Distance Pit to Tip (km)**	**Cycle Time Empty (min)**	**Cycle Time Full (min)**	**Loading Time (min)**	**Tipping Time SP (min)**	**Total Cycle Time SP (min)**	**Load Body Capacity (m^3)**	**Load Body Capacity (t)**	**Tonnes per Hour (tph)**	**Tonnes per Day (tpd)**	
RDT 26m³	85%	58%	11.8	26.7	18.0	2.83	6.4	9.4	5.4	0.7	21.9	26	37	100	1,181	
Loading Unit = SANY FEL 956H 4m³ Bucket																
RDT 26m³	85%	58%	11.8	26.	18.0	2.83	6.4	9.4	10.0	0.7	26.5	26	37	83	977	
Loading Unit = SANY excluding SY245H 1.3m³ Bucket																
RDT 26m³	85%	58%	11.8	26.7	18.0	2.83	6.4	9.4	4	0.7	20.5	26	37	107	1,263	
Loading Unit = SANY excluding SY750H 4.3m³ Bucket																

Table 53: Equipment Quantity

Number	Description	Total Units Phase 2
1	Sany SMG 200 Grader.	2
2	Front End Loader – SYL956H5.	3
3	Excavator - SY245H.	2
4	Excavator - SY750H.	2
5	Shantui Dozer - SD 22W.	3
6	Tipper 26m³ - Sinotruk Howo 7 84371.	14
7	Water Truck 20,000L - Sinotruk Howo 64290.	2
8	Single Drum Roller - Sany SSR100C-10.	2
9	Diesel Trailer (2,000 L).	2
10	Drill - TAIYE X45 DTH.	3
11	Hyundai County Bus – Diesel.	4
12	SYL956H5 FEL 4 m³ bucket.	3
13	Metso HM75 pump, Hatz 4L41C Diesel Engine.	1
14	Dewatering Pump - PAS 150 MF 250 (540m³/hr).	4
15	Pump – 150 mm, 6 m suction hose with fittings.	10
16	Pump – 150 mm, 100 m flexible hose with fittings.	50
17	Transfer Tank - electric motor, pump with 220 m vertical delivery head.	1
18	Lighting Plant – Kubato.	4

Front End Loader, Excavator and Haul Truck numbers are based on monthly production targets obtained from Deswik Schedule report.

16.8.4.1 Replacement Strategy and Assumptions

For the purpose of the "Molo Graphite Expansion Project", the equipment replacement was based on the OEM life of equipment hours, as indicated in Table 54 below.

Table 54: Original Equipment Manufacturer Life Cycle Hours

Name	OEM Life Cycle (Hours)
Sany_SY245H_Excavator	15,000
Sany_SY750H_Excavator	25,000
Sany_SYL956_Front End Loader	20,000
Sinotruk Howo 7 Tipper 26m^3	20,000
Shantui SD22W Bulldozer	20,000
TAIYE X45DTH Hydraulic Drill Rig	20,000
Sany SMG200C-8 Grader	20,000
Sinotruk Howa Water Tanker 20,000 Litres	20,000
Sany SSR100AC-8 Single Drum Roller	20,000
LDV – Diesel Trailer 2,000 L	20,000
Hyundai County Bus	20,000
LDV – Service Trailer	20,000
Supervision Vehicles	30,000
Metso HM75 Pump, Hatz 4L41C Diesel Engine	20,000
Dewatering Pump – PAS 150 MF 250 (540/m^3/Hour)	20,000
Kubato – Lighting Plant	20,000

16.9 Equipment Maintenance

16.9.1 Mechanical

A maintenance schedule for Earth Moving Equipment includes regular inspections, routine maintenance tasks, and periodic servicing to ensure the equipment operates efficiently and safely. Below is the maintenance and service schedule for the mines Earth Moving Equipment (EME).

The specific maintenance schedule and tasks vary depending on the manufacturer, model, and usage conditions of the earthmoving equipment. The equipment's manual is consulted, and the manufacturer's guidelines followed to ensure proper maintenance and safe operation.

16.9.1.1 Electrical

Ground Engagement Tools (GET) consumption assumptions.

Most GET, such as bucket teeth, cutting edges, and blades, have wear limits specified by the equipment manufacturer. These wear limits are typically measured in terms of the percentage of wear or the thickness of the material remaining. Once the GET approaches these limits, it is time for replacement.

The quality and type of GET can also impact their longevity. Higher quality GET may last longer and require less frequent replacement.

Regular maintenance, such as sharpening, or rebuilding GET, can extend their lifespan. Follow recommended maintenance procedures to get the most out of these components.

Ultimately, the replacement schedule for ground engaging tools should be determined by a combination of these factors, manufacturer recommendations, and the specific needs of your operation. Regular inspections and proactive maintenance are key to ensuring that GET are replaced in a timely manner, preventing excessive wear that could lead to more significant equipment damage or reduced productivity.

As more data regarding material hardness and GET replacements intervals is gathered by the mine personnel, more accurate replacement schedules will be able to be provided.

In the interim, GET replacement schedules based on the manufacturer's recommendations are used, as indicated in Table 55 below.

Table 55: Ground Engagement Tools (GET) Replacement Intervals

Equipment	Item	Replacement Interval (hours)
Excavator	Bucket Teeth Bucket Pins and Bushes	500 2,000
Loader	Bucket Teeth Bucket Pins and Bushes	500 1,000
Grader	Blade Cutting Edge Ripper Teeth	500 500
Bulldozer	Blade Cutting Edge Ripper Teeth	500 - 1,000 1,000

16.9.2 Capital Spares

Capital Spares - are classified as pieces of equipment, or a spare part, of significant cost that is maintained in inventory for use in the event that a similar piece of critical equipment fails or must be rebuilt. Capital spares cost has been included in the costing for the Expansion Project as stipulated by the equipment manufacturers.

Radiators for all types of EME.

Loaders – Bucket. To be replaced every 4,000 hours.

Excavator – Bucket. To be replaced every 4,000 hours.

Excavator and Drill Rig track chain links and sprockets. To be replaced every 4,000 hours.

Hydraulic Motors and Pumps on all EME, exchange expected at/or before 5,000 hours.

Engine per machine type.

Transmissions per machine type.

Differential and final drive per machine type.

Hydraulic Cylinders per machine type.

Splitter Box for Excavators, Dozers, Loaders and Graders.

TAIYE drill final drive.

Drill, Excavator and Dozer track chain links and sprockets.

Rotary motors and pumps.

16.9.3 Fuel

16.9.3.1 Consumption Schedule

Table 56: Equipment Fuel Consumption

Number	Description	Fuel Consumption (L/Hr)	Fuel Consumption Base
1	Sany SMG200 Grader	15.00	Sany SA
2	Front End Loader - SYL956H5 ($3m^3$ Bucket)	20.00	Sany SA
3	Front End Loader - SYL956H6 ($4m^3$ Bucket)	20.00	Sany SA
4	Excavator - SY245H	28.33	Sany SA
5	Excavator - SY750H	44.75	Sany SA
6	Shantui Dozer - SD 22W	22.25	D7E
7	tipper $26m^3$ - Sinotruk Howo 7 84371	19.00	ADT 730C2
8	Water Truck 20 000L - Sinotruk Howo 64290	19.00	ADT 730C2
9	Single Drum Roller - Sany SSR100AC-8	12.00	Sany
10	Diesel Trailer (2000 Litre) towed by LDV	15.00	LDV Diesel

Number	Description	Fuel Consumption (L/Hr)	Fuel Consumption Base
11	Drill - TAIYE X45 DTH	37.00	Cat Equiv.
12	Hyundai County Bus – Diesel	25.00	M/B Equiv.
13	Service Trailer - Towed by LDV	15.00	LDV Diesel
14	Supervision Vehicles	15.00	LDV Diesel
15	Metso HM75 pump, Hatz 4L41C Diesel Engine	10.00	Hatz
16	Dewatering Pump - PAS 150 MF 250 (540m³/hr)	6.60	Atlas Copco
17	Lighting Plant - Kubato	1.86	Atlas Copco

16.9.4 Re-fuelling Station

The re-fuelling station will be centralised on site and will be utilised by the plant and mine LDV fleet and the mining haul truck fleet. The re-fuelling station is located on the plant terrace central to the site. The re-fuelling station will have a direct feed from the diesel storage facility on site and no provision has been included for any storage of diesel at the re-fuelling station.

Open pit mining equipment that will require re-fuelling via a diesel trailer are the front-end loaders, excavators, bulldozers, blast-hole drills, lighting plants and dewatering pumps. These vehicles and equipment operate in remote locations within the mining site, making it impractical for them to re-fuel at a fixed re-fuelling station. Instead, they rely on mobile diesel trailers equipped with fuel tanks to provide them with the necessary diesel fuel. This -process is essential for ensuring the continuous operation of the mining equipment, allowing for efficient material extraction and transport in open pit mining operations. It involves safely connecting the trailer to the equipment and transferring fuel to maintain optimal performance and productivity. Regular re-fuelling schedules are crucial to prevent downtime and maintain the smooth operation of the mining site. (Figure 60).

Figure 60: Diesel Trailer – 200 Litre

16.10 Waste Rock Storage

Waste rock from the mining operation, estimated at 1.6 Mt LoM, will be transported to a designated stockpile area near the processing plant. It is envisaged that the material will be crushed when required loaded and transported on the conveyor system with the fine tails from the processing plant to the tailings storage facility. Detailed design of this operation will be done during the detailed design phase for the Expansion Project.

The waste rock is necessary for the construction of the TSF and forms part of the outer wall of the facility to improve stability.

The TSF is to be developed in phases leading from the eastern side, where the fixed conveyor line and spreader is located. The phases are a gradual development, spreading from the east in layers and gradually filling the basin by loading and spreading of the material. The TSF is then to be raised in these layers until final height (approximately 60m) is achieved.

Waste rock is identified as the primary source of structural wall building material, but needs to be limited to a maximum of 300 – 400 mm in size. This material will be placed within the TSF as indicated in Figure 61 below.

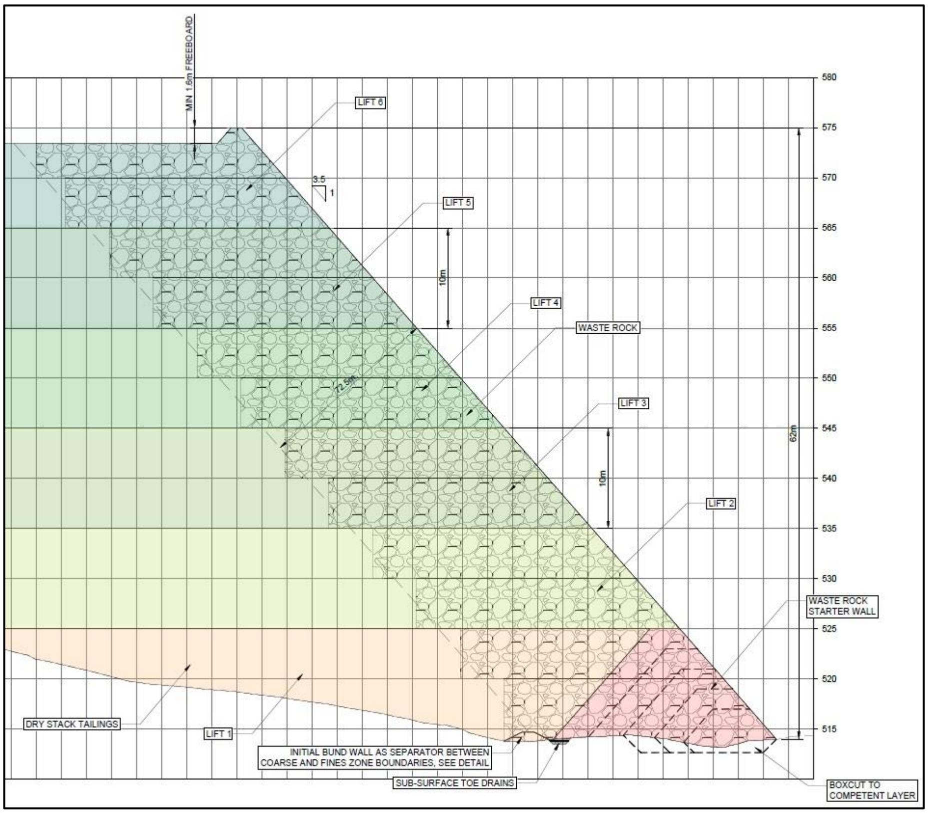


Figure 61: TSF Typical Section

16.11 Pit Dewatering

Ground water numerical modelling suggested a ground water inflow rate of 8 to 590 m^3/d for different mining stages over the operational life. The average ground water and storm water volume likely to be reporting to the open pit sump over the Life of Mine is about 900 m^3/d.

Mine storm water run-off areas were defined, and storm water management measured, was proposed that would mitigate potential surface water risks and ensure compliance with relevant legislation and guidelines in terms of environmental and work safety aspects. Flood peaks were calculated that would facilitate the final design of levees and drainage channels required to contain and control the flow of both “clean” and “dirty” water on the site.

Three lined PCDs (Pollution Control Dams) were identified and sized, i.e. pit / reject PCD, plant PCD and co-disposal PCD. Silt control measures were allowed for on all PCDs.

The potential impact that mining activities are likely to have on surface and ground water resources in the surrounding environment was analysed. These potential impacts were quantified, and a significance rating was undertaken.

Potential impacts relate to mine dewatering, wellfield abstraction, dirty water run-off, run-off reduction, contaminant seepage from the co-disposal facility and reject dump. None of the identified potential impacts posed a high, or serious risk to the downstream environment, and the majority of risks were mitigated by the Water Management Plan, which includes appropriate storm water management measures, dry tailings disposal and post closure residual mine waste capping layers.

As the mining operation expands, the pit will be divided into 4 face areas, as indicated under "Mining Sequence". A sump will be established at the lowest point of every face, to which any water influx into the pit will flow. Each sump will be serviced by an "Atlas Copco PAS 150 MF 250" dewatering pump, capable of pumping 540m^3/hour, with a suction head of +/- 7m and a vertical delivery head of 37m. The water from the 4 sumps will be pumped to a transfer tank, which will be situated at the base of the ramp, from where the water will be pumped to the pollution control dams, as all in pit water is classified as dirty water. The transfer tank will have to be fitted with an electrical pump, capable of a vertical delivery head of 220m and a volume of 2.2 ML/h.

A number of hydrogeological boreholes were drilled and tested to investigate local aquifer conditions. The main aquifer zone is hosted within the weathered and fractures rock in the upper 50m below surface. The gneiss at the proposed mine site has low aquifer potential, with the marble layers to the west having a higher aquifer potential.

A hydro-geochemical assessment was undertaken, including laboratory test work and modelling, on tailings material and representative waste rock. The results indicate that most Sulphur associated with ore, tailings and waste rock consists of secondary sulphate minerals such as jarosite. The reaction rates associated with the secondary sulphate minerals are generally slow, resulting in relatively low sulphate loads in rainfall run-off and short-term operational seepage associated with the mine waste facilities. However, higher sulphate loads could be expected in the long-term, specifically post-closure. The pH of water emanating from waste facilities will be neutral over the short term, potentially becoming slightly acidic in the long term. Metal concentrations will be low.

Appropriate closure rehabilitation measures are foreseen to manage the long-term post closure water quality emanating from residual waste product.

16.12 Mining Operations

16.12.1 Drilling and blasting

16.12.1.1 Different Blast Designs in Different Material

As the Initial topography of the pit area is very uneven, blast holes will have to be individually designed, as depth varies across the block to be blasted. For this purpose, blasts are designed in Deswik OPDB (Open Pit Drill and Blast) software to achieve the required powder factor, column rise and stemming for each hole. Refer to Figure 62 to Figure 65 below.

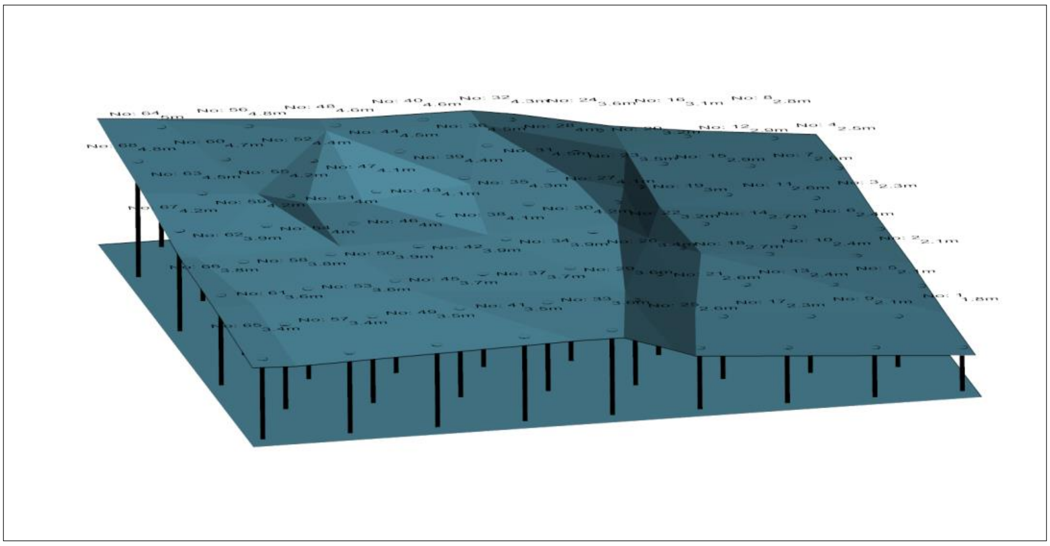


Figure 62: Deswik – Blast design per blast hole for 30m x 30m block

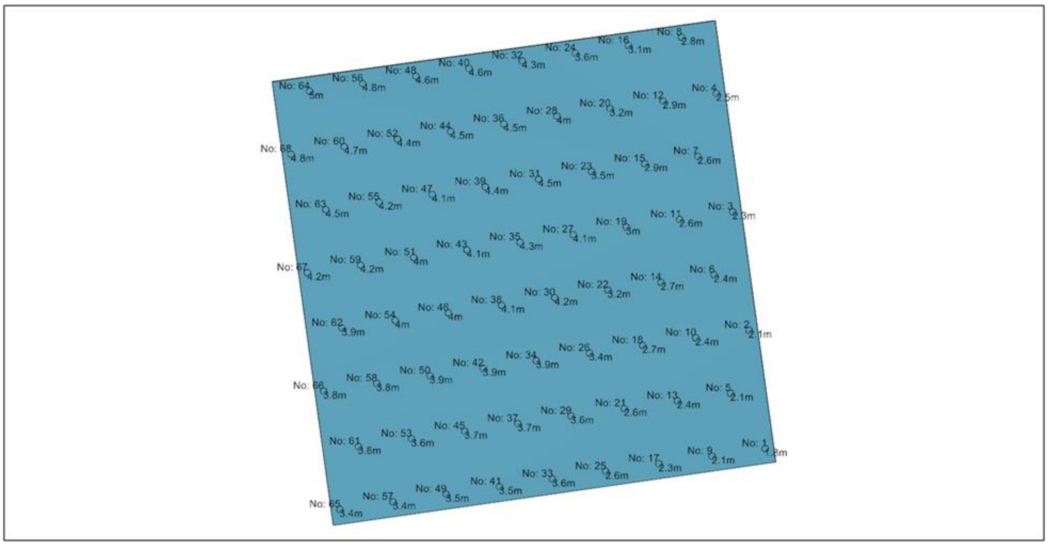


Figure 63: Deswik – Blast Hole Layout per 30m x 30m block

The drill and blast design parameters are as follows:

- The single design is applicable to both ore and waste:
  - Waste density = 2.65 / Ore Density = 2.36.
  - UCS range = 15 to 32 MPa.
- The drill and blast products assumed are:
  - Anfo.
  - 150g Boosters.
  - Electric Detonators.

    - Detonating cord.
    - Noiseless Trunk line Delays (NTD's), 25 ms between holes and 42 ms between rows.
    - Down hole delays – 600 ms.
    - Basting wire.
- The drill and blast parameters:
    - Hole diameter = 115 mm.
    - Bench height = based on Topo thickness.
    - Stemming length = approximately 30% of blast hole volume.
    - Burden = 3.6m.
    - Spacing = 4m.
    - Sub-drill = 0.4m.
    - Explosive density 0.85 g/cm³.
    - Powder factor = 0.43.
    - Charge mass = Dependent on blast hole depth.

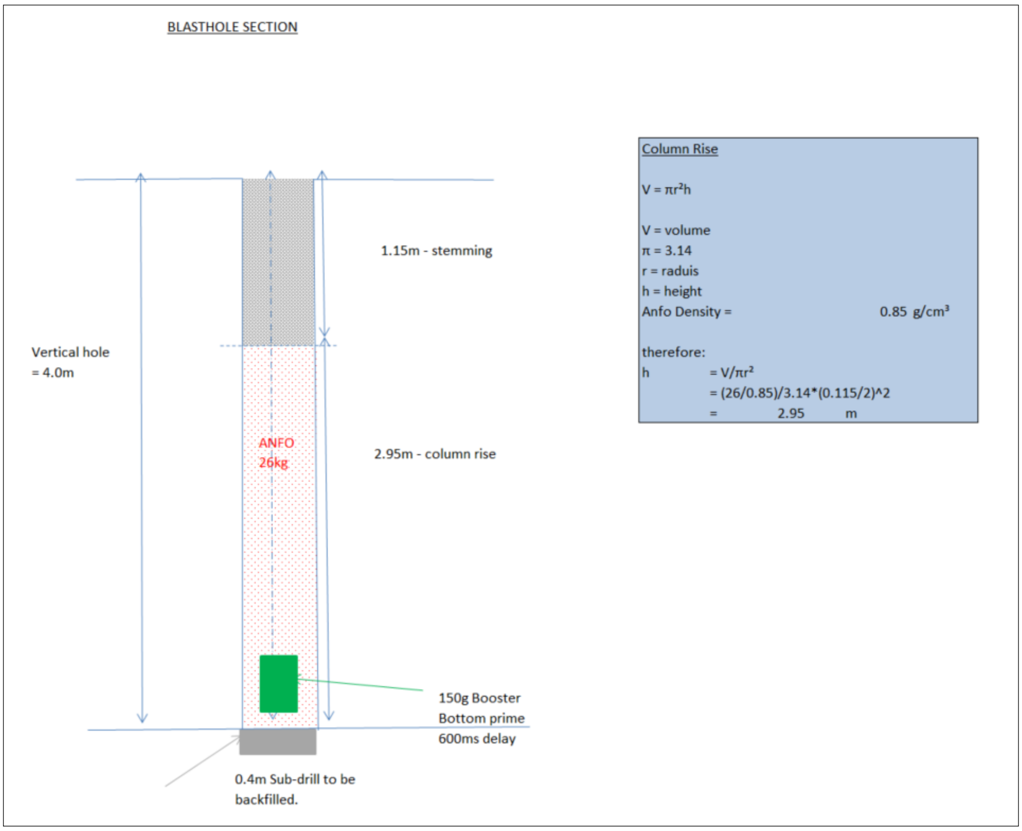


Figure 64: Blast Hole Charge placement and Bottom Priming using 600 ms down hole delay

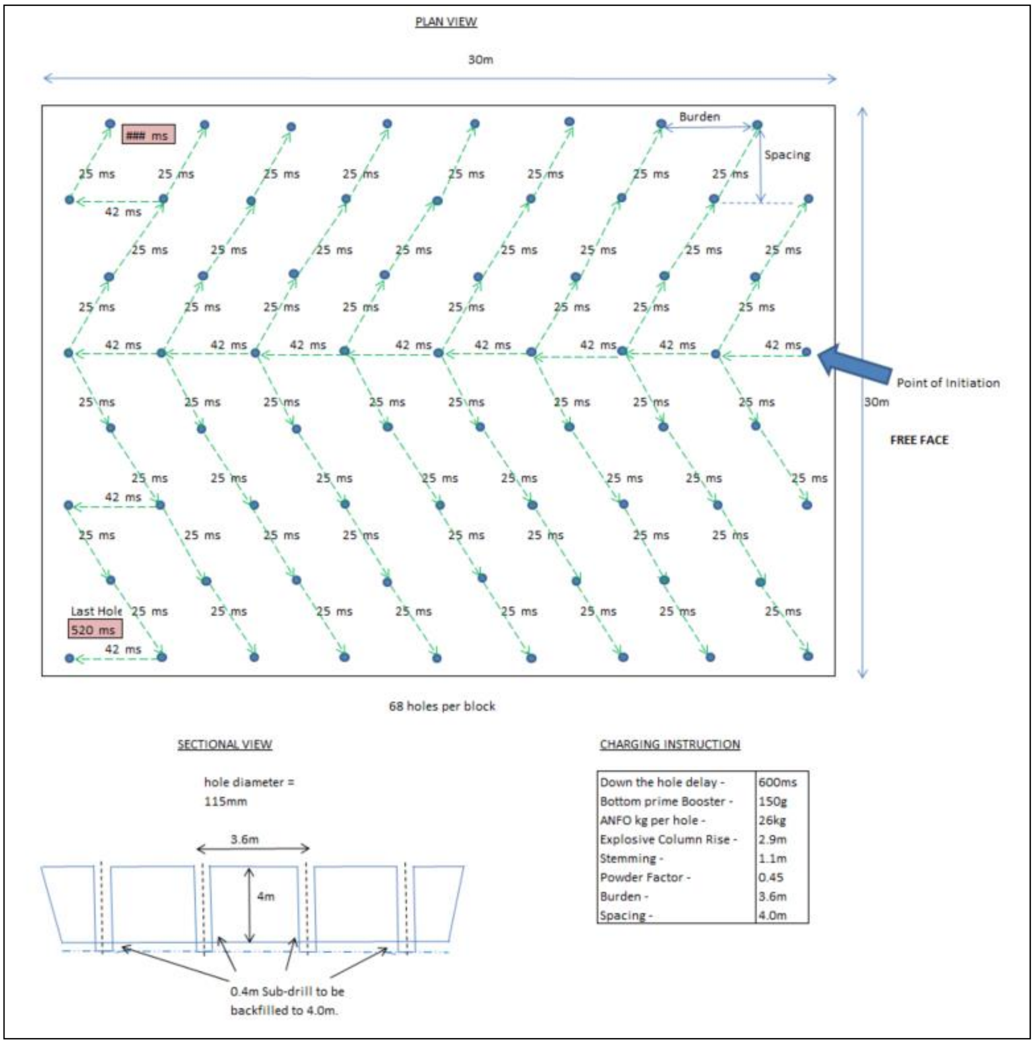


Figure 65: Blast Block surface timing

Table 57: Charging Instruction

CHARGING INSTRUCTION								
Hole No	**Drill Depth (m)**	**Burden (m)**	**Spacing (m)**	**P.F.**	**Explosive (kg)**	**Column Rise (metres)**	**Stemming (m)**	**Stem (%) of hole depth**
1	1.8	3.6	4.0	0.43	11	1.3	0.5	30%
2	2.1	3.6	4.0	0.43	13	1.5	0.6	30%
3	2.3	3.6	4.0	0.43	14	1.6	0.7	30%
4	2.5	3.6	4.0	0.43	15	1.8	0.7	30%
5	2.1	3.6	4.0	0.43	13	1.5	0.6	30%
6	2.4	3.6	4.0	0.43	15	1.7	0.7	30%

A charging instruction is calculated for every blast, which includes the explosive charge required for every hole in the block to be blasted.

The drill and blast design parameters are as follows:

- The design is applicable to blocks with a mixture of fresh rock and saprolitic rock:
  - Waste density = 2.65 / Ore Density = 2.36.
  - UCS range = 15 to 32 MPa.
- The drill and blast parameters:
  - Hole diameter = 115 mm.
  - Bench height = based on Topo thickness.
  - Stemming length = approximately 48% of blast hole volume.
  - Burden = 3m.
  - Spacing = 3m.
  - Sub drill = 0.4m.
  - Explosive density 0.85 g/cm^3.
  - Powder factor = 0.50.
  - Charge mass = Dependent on blast hole depth.

If there is a mixture of fresh and weathered material in the blast block, penetration rate contours achieved whilst drilling, will be plotted, and all blast designs done accordingly. As explosive energy always takes the “easiest or shortest” route, the explosive charged will be placed in the vicinity of the hard material, using deck charging. In this case, an additional booster will be required, termed double priming.

Refer to Figure 66 below.

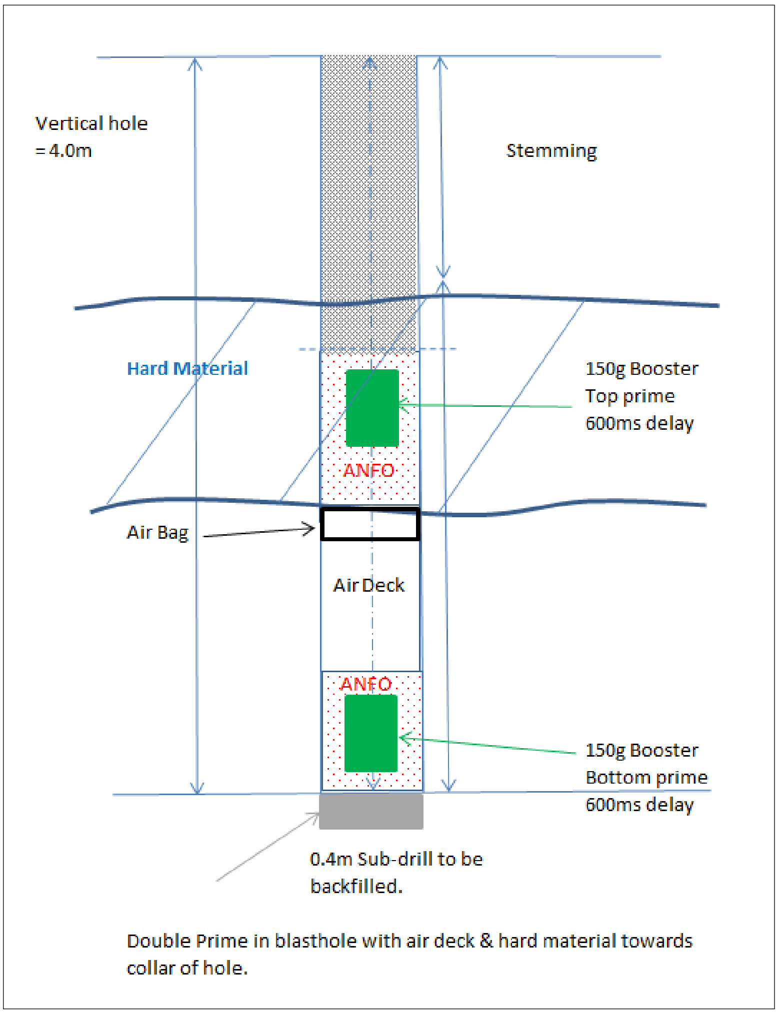


Figure 66: Blast Hole Charge double priming with air deck. Placement of charge in hard material

16.12.2 Explosive Consumption and Storage

The average explosive and accessory consumption per month is shown in Table 58. Only two companies in Madagascar are allowed to import explosives and accessories into the country, with a lead time of approximately 3 months from date of order, thus the mine will keep 4 months explosive and accessory consumption at the explosive magazine.

Refer to Table 58 below.

Table 58: Explosive & Accessory, Average Consumption per Month

Name	Unit	Monthly Average
Total Mining Tonnes	t	**249,291**
Waste Component	t	29,290
Ore Component	t	220,001
Total Blast Volume	**m^3**	**104,662**
Waste Component	m^3	11,441
Ore Component	m^3	93,221
Explosive and Accessory Components		
Anfo @ 0.45 P.F. (kg per m^3)	kg	51,808
Down Hole Delays 600 ms	Each	1,999
Surface Connector with 25 ms Delay	Each	1,791
Surface Connector with 42 ms Delay	Each	352
150 g Boosters	Each	1,999
Primadet Lead-in-line	M	15,990
Initiation Detonators	Each	262
30 metre x 30 metre Blocks	Each	29.1

16.12.3 Initiation

An electric detonator is connected to the surface lead in trunk line, which must have a minimum length of 500m from the edge of the blast block. The electric detonator is then further connected to an electric cable, which, in turn, is linked to a blasting battery used to initiating the blast.

16.12.4 Loading

The loading operation in an open pit mine, involves the removal of overburden, (waste material), and the extraction of ore. The two primary pieces of heavy equipment used for loading in the open pit mine are front end loaders (FEL) and excavators (Exc).

To achieve a stable average Ore Grade from the pit and reach the mine production targets, 4 loading faces are required. Two faces utilizing Sany SYL956H FEL's with 4m³ buckets and two faces utilizing Sany SY750H Excavators with 4.5m³ buckets.

The "Pit Shell" was created using 8m high benches, with 6m offsets in weathered rock and 8m offsets in fresh rock. For safety purposes, the FEL needs to be able to place the bucket on top of the face being loaded, thus ensuring that it need not undercut the face during the loading process. As the Sany SYL956H FEL can raise the base of the bucket to 4.2m height, the 8m high bench is mined in 2 x 4m lifts. Refer to Figure 67 below.

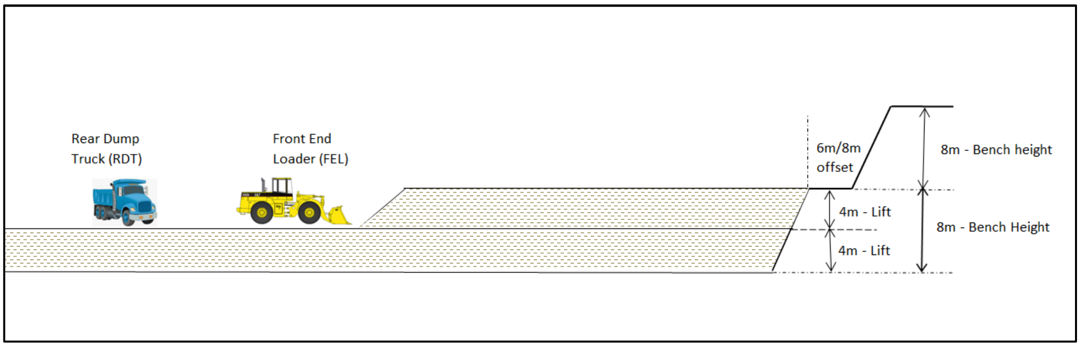


Figure 67: Front End Loader, loading a Rear Dump Truck

The Sany SYL956H6 FEL will load the blasted material, (muck pile), from pit floor elevation, and is fitted with a 4m³ bucket, which will be capable of loading the 26m³ rear dump truck (RDT) in 7 passes. This is slightly higher than the "mining rule of thumb" for matching load and haul equipment, of 4 to 6 passes, however, will be productive enough to achieve Phase 2 mining targets.

The 3 x Sany SYL956H6 FEL's with 3m³ buckets, were purchased for Phase 1 of the mining operation, and matched with the 18m³ RDT's. However, as the 18m³ RDT's were not available, the 26m³ RDT's were purchased creating a mismatch in the loading configuration. As the Sany SYL956H6 FEL has a rated load capacity of 5t, it was decided not to upgrade the bucket to larger than 4m³.

- Rated load capacity of 4m³ bucket:
  - Ore bulk density = 1.56 t/m³.
  - Fill Factor = 90%.
  - Thus, Load mass = 4m³ x 1.56 t/m³ x 90% = 5.6t per bucket load.

This will involve adding an additional counterweight at the rear of the FEL, which will put more stress on the equipment's frame, and will have to be monitored by the engineering department.

The Sany SY750H with 4.5m³ bucket, will be positioned on the top of the muck pile, and load the RDT's in a drive-by configuration, which reduces RDT spotting time, and makes the operation very productive, if compared to other loading configurations.

The "Pit Shell" was created using 8m high benches, with a 6m off-sets in weathered rock and 8m off-sets in fresh rock. The mining operation is performed in 2 x 4m lifts, to facilitate the Sany SY750H having a maximum digging depth of 7.6m. Refer Figure 68 below.

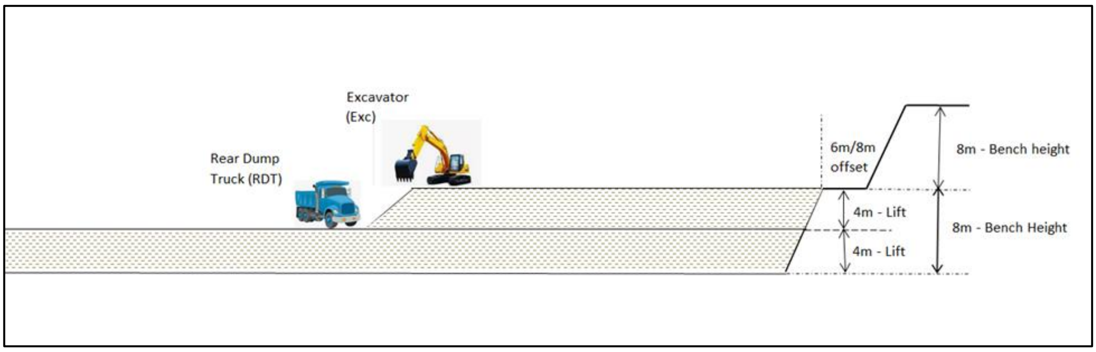


Figure 68: Excavator, loading a Rear Dump Truck

- Rated load capacity of 4.5m³ bucket:
  - Ore bulk density = 1.56 t/m³.
  - Fill Factor = 90%.
  - Thus, Load mass = 4.5m³ x 1.56 t/m³ x 90% = 6.3t per bucket load.
  - No of passes to load 26m³ RDT.
  - Exc. bucket capacity = 4.5m³.
  - RDT bucket capacity = 26m³.
  - No of passes = 26m³ ÷ 4.5m³ = 5.8, i.e., **6 passes**.

The excavator bucket size of 4.5m³ allows it to load the 26m³ RDT's in 6 passes, which falls within the 4 to 6 pass theory, ensuring the loading operation is productive.

16.12.5 Hauling

The number of trucks required to achieve the mine production target is determined from the LoM Monthly Schedule.

For the purpose of calculating RDT productivities to be included in the Deswik Schedule, 4 time frames evenly spread across the LoM schedule where selected, and productivities calculated over the measured haul distances. (As per Section 16.10 – Equipment Matching Philosophy, Haul Profiles, Cycle Times and Calculations).

16.12.6 Calculation to Determine the Number of RDT's Required

Equipment Units = SchedROMT ÷ Calendar Effective Time ÷ Equipment Productivity (tph).

16.12.7 Ex-pit Production

Based on the maximum number of RDT's required to achieve the "Mine Production Target", 12 x rear dump trucks are allocated to ex-pit mining, 3 per loading face.

16.12.8 ROM Stockpile Blending

All Ex-pit RoM tons will be hauled to the ROM stockpile and placed on either the Low Grade (Large Flake), or High Grade (Fine Flake) stockpiles, in accordance with the block flag in the Geological Model.

Based on the blend required by the Process Plant, 1 x Sany SYL956H FEL will load 2 x Sino Howo 26m³ RDT's from the Low Grade and High Grade stockpiles and the RDT's will haul the ore to the Process Plant tip.

Table 59: Rear Dump Truck Productivity from ROM Stockpile to Tip

RDT PRODUCTIVITY															
Description	% Avail	% Util	Hours/ Day	Average Speed Empty (km/h)	Average Speed Full (km/h)	Haul Distance Pit to Tip (km)	Cycle Time Empty (min)	Cycle Time Full (min)	Loading Time (min)	Tipping Time SP (min)	Total Cycle Time SP (min)	Load Body Capacity (m³)	Load Body Capacity (tons)	Ton per Hour	Tons per Day
RDT 26m³	85%	70%	14.3	16.6	8.1	0.24	0.9	1.8	5.4	0.7	8.8	26	27	250	3,654
Loading Unit = SANY FEL 956H 4m³ Bucket															

Table 60: Haul Distance from ROM Stockpile to Tip

Haul Distance ROM Stockpile to Tip	
Truck Haul - S/P –Tip	**Distance (m)**
Flat on Bank	120
Up/down Ramp	110
Flat on Pad, Tip	10

As the Process Plant will operate on 24 hours x 7 days per week basis, the equipment loading from the stockpile and hauling to the Process Plant tip will be required to operate the same hours as the Process Plant.

16.13 Shift Structure and Personnel Requirements

Table 61: Mining Compliment

Mining Compliment		
Open Pit 2 Shift Cycle, 2 x 8 Hour Working Shifts, 5 Days a Week		
Designation	**Compliment**	**Patterson Scale Category**
Mining Manager	1	D4
Mining Superintendent	1	D2
Mining Services Foreman	1	C5
Mining Shift Foreman	2	C5
Blasting Technician / Magazine Master	2	C1
Blasting Assistants	4	B1
Operator – Grader	4	B3
Operator – Front End Loader (FEL)	4	B3
Operator – Excavator (Exc)	4	B3
Operator – Dozer	6	B3
Operator – Rear Dump Truck (RDT)	24	B3
Operator – Water Truck	4	B3
Operator - Compactor	4	B3
Operator – Diesel / Bowser Trailer	4	B3
Driver - LDV	6	B3
Operator - Drill	6	B3
Road and Ramp Construction Crew	3	B3
Drill Assistants	6	B1
Service Assistants	4	B1
Operator - Multi Skilled Relief	12	B3
Total	**102**	
ROM Stockpile 4 Shift Cycle, 3 x 8 Hour Working Shifts, 7 Days a Week		
Operator – Front End Loader (FEL)	4	B3
Operator – Rear Dump Truck (RDT)	8	B3

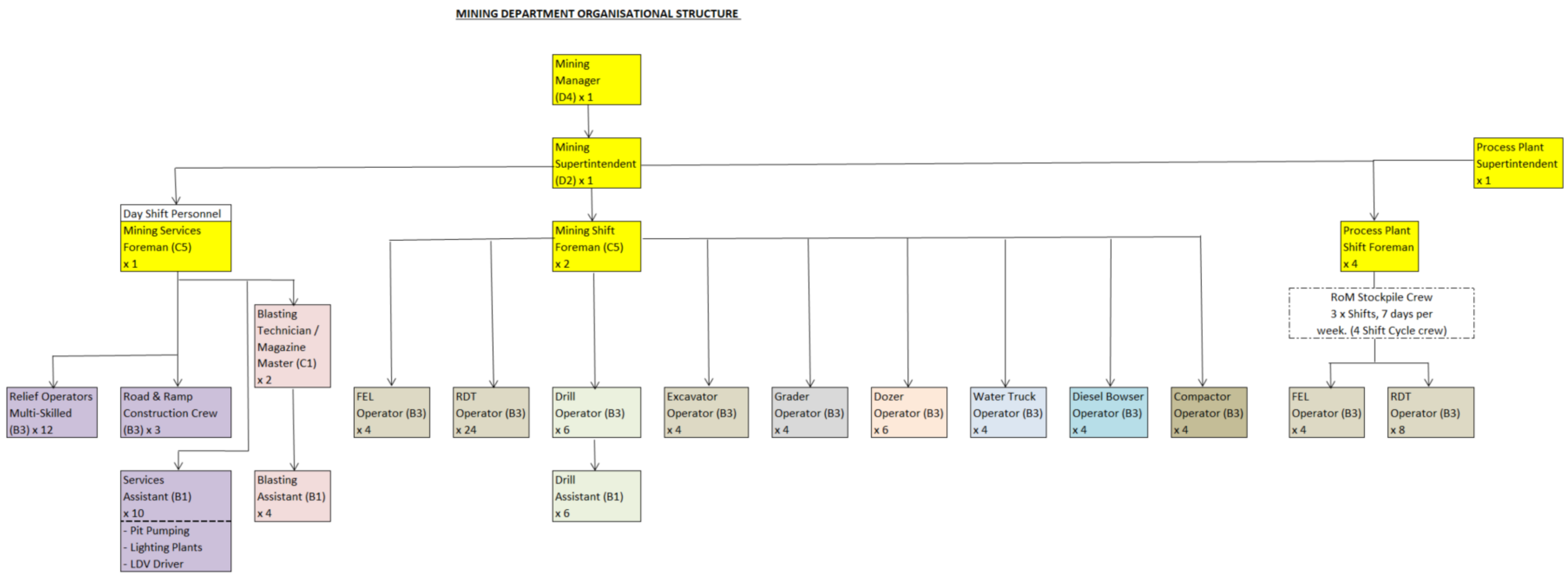


Figure 69: Mining Department Org Chart

Table 62: Technical Services Department - Labour Compliment

Mining Technical Services Department Labour Compliment		
Designation	**Compliment**	**Patterson Scale Category**
Technical Manager	1	D4
Surveyor	1	D2
Geologist	1	D2
Mine Planning Engineer	1	D2
Geological Technician	2	B5
Geological Grade Control	2	B5
Survey Assistant	2	B3
Mine Planning Clerk	2	B3
Total	**12**	

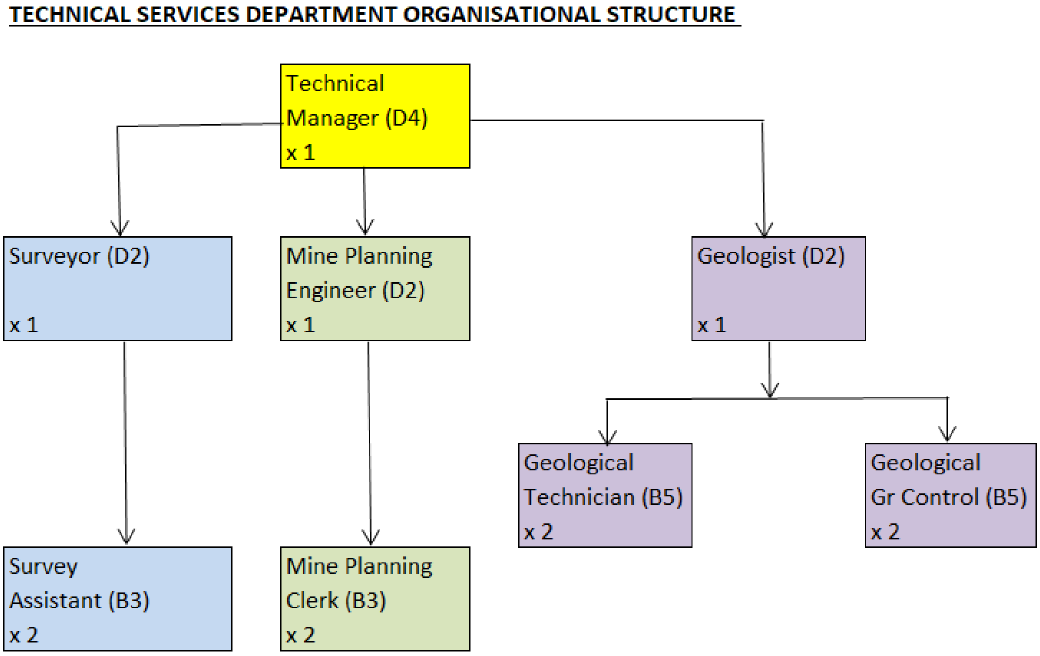


Figure 70: Technical Services Department – Org Chart

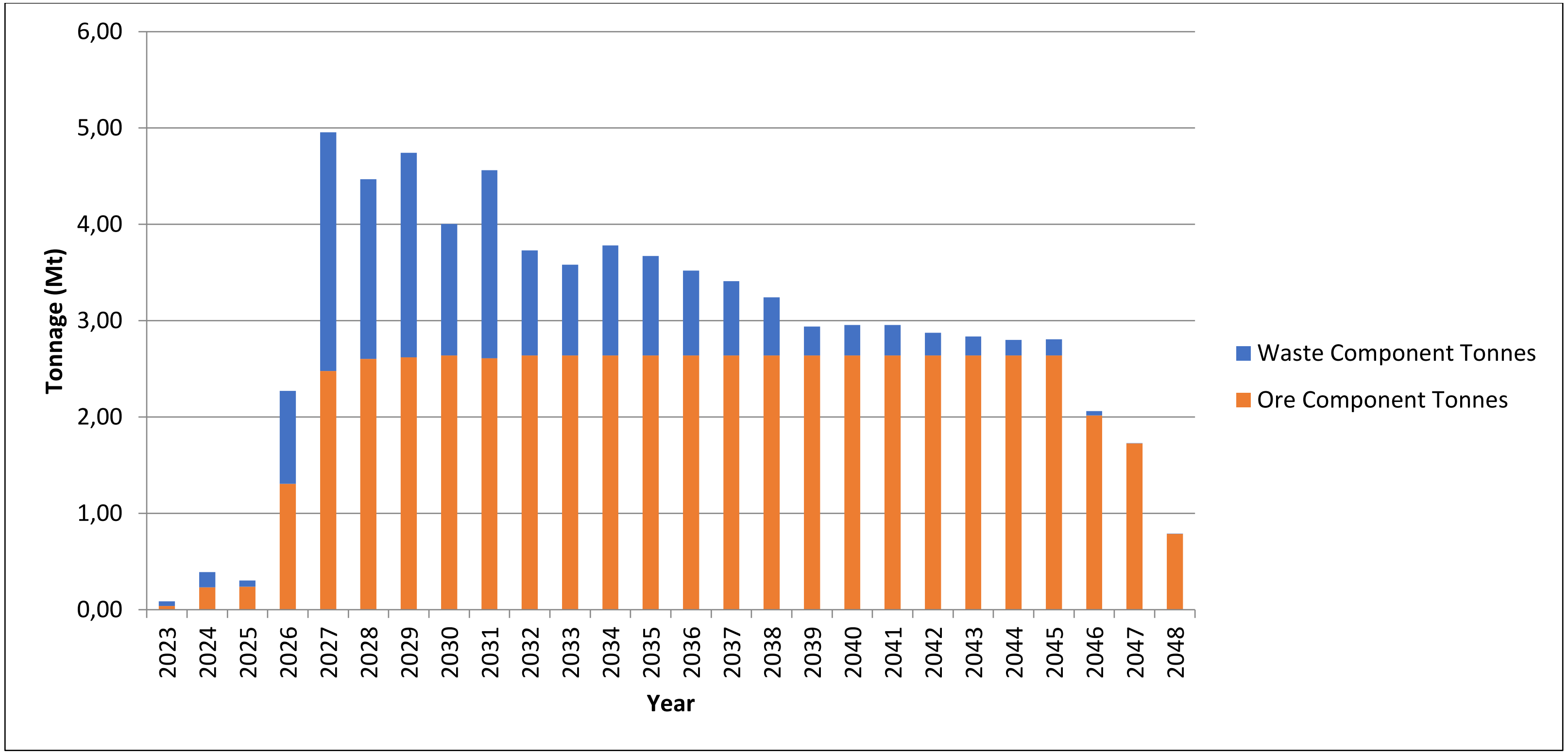
16.14 Life of Mine Production

Table 63: Life of Mine Production Schedule Annually

			Period	1	2	3	4	5	6	7	8	9	10	11	12	13
Name	Description	Field	Row total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Topsoil operation		SchedROMT	1678338	60230	296033	59998	1049290	212786	0	0	0	0	0	0	0	0
Waste component		H_WasteROMT	1073428	40952	153428	59998	638422	180628	0	0	0	0	0	0	0	0
Ore component		F_OreToPlant	604910	19279	142605	0	410868	32158	0	0	0	0	0	0	0	0
Mining operation		SchedROMT	73785693	26749	93455	243238	1222320	4742889	4469886	4742271	4005365	4562115	3730679	3580402	3782512	3672096
Waste component		H_WasteROMT	18124801	6028	2620	3238	325276	2296447	1864823	2122450	1365369	1951430	1090681	940414	1142516	1032098
Ore component		F_OreToPlant	55660892	20721	90834	240000	897043	2446442	2605063	2619821	2639996	2610685	2639998	2639988	2639997	2639999
Total mining tonnes			75464031	86980	389488	303236	2271610	4955676	4469886	4742271	4005365	4562115	3730679	3580402	3782512	3672096
Waste component			19198229	46980	156048	63236	963698	2477075	1864823	2122450	1365369	1951430	1090681	940414	1142516	1032098
Ore component			56265802	40000	233440	240000	1307911	2478601	2605063	2619821	2639996	2610685	2639998	2639988	2639997	2639999
RoM Stockpile			25000	25000	25000	25000	25000	25000	25000	25000	25000	25000	25000	25000	25000	25000
Plant Feed incl contamination		A_PlantFeedROMT	56265802	40000	233440	240000	1307911	2478601	2605063	2619821	2639996	2610685	2639998	2639988	2639997	2639999
Ore Head Grade C		Calculation	6.07	8.27	7.98	7.78	5.86	6.12	6.22	6.18	5.95	6.18	6.34	6.17	6.36	6.34
Contained Graphite		Calculation	3413620	3307	18640	18661	76634	151807	162137	161775	157017	161446	167363	162780	167972	167462
Mined - Ore Classification %			100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
% Proven Reserve - a part of the Measured Resource			42%	100%	69%	75%	89%	71%	71%	73%	78%	71%	64%	72%	67%	53%
% Probable Reserve - a part of the Indicated Resource			54%	0%	29%	22%	5%	22%	18%	20%	20%	21%	29%	24%	30%	42%
% Possible Reserve - a part of the Inferred Resource			4%	0%	2%	3%	6%	6%	12%	8%	1%	9%	7%	4%	3%	5%

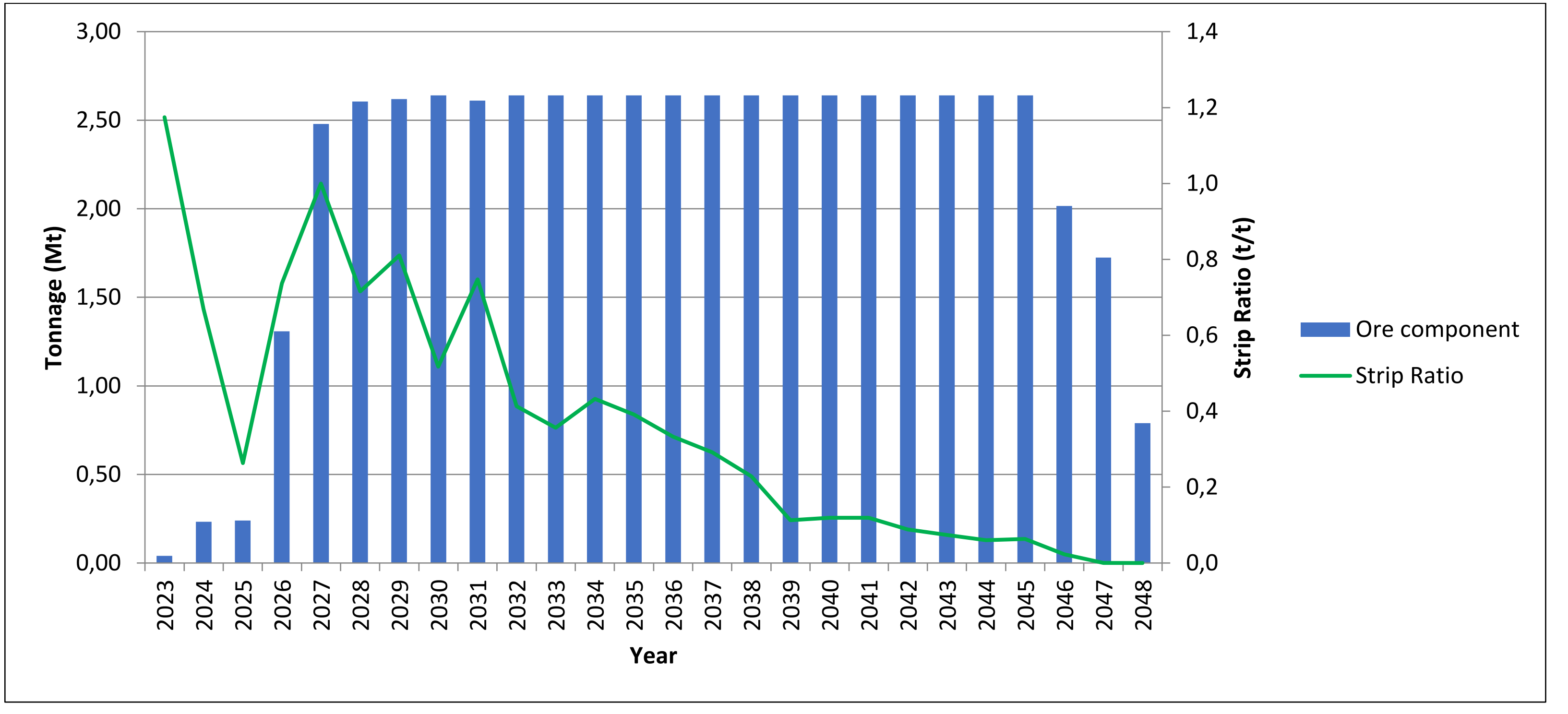
			Period	14	15	16	17	18	19	20	21	22	23	24	25	26
Name	Description	Field	Row total	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Topsoil operation		SchedROMT	1678338	0	0	0	0	0	0	0	0	0	0	0	0	0
Waste component		H_WasteROMT	1073428	0	0	0	0	0	0	0	0	0	0	0	0	0
Ore component		F_OreToPlant	604910	0	0	0	0	0	0	0	0	0	0	0	0	0
Mining operation		SchedROMT	73785693	3519682	3409848	3242830	2938565	2953964	2954664	2874198	2835849	2799081	2806552	2062661	1725106	788713
Waste component		H_WasteROMT	18124801	879682	769856	602832	298569	313963	314665	234201	195851	159085	166553	46148	4	0
Ore component		F_OreToPlant	55660892	2640000	2639991	2639998	2639996	2640001	2639999	2639997	2639998	2639996	2639999	2016513	1725102	788713
Total mining tonnes			75464031	3519682	3409848	3242830	2938565	2953964	2954664	2874198	2835849	2799081	2806552	2062661	1725106	788713
Waste component			19198229	879682	769856	602832	298569	313963	314665	234201	195851	159085	166553	46148	4	0
Ore component			56265802	2640000	2639991	2639998	2639996	2640001	2639999	2639997	2639998	2639996	2639999	2016513	1725102	788713
RoM Stockpile			25000	25000	25000	25000	25000	25000	25000	25000	25000	25000	25000	25000	25000	25000
Plant Feed incl contamination		A_PlantFeedROMT	56265802	2640000	2639991	2639998	2639996	2640001	2639999	2639997	2639998	2639996	2639999	2016513	1725102	788713
Ore Head Grade C		Calculation	6.07	6.37	6.15	6.15	6.28	6.19	5.80	5.76	5.82	5.57	5.95	5.65	5.63	5.12
Contained Graphite		Calculation	3413620	168251	162382	162465	165825	163479	153189	151969	153645	146959	157042	113906	97125	40382
Mined - Ore Classification %			100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
% Proven Reserve - a part of the Measured Resource			42%	34%	26%	20%	18%	16%	15%	6%	3%	0%	0%	0%	0%	0%
% Probable Reserve - a part of the Indicated Resource			54%	61%	70%	75%	79%	79%	82%	90%	95%	98%	98%	99%	100%	100%
% Possible Reserve - a part of the Inferred Resource			4%	5%	4%	6%	3%	5%	3%	3%	2%	2%	2%	1%	0%	0%

The following Graph 35 to Graph 38 shows the results of the LoM production schedule. The graph below shows the waste component required to be moved in order to access the graphite ore component required to be fed to the Process Plant, on an annual basis.



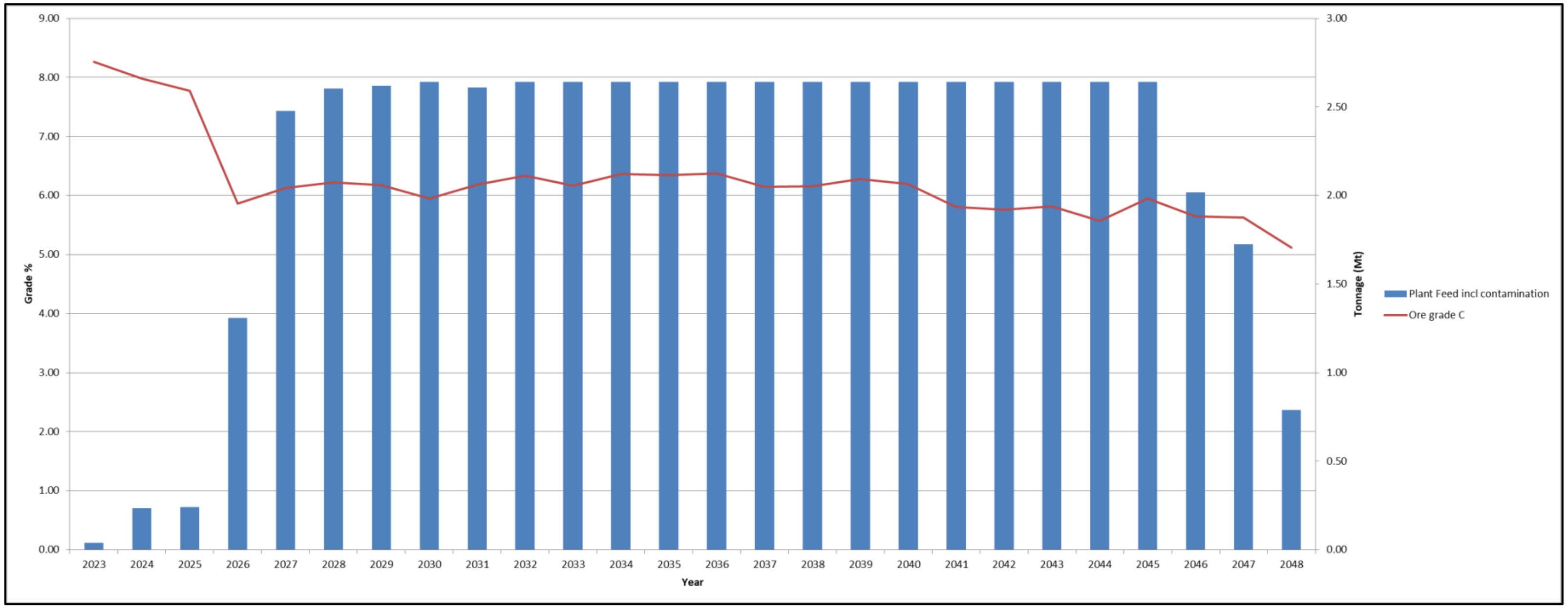
Graph 35: Waste and Ore Tonnages per year

The strip ratio is a common metric used to express the relationship between waste and ore in open-pit mining. It is defined as the ratio of waste material removed to the amount of ore removed. A lower strip ratio indicates a more efficient mining operation, as less waste needs to be removed for each unit of ore extracted. Molo's low strip ratio decreases over the LoM as shown below.



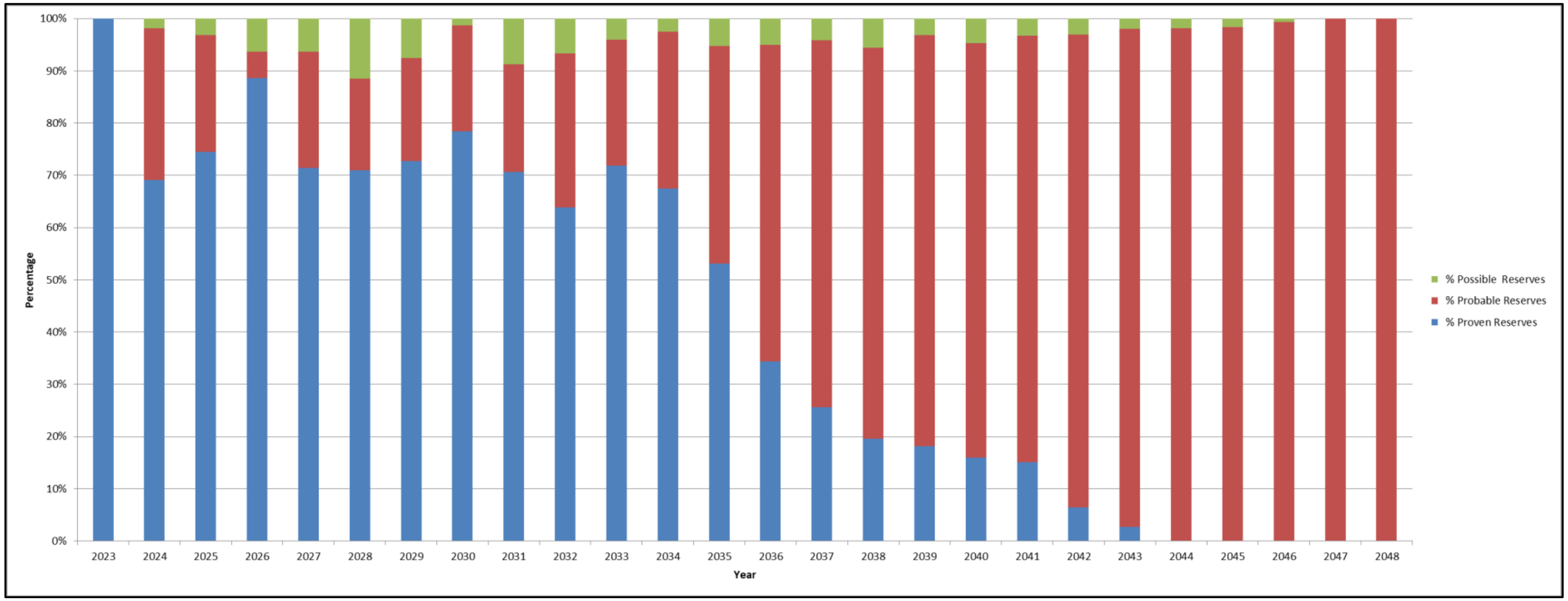
Graph 36: Ore Tons and Strip Ratio per year

The graph below shows the ore tonnages for each year, as well as the average Head Grade (C) for the ore fed to the Process Plant.



Graph 37: Plant Feed and Tonnes and Ore Grade C per year

The ore class percentile graph below shows the ratio of the reserve classification extracted from the pit to make up the ROM feed.



Graph 38: Ratio of the reserve classification extracted from the pit to make up the ROM feed per year.

The following layouts (Figure 71 to Figure 74) shows the mine development at different periods over the LoM and the working areas of each excavator.

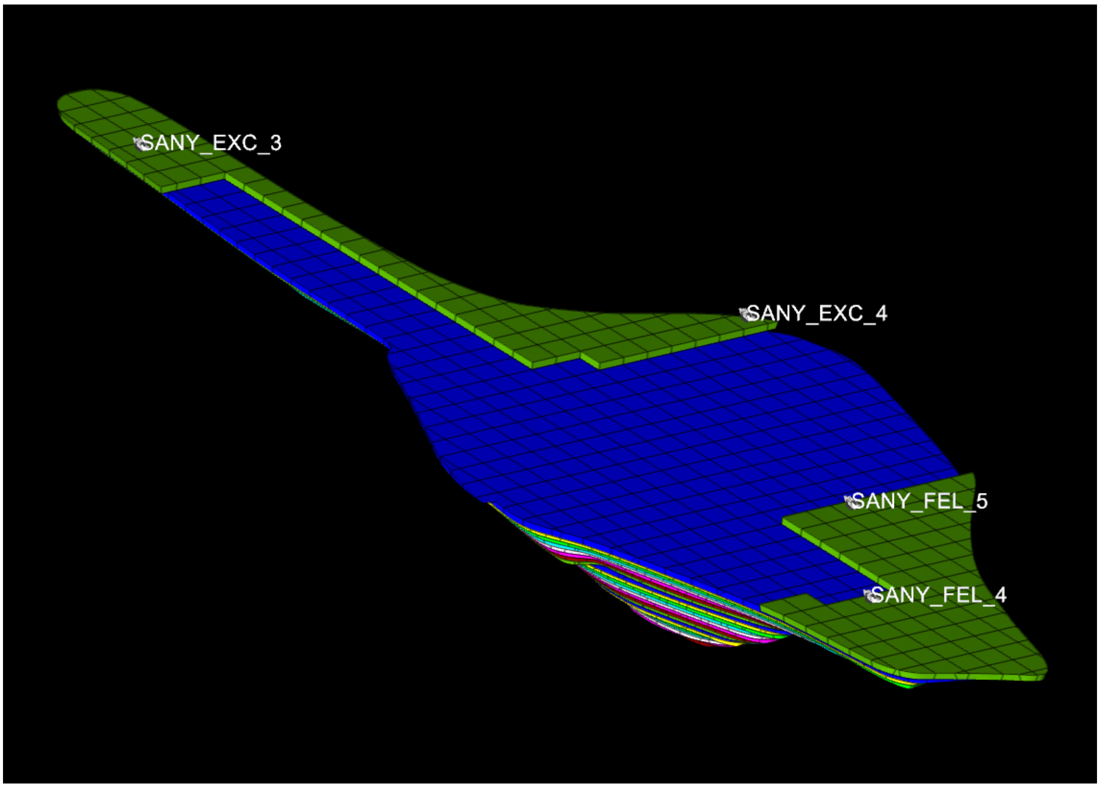


Figure 71: Period ending – January 2031

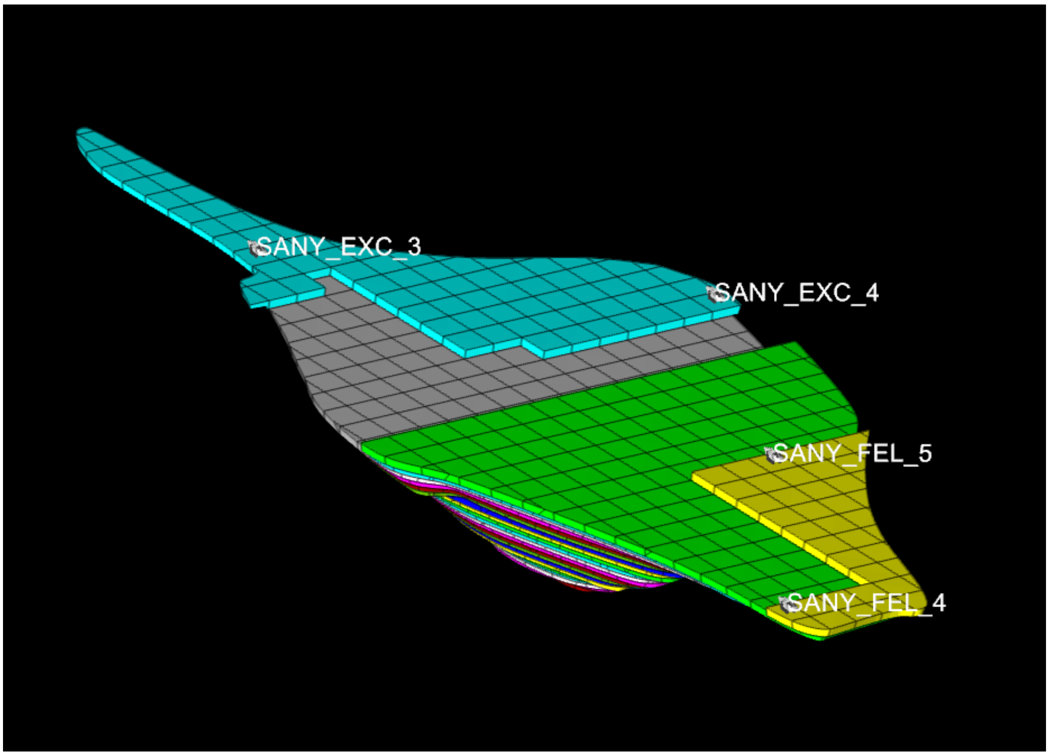


Figure 72: Period ending – January 2036

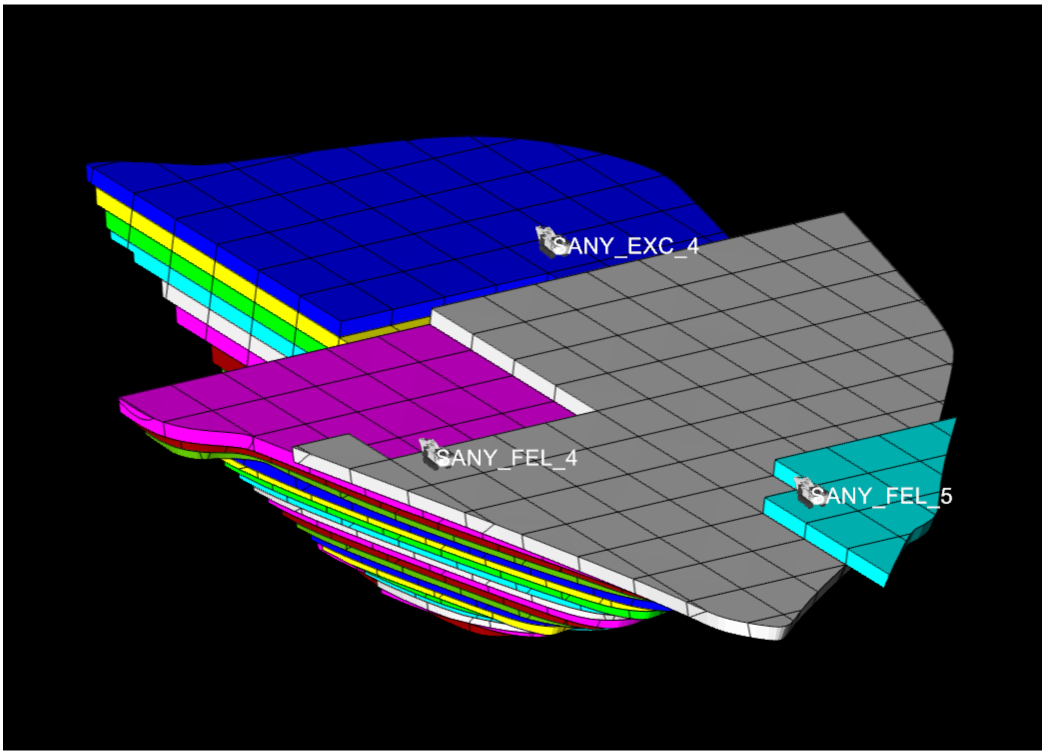


Figure 73: Period ending – Jan 2041

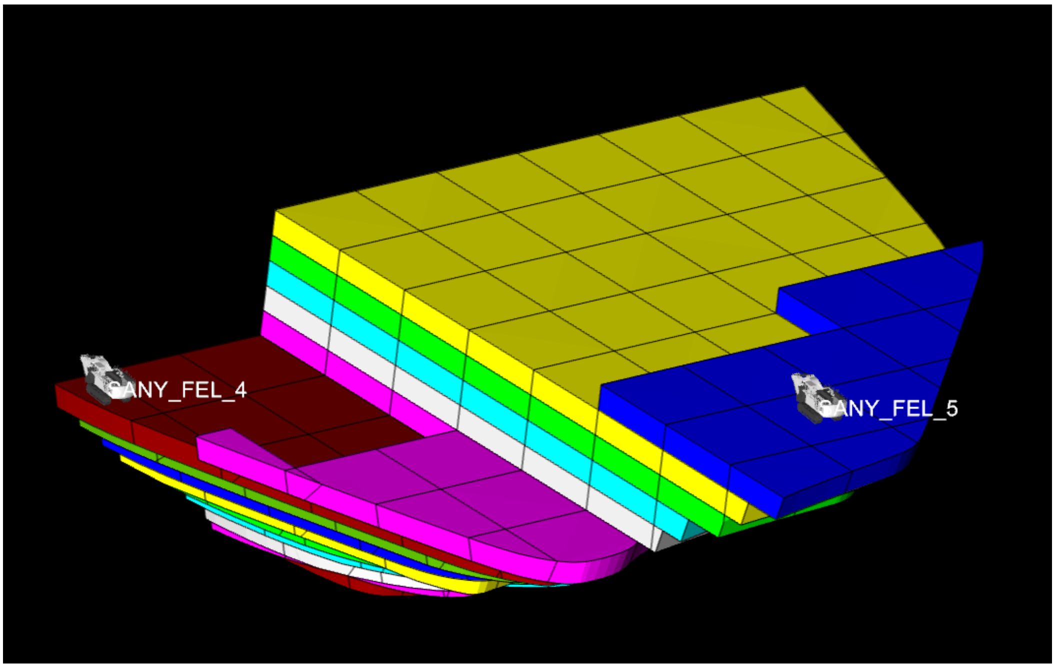


Figure 74: Period ending – Jan 2046

16.15 Rehabilitation and Closure

Rehabilitation for the closure of Molo Graphite open pit mine is a critical process aimed at restoring the site to a safe and environmentally sustainable condition. The key steps and considerations involved in open pit mine rehabilitation and closure are:

- Planning and Assessment
- Safety Measures
- Removing Equipment and Infrastructure
- Stabilization and Containment
- Soil and Vegetation Restoration
- Water Management
- Rehabilitation Monitoring
- Regulatory Compliance
- Stakeholder Engagement
- Financial Assurance
- Post-Closure Management
- Public Reporting

More detail on the integrated closure plan can be found in section 20.8.

17 RECOVERY METHODS

This section describes the process design basis adopted to define the concentrator process design criteria, develop the mass, and water balance, and identify and size the major equipment required to process the Molo ore in accordance with the mining design basis set out in Section 16 above.

The ore processing circuit consists of two-stage crushing followed by primary milling and classification, a flotation separation and concentrate upgrading circuit, and final graphite product and tailings effluent handling facilities.

The process is designed, based on metallurgical test work conducted and described in Section 13, for an expected overall graphite recovery of 88.3% to final concentrate, at the required grade, from an average plant feed head grade of 6.23%. The Molo Phase 1 and expansion processing plants will produce an estimate 150 ktpa of final concentrate over the LoM. The head grade is based on the hypothetical process feed at the ROM pad, for the LoM mine plans. A further description is included in the design basis section, below.

17.1 Process Design Basis

The data contained in Table 64 below form the basis for the design of the Molo Expansion Project and allow for the generation of the process flow diagrams, mass balance and specifications for process equipment.

Table 64: Design Criteria

Design Criteria			
Production Schedule	**Units**	**Design**	**Source**
General Concentrator			
Operating Schedule	Days/Year	365	C
Operating Schedule	Hours/Day	24	C
Operating Schedule	Shifts/Day	3	C
Operating Schedule	Hours/Shift	8	C
Total Available Hours Per Year	Hours/Year	8,760	D3
Plant Annual Utilised Hours	Hours	7,406	D3
Plant Utilisation	%	85%	C
Plant Availability	%	93%	D3
Plant Throughput	tpa	2,653,116	D3
Plant Throughput	tpm	221,093	D3
Plant Throughput	tph	358	D3
Concentrate Product Production			

Design Criteria			
Production Schedule	**Units**	**Design**	**Source**
Concentrate Product Production	tpa	150,000	C
Head Grade	%C	6.23%	C
Recovery	%C	88%	D1
Concentrate Grade	%C	97%	D1
ROM Ore Handling & Crushing			
Operating Hours	Hours/day	16	D2
Crusher annual run hours	Hours/year	5,840	D3
Crusher utilisation/availability	%	67%	D3
Crusher Plant Throughput	tph	537	D3
ROM Ore Density	g/t	2.62	D1
Moisture	%	3%	E
The following source codes are used to reference the origin of each item of information that appears in the design criteria.			
Code	**Description**		
A	Assumption based on most current, available information		
B	Previously reported		
C	Client Input		
D1	Based on Testwork results		
D2	Based on design experience		
D3	Calculated from other parameters		
D4	Based on OEM recommendations		
E	No information available		

The metallurgical data on the size fraction analysis and overall graphite recovery in Table 65 below was considered the most representative for Molo expansion flowsheet design from all test work. However, the current graphite market demand requires product classification into four groups, namely: Extra-Large or Jumbo flake, large flake, medium flake and small, or Fine flake products. Therefore, in Table 65 the reconstructed product size distribution grouping and product grade from the metallurgical data received and represents the final product output design parameters.

Table 65: Reconstructed Product Size Distribution Grouping and Product Grade from the Metallurgical Data Received

Product Mesh	Microns	Mass (%)	Grade (% Total Carbon)
+48	+297	24.3	96.9
-48/+65	-297/+210	18.0	97.1
-65/+80	-210/+177	9.8	97.0
-80/+100	-177/+149	7.0	97.2
-100/+150	-149/+106	13.0	97.3
-150/+200	-106+74	9.0	98.1
-200	74	19.0	97.5
Total		**100.0**	**97.2**

Table 66: Metallurgical Data – Flake Size Distribution and Product Grade

Product Size	% Distribution	Product Grade (%) Carbon
>50 mesh	24.3	96.9
-50 to +80 mesh	27.8	97.1
-80 to +100 mesh	7.0	97.2
-100 mesh	41.0	97.6

Table 67: Product Size Distribution per Ore Type (% of yielded product)

Mesh Size	Grade (%Cg)	Product Size Distribution per Ore Type (% of Yielded Product)	
		Low Grade Ore	High Grade Ore
+48	96.9	26.2	19.3
-48 +80	97.0	27.7	27.1
-80 +100	97.2	7.1	8.5
-100	97.6	39.0	45.1
TOTAL	**97.25***	**100.0**	**100.0**

* Weighted average for the Project

17.2 Process Flow sheets

Process Flow diagrams have been developed on the basis of the Process Design Criteria developed from the metallurgical testwork. The process Flow diagrams show all the major equipment required for each plant area and the process routs to be used. The process is described in section 17.3 based on PFD register below. (Table 68).

Table 68: Process Flow Diagrams Register

#	Title/Area	Drawing Number	Revision
1	Ore receiving, Primary and Secondary Crushing and Screening	P9239-PFD-1100-1	B
2	Ore receiving, Primary and Secondary Crushing and Screening	P9239-PFD-1100-1	B
3	Primary Milling and Flash Flotation	P9239-PFD-1200-1	B
4	Primary Milling and Flash Flotation	P9239-PFD-1200-2	B
5	Rougher Flotation	P9239-PFD-1300	B
6	Primary Concentrate Cleaning	P9239-PFD-1310	B
7	Fine Flake Cleaning	P9239-PFD-1340	B
8	Attrition Cleaning	P9239-PFD-1360	B
9	Secondary Attrition Cleaning	P9239-PFD-1380	B
10	Final Tailings Handling Filtration and Disposal	P9239-PFD-1400	B
11	Final Concentrate Handling	P9239-PFD-1500	B
12	Concentrate Handling and Drying	P9239-PFD-1570	B
13	Water Services	P9239-PFD-1600	B
14	Air Services	P9239-PFD-1640	B
15	Collector Storage and Dosing	P9239-PFD-1800	B
16	Frother Storage and Dosing	P9239-PFD-1860	B
17	Flocculant Storage and Dosing	P9239-PFD-1880	B
18	Coagulant Storage and Dosing	P9239-PFD-1890	B

17.3 Process Description

The Molo graphite processing concentrator plant consist of conventional crushing, milling and flotation circuits with, dewatering, bagging, and tailings processing circuits. The Molo graphite concentrator plant shall consist of the following areas:

- Ore receiving and primary crusher circuit.
- Screening and secondary crusher circuits.
- Primary milling and flash flotation.
- Rougher flotation.
- Primary concentrate cleaning.
- Fine flake cleaning.
- Attrition cleaning circuit.
- Secondary attrition cleaning circuit.
- Final tails handling and disposal.
- Final tails filtration and disposal.
- Final concentrate handling circuit.
- Concentrate drying and bagging.
- Reagents mixing and dosing sections.
- Air and water services.

The ROM material gets crushed in two crushing stages namely primary and secondary crushing. The material then goes through primary milling, flash, and rougher flotation.

The material then gets subjected to primary concentrate cleaning, fine flake cleaning and attrition cleaning, (alternatively secondary attrition cleaning).

The concentrate material gets thickened to produce a concentrate filter cake to less than 20% moisture, which will be dried to reduce the moisture content to less than 0.5%. The concentrate gets screened into four different size fractions and bagged separately as final product. The tailings material gets thickened and filtered to produce a tailings filter cake, which is then transported via a conveyor system to the tailings facility.

17.4 Ore Receiving and Crushing

The ROM is fed into the ore receiving and primary crushing circuit via tippers, tipping into the ROM bin. The ROM material is fed onto the ROM static grizzly screen 1100-SR-004. The ROM static grizzly is fitted with a dust suppression system. The ROM static grizzly is used as a scalping screen to control the size of material to the primary cone crusher 1100-CR-100. The material gets screened using a static grizzly screen situated at the top of the primary crusher bin 1100-BN-006 to reject rocks larger than 300 mm as the oversize material.

The oversize material gets stockpiled near the feed bin and may be manually crushed using rock breaker 1100-RB-001.

The static grizzly undersize material less than 300 mm discharges into the primary crusher bin 1100-BN-006. The primary crusher bin discharges the material on the grizzly feeder 1100-FD-008. The material gets further screened using the grizzly feeder. The grizzly feeder is fitted with a dust suppression system.

The undersize material from the grizzly feeder is discharged on the primary cone crusher discharge conveyor belt 1100-CV-011. The oversize material from the grizzly feeder is discharged into the primary cone crusher 1100-CR-010. The crusher product and the grizzly undersize are discharged onto the primary crusher discharge conveyor 1100-CV-011. The primary crusher discharge conveyor belt is fitted with a dust suppression system.

The primary crusher product conveyor belt discharges on to the classification screen feed conveyor belt 1100-CV-012 and 1100-CV-013. The classification screen feed conveyors are fitted with a dust suppression system, tramp metal magnets 1100-EE-046A/B and weightometer 1100-ZM-069A/B. The classification screen feed conveyors feeds into two off double deck classification screens 1100-SR-040A/B, running in parallel. The classification screens are fitted with 2 decks; the top deck is fitted with 35 mm screen panels and the bottom deck is fitted with 16 mm panels. The oversize from the top and bottom decks discharges onto the secondary crusher feed bins 1100-BN-022 and 1100-BN-024. The material from the secondary crusher feed bins is drawn via the secondary crusher feeders 1100-FD-024 and 1100-FD-026 on to the secondary crusher feed conveyor belt 1100-CV-034. The secondary crusher feed conveyor belt discharges into the secondary crusher 1100-CR-026. The secondary crusher discharges its product onto the primary crusher discharge conveyor belt 1100-CV-011 as a recirculating load.

The classification screens undersize, less than 16 mm size fraction, gets discharged onto the primary mill feed conveyor 1100-CV-044. The primary mill feed conveyor belt is fitted with a dust suppression system. The primary mill feed conveyor transfers material onto the primary mill feed bin conveyor 1100-CV-046, which discharges the material into the mill feed bin 1200-BN-058.

17.5 Primary Milling and Flash Flotation

The primary mill feed bin is equipped with three vibrating feeders 1200-FD-060/062/064. The primary mill feed bin feeders discharge the material onto the primary mill feed bin discharge conveyor 1200-CV-064. The primary mill feed bin discharge conveyor transfers the material onto primary mill feed conveyors 1200-CV-066 and 1200-CV-068 fitted with hammer samplers 1200-SA-098/100 and weightometers'1200-ZM-068/070.

The primary mill feed conveyors feeds onto the primary mill scalping screens 1200-SR-070/076 running in parallel. The scalping screens consist of 2 screening decks; top deck consists of 6 mm aperture panels whilst the bottom deck is fitted with 2 mm slots. Any spillage material in the primary milling circuit is pumped to the scalping screens through the mill area spillage pumps 1200-PP-094/096. The material greater than 2 mm, (top and bottom deck oversize) feeds into 2 primary ball mills 1200-ML-078/080 running in parallel.

The primary mills are expected to produce a product with a grind of 80% passing 500 microns (µm). The primary mills product is discharged into the primary ball mill discharge sumps 1200-TK-082/084, respectively. The primary mill scats report into the primary mill scats bins 1200-BN-080/084.

The material less than 2 mm from the scalping screen flows by gravity into the flash flotation cells 1200-FC-076/078. The flash flotation cells are installed to take advantage of graphite liberation that takes place at coarse particle sizes.

Flash flotation produces a concentrate that reports to the total concentrate sump 1300-TK-134. The flash floatation concentrate discharge into the flash flotation concentrate sumps 1200-TK-009/010 and gets pumped to the total concentrate sump via flash flotation sump pumps 1200-PP-009A/B and 1200-PP-010A/B. The flash flotation tailings discharge into the flash floatation tailings sumps 1200-TK-088/090 and gets pumped to the primary mill discharge sumps via floatation tailings sump pumps 1200-PP-090/092 and 1200-PP-094/096.

The slurry from the primary mill discharge sumps gets pumped via the primary mill discharge pump 1200-PP-084/086 and 1200-PP-088/090 to the primary mill classification screens 1200-SR-072/080, which are fitted with a 600 µm screen panels. The oversize material of the classification screens greater 600 µm discharges into the primary mills. The undersize material of the classification screens less 600 µm reports to the rougher flotation circuit.

17.5.1 Rougher Flotation

The rougher flotation surge tank 1300-TK-728 is fitted with an agitator 1300-AG-728 and is fed by the primary mill linear screens undersize material. The slurry consists of the flash flotation tailings and the mill product. The rougher flotation banks 1300-FC-110/112/114/116 and 1300-FC-118/120/122 are fed from the rougher flotation surge tank. The frother and collector reagents are added into rougher cells. The rougher concentrate discharges into the total concentrate sump 1300-TK-134-1/2. The concentrate from flash flotation, primary scavenger concentrate, and the primary rougher concentrate are combined in the total concentrate sump and pumped to the primary concentrate cleaning circuit using the pumps 1300-PP-136/137.

The tailings from the rougher flotation circuit are discharged into the combined tailings sump 1300-TK-138-1/2. The combined tailings sump feeds from the attrition cleaner tailings, fine flake cleaner, the primary cleaner scavenger tailings, and rougher tailings.

The material from combined tailings sump is then pumped to the final tailings' thickener 1400-TH-310 via the combined tailings pump 1300-PP-140/142. The spillage from the rougher flotation circuit is pumped to the primary ball mill scalping screen using the rougher flotation spillage pump 1300-PP-144.

17.6 Primary Concentrate Cleaning

The primary cleaner concentrate dewatering screens 1300-SR-146A/B/C/D/E/F are fed from the combined flotation concentrate. The primary cleaner concentrate dewatering screens are fitted with 74 µm panels. The oversize material from the primary cleaner concentrate dewatering screens is discharged into the primary cleaner concentrate oversize sumps 1300-TK-146A/B.

The material from the primary cleaner concentrate oversize sumps feeds into the primary polishing mill 1300-ML-148 via the primary cleaner concentrate oversize sump pumps 1300-PP-148/150. The primary polishing mill product discharges into the primary polishing mill discharge sumps 1300-TK-150A/B. The dewatering screen undersize discharges into the primary polishing mill discharge sumps.

The combined material from the primary polishing mill discharge sumps gets pumped to the primary cleaner column cells 1300-FC-154-1/2/3 via the primary polishing mill discharge sump pumps 1300-PP-152/154.

The primary cleaner column cell concentrate gets discharged onto the primary cleaner column flotation cell concentrate classification screen 1300-SR-190-1/2/3 fitted with 212 µm screen panels via distribution boxes 1300-DB-191/192/193. The oversize material from the primary cleaner column concentrates classification screen discharges into the primary cleaner column concentrate classification screen sumps 1300-TK-278A/B/C and gets pumped to the secondary attrition circuit via the discharge pumps 1300-PP-280/282. The undersize material of the primary cleaner column concentrates screen discharges into the concentrate classification screen undersize sumps 1300-TK-192A/B/C. The material from the classification screen undersize sump gets pumped to the fine flake cleaning circuit using the primary column cell concentrate undersize pumps 1300-PP-196/198.

The primary cleaner column cell tailings get discharged into the primary cleaner column cell tailings sump 1300-TK-160. The material then gets pumped to the primary cleaner flotation bank 1300-FC-164/166/168 via 1300-PP-162/164. The concentrate from the primary cleaner flotation bank gets discharged into the primary cleaner flotation concentrate sump 1300-TK-168 and recycled back to the primary polishing mill discharge sump via the primary cleaner flotation concentrate sump pumps 1300-PP-166A/B. The tailings from the primary cleaner flotation bank gets discharged into the primary scavenger cleaner flotation bank 1300-FC-170/172/174. The concentrate from the primary scavenger cleaner flotation bank gets discharged into the primary scavenger cleaner flotation concentrate sump 1300-TK-174 and gets pumped to the combined concentrate sump via the primary scavenger cleaner concentrate sump pumps 1300-PP-182/184. The tailings from the primary scavenger cleaner flotation bank gets discharged into primary scavenger cleaner flotation tailings sump 1300-TK-175 and gets pumped to the combined tailings sump via the primary scavenger cleaner flotation tailings sump pumps 1300-PP-168A/B.

The final cleaner tailings sump 1300-TK-184 gets fed from the attrition cleaner tailings and the fine flake cleaner tailings. The combined tailings from final cleaner tailings sump gets pumped to the combined tailings sump via the final cleaner tailings' sump pumps 1300-PP-186/188.

17.7 Fine Flake Cleaning

The fine flake dewatering screen 1300-SR-267 is fed from the primary column concentrate screen undersize and the fine flake cleaner scavenger concentrate. The fine flake dewatering screen is fitted with 75 µm panels. The oversize material from the fine flake dewatering screen is discharged into the fine flake polishing mill feed sump 1300-TK-268.

The material then gets pumped to the fine flake polishing mill 1300-ML-268 via the feed sump pumps 1300-PP-268A/B. The fine flake polishing mill product discharges into the fine flake polishing mill discharge sump 1300-TK-270. The dewatering screen undersize discharges into the fine flake polishing mill discharge sump.

The combined material from the fine flake polishing mill discharge sump gets pumped to the fine flake cleaner column cells 1300-FC-274A/B via the fine flake polishing mill discharge sump pumps 1300-PP-272/274. The fine flake cleaner column cell concentrate gets discharged into the fine flake cleaner column cell concentrate sump 1300-TK-200. The fine flake cleaner column cell concentrate then gets discharged into the attrition thickener 1300-TH-001 via the fine flake cleaner column cell concentrate sump pumps 1300-PP-200A/B.

The fine flake cleaner column cell tailings get discharged into the fine flake cleaner column cell tailings sump 1300-TK-282. The material then gets pumped to the fine flake cleaner flotation bank 1300-FC-286/288/290 via 1300-PP-284/286. The concentrate from the fine flake cleaner flotation bank gets recycled back to the fine flake polishing mill discharge sump. The tailings from the fine flake cleaner flotation bank gets discharged into the fine flake scavenger cleaner flotation bank 1300-FC-292/294. The concentrate from the fine flake scavenger cleaner flotation bank gets discharged into the fine flake scavenger cleaner discharge sump 1300-TK-292. The material then gets pumped back to the fine flake polishing mill dewatering screen via the fine flake scavenger cleaner discharge sump pumps 1300-PP-300/302.

The fine flake scavenger cleaner flotation bank tailings get discharged into the tailing sump 1300-TK-294 and gets pumped to the final cleaner tailing sump via the fine flake scavenger cleaner flotation tailings sump pumps 1300-PP-294A/B.

17.8 Attrition Cleaning

The attrition concentrate thickener 1300-TH-001 gets fed from the combined cleaner concentrate, secondary classification screen undersize, attrition cleaner scavenger concentrate, and the fine flake cleaner column concentrate via the attrition concentrate thickener basket 1300-SR-004. The overflow of the attrition concentrate thickener get discharged into the process water tank. The underflow of the attrition concentrate thickener gets drawn via the attrition concentrate thickener pumps 1300-PP-002/003 to the attrition mill 1300-ML-005, or by-pass stream to the secondary attrition scrubber.

The product from the attrition mill flows into the attrition mill discharge sump 1300-TK-010. The material from the attrition mill discharge sump gets pumped to the attrition cleaner column cells 1300-FC-020-1/2 via the attrition mill discharge pump 1300-PP-015/017. The concentrate from the attrition cleaner column cells gets discharged into the attrition cleaner column cell concentrate discharge sump 1300-TK-065.

The material from the attrition cleaner column cell concentrate discharge sump gets pumped to the final concentrate thickener circuit via the attrition cleaner column cell concentrate discharge sump pumps 1300-PP-070/072.

The tailings from the attrition column cells are discharged into the attrition column cell tailings sump 1300-TK-030 equipped with an agitator 1300-AG-030. The material from attrition column cell tailings sump gets pumped to the attrition cleaner flotation bank 1300-FC-040/045 via the attrition column cell tailings sump pumps 1300-PP-035/037. The concentrate from the attrition cleaner flotation bank gets recirculated back to the attrition mill discharge sump via gravity.

The tailings from the attrition cleaner flotation bank gets discharged to the attrition scavenger flotation cell 1300-FC-050. The concentrate of the attrition scavenger cleaner flotation cell gets discharged into the attrition scavenger cleaner flotation tailings sump 1300-TK-050 and gets pumped to the attrition concentrate thickener via the attrition scavenger cleaner flotation cell pumps 1300-PP-060/062. The tailings from the attrition scavenger cleaner cell are recycled by gravity back to the final cleaner tailings' sump in the primary concentrate cleaning circuit.

The spillage from the attrition cleaning circuit is pumped to the attrition cleaning thickener via the attrition cleaning spillage pump 1400-PP-204.

17.9 Secondary Attrition Cleaning

The secondary attrition cleaning circuit is an alternative, or bypass circuit to the attrition cleaning circuit.

The secondary attrition classification screens 1300-SR-700-1/2 gets fed from the combined cleaner concentrate and secondary attrition cleaner scavenger concentrate. The undersize material of the secondary attrition screens gets discharged into undersize sumps 1300-TK-702-1/2 and gets pumped to the attrition concentrate thickener via the secondary attrition classification screen undersize pump 1300-PP-728/730. The oversize material from the secondary attrition classification screen is discharged into the oversize sumps 1300-TK-701-1/2. The material from the secondary attrition classification screen sump gets pumped to the secondary attrition mill 1300-ML-702, or it can be bypassed to the attrition mill via the secondary attrition classification screen sump pumps 1300-PP-703/705.

The product from the secondary attrition mill and the secondary attrition cleaner concentrate flows into the secondary attrition mill discharge sump 1300-TK-704. The material from the secondary attrition mill discharge sump gets pumped to the secondary attrition cleaner column cell 1300-FC-708 via the attrition mill discharge sump pump 1300-PP-706/708. The concentrate from the secondary attrition cleaner column cell gets discharged into the secondary attrition cleaner column cell concentrate discharge sump 1300-TK-724.

The material from the attrition cleaner column cell concentrate discharge sump gets pumped to the final concentrate thickener circuit via the secondary attrition cleaner column cell concentrate discharge sump pump 1300-PP-726/732.

The tailings from the secondary attrition column cell are discharged into the secondary attrition column cell tailings sump 1300-TK-710. The material from the secondary attrition column cell tailings sump gets pumped to the secondary attrition cleaner flotation bank 1300-FC-714/716 via the secondary attrition column cell tailings sump pump 1300-PP-712/714.

The concentrate from the secondary attrition cleaner flotation bank gets discharged into the secondary attrition cleaner flotation concentrate sump and gets pumped back to the secondary attrition mill discharge sump via the secondary attrition cleaner flotation concentrate sump pumps 1300-PP-716A/B.

The tailings from the secondary attrition cleaner flotation bank gets discharged to the secondary attrition scavenger flotation cell 1300-FC-718. The concentrate of the secondary attrition scavenger cleaner flotation cell gets discharged into the secondary attrition scavenger cleaner flotation concentrate sump 1300-TK-718 and is pumped to the secondary attrition concentrate thickener via the secondary attrition scavenger cleaner flotation cell pumps 1300-PP-722A/B. The tailings from the secondary attrition scavenger cleaner cell are recycled by gravity back to the final cleaner tailing sump in the primary concentrate cleaning circuit.

17.10 Final Tailings Handling

17.10.1 Tailings Handling and Disposal

The final tailings thickener 1400-TH-310 gets fed from the combined tailings sump via the final tailing thickener basket 1400-SR-311. The overflow of the final tailing thickener reports to the process water tank. The underflow of the final tailing thickener gets pumped to the tailings filter feed tank via the final tailing thickener underflow pumps 1400-PP-312/314. The spillage from the final tailings handling and disposal circuit is pumped to the final tailing thickener via the final tailing spillage pump 1400-PP-338.

17.10.2 Tailings Filtration and Disposal

The final tailings filtration and disposal circuit is operated in a semi-batch manner, and it is anticipated to produce a tailings filter cake with 15% moisture. The final tailings filtration feed tank 1400-TK-040 is fed from the final tailing thickener underflow. The final tailings filtration feed tank is equipped with an agitator 1400-AG-040.

The final tailings thickener underflow material from final filtration feed tank is pumped to the final tailings filter 1400-FL-052 via the final filtration feed tank pumps 1400-PP-048/050. The filtrate from the final tailings filter is discharged into the final tailings filtration spillage sump and recycled to the final tailing thickener via the final tail filtration spillage pump 1400-PP-060. The filter cake is discharged onto the final tailings filter cake conveyor belt 1400-CV-068. The final tailings filter cake conveyor transfers the material to the final tailings stacking conveyor which discharges and distributes the material at the final tailings deposition area. The spillage from the final tailings filtration and disposal circuit is pumped to the final tailing thickener feed tank via the final tailings filtrate spillage pump.

17.11 Final Concentrate Handling

17.11.1 Concentrate Handling

The final concentrate thickener 1500-TH-342 is fed from the combined concentrate sump, final concentrate belt filter filtrate and secondary attrition column concentrate sump via the final concentrate thickener basket 1500-SR-343.

The final concentrate thickener overflow is discharged into the process water tank. The final concentrate thickener underflow is pumped to the final concentrate belt filter feed tank 1500-TK-350 via the final concentrate thickener underflow pumps 1500-PP-344/346. The final concentrate belt filter feed tank is equipped with an agitator 500-AG-348.

The final concentrate thickener underflow is pumped to the final concentrate belt filter 1500-FL-358 via the final concentrate belt filter feed tank pump 1500-PP-352. The final concentrate belt filter is equipped with a final concentrate filter vacuum pump 1500-PP-360. The final thickener underflow is filtered in the final concentrate belt filter that is expected to produce a filter cake with less than 20% moisture. The filtrate from the final concentrate belt filter is pumped back to the final concentrate thickener via the final concentrate belt filter filtrate pump 1500-PP-370. The filter cake from the final concentrate belt filter is discharged onto the final concentrate filter cake conveyor 1500-CV-374 and fed to the final concentrate drying and bagging circuit.

The spillage from the final concentrate handling circuit is pumped to the final concentrate thickener via the final concentrate filtrate spillage pump 1400-PP-370.

17.11.2 Concentrate Drying and Bagging

The concentrate from the final concentrate filter cake conveyor is fed into a diesel fired dryer 1500-DR-376. The diesel fired dryer produces the fine dry graphite concentrate powder and the coarse dry graphite concentrate flakes. The dry concentrate feeds into the concentrate surge bin 1500-BN-386-1/2/3/4/5/6/7/8 using a pneumatic system. The dry concentrate from the concentrate surge bin is fed onto the sifter screens 1500-SR-396-1/2/3/4/5/6/7/8 via gravity. The dried concentrate is screened into four size fractions namely, +400 µm, -400 +177 µm, -177 µm, +149 µm and -149 µm. Each of the size fractions are bagged separately as final product. The sifter screens are equipped with top deck screen panels of 400 µm and bottom deck screen panels of 177 µm. The oversize material of the top deck reports to the 400 µm bagging plant and the oversize material of the bottom deck repots to the 177 µm bagging plant. The undersize of the double deck sifter screens feeds onto the single deck sifter screen 1500-SR-404 equipped with 149 µm screen panels. The oversize material of the single deck sifter screens reports to the 149 µm bagging plant. The undersize material of the single deck sifter screens reports to the -149 µm bagging plant.

17.12 Water Services

The process water tank 1600-TK-472 is fed from the final concentrate thickener overflow, attrition concentrate thickener overflow, and the tailings thickener overflow.

Make-up water is pumped from a well-field into the well field collection tank 1600-TK-450. The water from the well field collection tank feeds into the process water tank, plant water tank and to the potable water plant 1600-ZM-454 via the well field collection tank pumps 1600-PP-466/464/468, respectively. Additionally, the storm / pollution control dam 600-TK-478 feeds into process water tank via storm / pollution control dam pumps 1600-PP-479/481.

The water is distributed to the following plant areas from process water tank using process water pumps 1600-PP-474/476:

- Attrition cleaning.
- Fine flake cleaning.
- Primary milling and flotation.
- Rougher floatation.
- Primary concentrate cleaning.

- Secondary attrition cleaning.
- Final concentrate thickener.
- Tailings thickener.

Treated water from the potable water plant is stored into the potable water storage tank 1600-TK-456. The water from the potable water storage tank is drawn via the potable water storage tank pumps 1600-PP-458/470 and 1600-PP-460/472 to the potable water camp and to the potable water plant tanks, respectively. The plant potable water tank is fed via the potable water storage tank pump 1600-PP-460. The potable water plant is fed via the plant potable water tank pump 1600-PP-465/467.

The well-field water tank is fed via the plant potable water tank pump 1600-PP-466/478. The well-field water tank distributes water for dust suppression, gland service water and fire water purposes via the well-field water tank pumps to the following areas:

- Well-field water tank pump 1600-PP-432/434:
  - Flocculant plant.
  - Ore receiving dust suppression.
  - Secondary / tertiary crusher dust suppression.
  - Coagulant plant.
- Well-field water tank pump 1600-PP-436/438:
  - Primary milling gland service water.
  - Rougher flotation gland service water.
  - Primary cleaner gland service water.
  - Tailings thickener underflow pumps gland service water.
- Well-field water tank pump 1600-PP-448/450 and 1600-PP-552/554:
  - Fire water.

17.13 Air Services

The flotation blower 1600-HB-484 distributes air to the following circuits:

- Flash flotation.
- Rougher flotation.
- Primary cleaning.
- Fine flake cleaning.
- Attrition cleaning.
- Secondary attrition cleaning.

The instrument air compressor 1600-HA-488 delivers air to the instrument dryer 1600-DR-492 and the air receiver 1600-TK-498. The instrument dryer feeds into the instrument air receiver 1600-TK-496 which provides instrument air. The air receiver distributes air to the following equipment:

- Primary column cleaner.
- Fine flake column.
- Attrition cleaning column.
- Secondary cleaning column.

The tailings filter air compressor 1600-HA-491 feeds into the tailings filter air receiver 1600-TK-499 which provides tailings filtration air.

17.14 Collector Storage and Dosing

The collector storage isotainer 1800-TK-500 distributes the frother to the following equipment via the following pumps:

- Flash flotation collector via the collector pump 1800-PP-502.
- Rougher flotation cell #1 collector via the collector pump 1800-PP-504.
- Rougher flotation cell #3 collector via the collector pump 1800-PP-506.
- Primary cleaner column cell collector via the collector pump 1800-PP-508.
- Primary cleaner collector via the collector pump 1800-PP-510.
- Primary cleaner scavenger collector via the collector pump 1800-PP-512.
- Fine flake column cell collector via the collector pump 1800-PP-514.
- Fine flake cleaner collector via the collector pump 1800-PP-516.
- Fine flake cleaner scavenger collector via the collector pump 1800-PP-518.
- Attrition column cell collector via the collector pump 1800-PP-519.
- Attrition cleaner collector via the collector pump 1800-PP-520.
- Secondary attrition column cell frother via the collector pump 1800-PP-521.
- Secondary cleaner collector via the collector pump 1800-PP-522.

17.15 Froth Storage and Dosing

The frother storage isotainer 1800-TK-524 distributes the frother to the following equipment via the following pumps:

- Flash flotation frother via the frother pump 1800-PP-528.
- Rougher flotation cell #1 frother via the frother pump 1800-PP-530.
- Rougher flotation cell #3 frother via the frother pump 1800-PP-532.
- Primary cleaner column cell frother via the frother pump 1800-PP-534.
- Primary cleaner frother via the frother pump 1800-PP-536.
- Primary cleaner scavenger frother via the frother pump 1800-PP-538.
- Fine flake column cell frother via the frother pump 1800-PP-540.
- Fine flake cleaner frother via the frother pump 1800-PP-542.
- Fine flake cleaner scavenger frother via the frother pump 1800-PP-544.
- Attrition column cell frother via the frother pump 1800-PP-546.
- Attrition cleaner frother via the frother pump 1800-PP-548.
- Secondary attrition column cell frother via the frother pump 1800-PP-550.

- Secondary cleaner frother via the frother pump 1800-PP-552.

17.16 Concentrate Flocculant Mixing and Dosing

The dry flocculant gets deposited into the concentrate flocculant hopper 1800-TK-574. The dry flocculant gets drawn via the concentrate flocculant hopper screw feeder 1800-FD-576. The flocculant from the feeder gets discharged into the concentrate wetting mixing head 1800-ZM-578, which discharges into the concentrate flocculant mixing tank 1800-TK-580. The concentrate flocculant mixing tank is equipped with an agitator 1800-AG-582. The mixed flocculant is transferred to the concentrate flocculant dosing tank 1800-TK-584. The concentrate flocculant is distributed to the final concentrate thickener and attrition concentrate thickener via 1800-PP-586 and 1800-PP-587, respectively.

17.17 Tailings Flocculant Mixing and Dosing

The dry flocculant gets deposited into the tailings’ flocculant hopper 1800-TK-558. The dry flocculant gets drawn via the tailings’ flocculant hopper screw feeder 1800-FD-560. The flocculant from the feeder gets discharged into the tailings wetting mixing head 1800-ZM-562, which discharges into the tailings’ flocculant mixing tank 1800-TK-564. The tailings flocculant mixing tank is equipped with an agitator 1800-AG-566. The mixed flocculant is transferred to the tailings’ flocculant dosing tank 1800-TK-568. The tailings flocculant gets pumped to the final tailings’ thickener via 1800-PP-570.

The flocculant spillage gets pumped to the final tailings’ thickener via the flocculant spillage pump 1800-PP-590. The flocculant mixing and dosing circuit is equipped with a flocculant / coagulant area safety shower.

17.18 Coagulant Mixing and Dosing

The dry coagulant gets deposited into the coagulant hopper 1800-TK-594. The dry coagulant gets drawn via the concentrate flocculant hopper screw feeder 1800-FD-596. The coagulant from the feeder gets discharged into the coagulant wetting mixing head 1800-ZM-598, which discharges into the coagulant mixing tank 1800-TK-600. The coagulant mixing tank is equipped with an agitator 1800-AG-602. The mixed coagulant is transferred to the coagulant dosing tank 1800-TK-604. The coagulant is distributed to the final tailings’ thickener, final concentrate thickener and attrition concentrate thickener via 1800-PP-606, 1800-PP-612 and 1800-PP-615, respectively.

18 PROJECT INFRASTRUCTURE

18.1 Tailings Storage Facility

18.1.1 Basis of Design

The basis information of the current design of the TSF continues from previous design considerations and approach that was investigated and reported by SRK Consulting during the Molo 2015 FS and further the Epoch design Molo 2019 FS, (but very limited information was available from the Epoch design to consider). An existing smaller operational TSF is currently developing as part of the initial phase and physical performance and characteristics of the tailings generated on site is also considered for this second TSF design.

Highlighting key information from the previous studies and pertinent design components includes:

- Site selection report 2014:
    - A total of 6 potential TSF sites was identified and evaluated through various criterium and ranked in a matrix system.
    - The site ranking matrix results indicates that TSF 3 and TSF 6 rank significantly higher than the other remaining TSF sites, while TSF 1 and TSF 2 rank lower.
    - TSF 3, which is located west of the proposed plant, was identified as the most viable option based on the economic and environmental aspects for the development of a tailings dam disposal facility based on the ranking criteria.
- The Molo 2015 FS design where:
    - For 22.5 Mt for 30 years at 750,000 tpa.
    - Three deposition options were reviewed, including slurry deposition thickened and dry. It was requested by the Client that the base case be thickened tails with a lined facility. The report in principle assumed that the design for slurry and thickened tailings was essentially the same.
    - The geotechnical study indicated shallow soils with weathered gneiss bedrock at an average depth of 0.3m. This remains problematic for founding conditions and material availability for TSF construction.
    - The geochemical characterisation of the tailings was carried out which indicated that the tailings have the potential to pollute in the long term under wet conditions, however, the contaminant plume can be effectively controlled by installing drainage measures to intercept the plume.
    - Measures for minimising erosion, dust generation and rainfall infiltration was not included in the report.

- The Molo 2019 FS Design were:
  - The report mentions that due to the substantially reduced tonnages for the project as previously envisaged (2015), tailings will now be viewed and designed for as dried and co-disposed with the waste rock generated from the open cast mining, (Section 17.2 of the NI 43-101 dated July 13, 2017). A contributing factor is also that water availability is a challenge, with the area being relatively arid.
  - With the decision to go dry tailings disposal and co-disposal with the waste rock, the need for a tailings disposal facility is not warranted as it forms part of the waste rock dump, with the majority of the waste mass being waste rock.
  - The footprint was moved adjacent to the pit.

Leading from the key information mentioned above, the scope was to continue with the current and review and possibly improve the layout, design, TSF footprint, co-disposal design and planned tonnages and operations with the available historical and current information. From the information the following has been noted to indicate the migration of the TSF footprint from the previous designs received and the information considered in the current design.

The original SRK TSF footprint of Molo 2015 FS was relatively in the same area to the current identified footprint and, therefore, most of the field investigation done by SRK formed the basis, influencing the current design in terms of founding conditions and the formation of the containment structures. The Epoch TSF footprint (Molo 2019 FS), was located adjacent to the pit, which was a significant distance away and, therefore, not considered in this design.

In summary the principal evolution of the TSF design is that the original TSF (2015) was designed in terms of a wet / thickened tailings deposition onto a TSF that was to be constructed by the upstream deposition method. The facility had a relative extensive internal drainage system to manage the internal phreatics of the TSF and the basin was lined with a geomembrane. The growth and development of the facility was, therefore, dependant on the hydraulic deposition and segregation of course and fine material to form the outer structural zone and the inner "soft" (contained zone).

This premise was significantly altered by the change in tonnages and decision of dry deposition (Section 17.2 of the NI 43-101 dated July 13, 2017). After this change the deposition changed to a more mechanical approach with the outer containment structure being constructed from a coarse waste rock, (sourced as part of operations), and dry fines contained within this coarse outer structure. The design information of this new TSF was not available, but the indications was that the facility was not lined, (on the premise of being a dry material), and internal drainage systems was limited.

After evaluation of the deposition space requirements, the indicated mining boundaries and information available, zone of influence and risk to people involved, the final decision was made to move the TSF to the current layout and proceed with a dry stack facility design as per the indicated material parameters. The deposition is to be done by means of an overland conveyor for the tailings' fines and truck haul for the coarser overburden waste rock.

The facility is designed in concept to have the capacity to cater for a LoM of 25 years and a tailings feed that consists of:

- 51.6 Mt Fine Tailings.
- 1.63 Mt Waste Rock.

The facility will, therefore, have a storage capacity of 58 mm^3.

The reporting section contains extractions from the mentioned SRK Molo 2015 FS report and where appropriate, additional designs and information is included.

18.1.2 Site Characteristics

18.1.2.1 Climate and Vegetation

From the information reviewed the general area considered for the tailings' facility is sparsely vegetated. This includes open grassland with sporadic small trees and shrubs. Trees and shrubs are mostly concentrated near drainage features and stream beds.

The area under consideration falls within the semi-desert, southern zone of Madagascar. The Mean Annual Precipitation (MAP) is approximately 858.9 mm with a standard deviation of 246.9 mm and the Mean Annual Evaporation (MAE) rate is approximately 1,650 mm (S-Pan), therefore, the mine falls in an area where there is a negative moisture index, i.e., the potential for evaporation exceeds the average rainfall. Approximately 75% of the rainfall occurs during December to March. The summer rainfall generally occurs with heavy downpours over short periods of time and can, therefore, lead to flash floods.

Temperatures vary from a daily average of 22°C during the autumn and winter months, (April to October), to an average of 30°C during spring and summer months (November to March).

18.1.2.2 Topography and Drainage

Generally, the area is not associated with major topographic variance. Elevations within the target area are expected to range between approximately 486 mamsl to 565 mamsl, (based on Google Earth assessment and topography maps issued with hydrological report). The highest elevated areas are in the north-eastern parts of the target zone with elevations dropping towards the west. Refer Figure 75 below.

In areas of low relief, shallow surface soils are present resulting in areas of surface bedrock.

The mining area falls close to the edge of a water shed region, generally leading north and south, with the operations located south of the water shed.

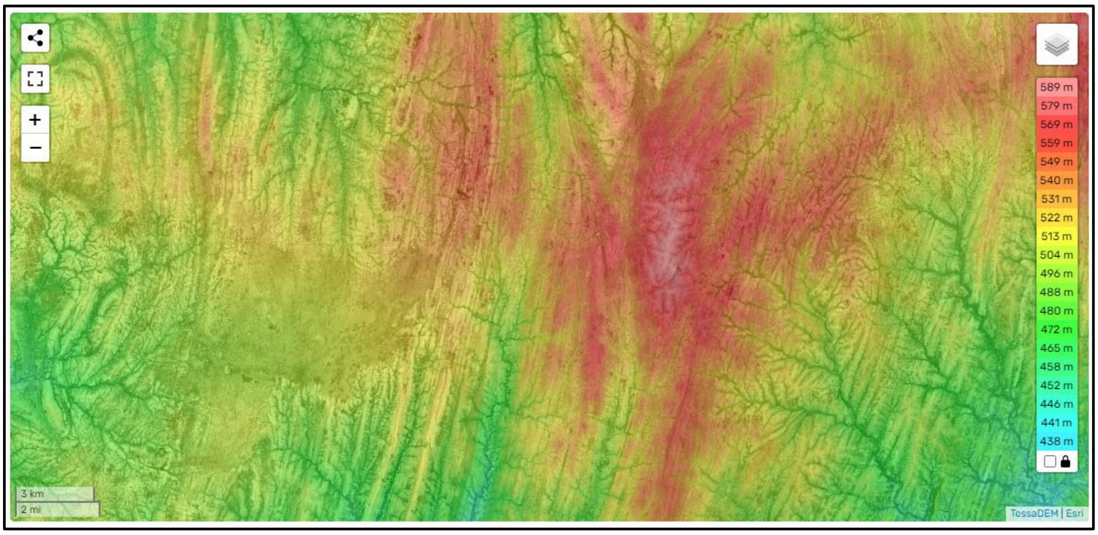


Figure 75: Topographic Heat Map

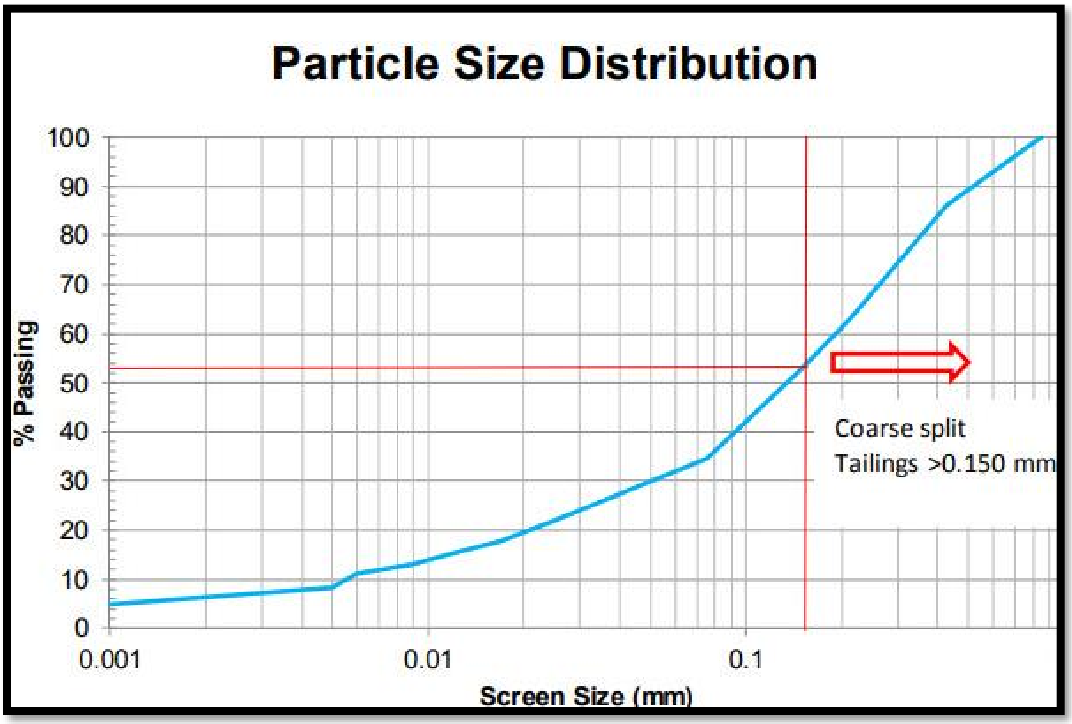
From satellite imagery various tributary drainage features can be observed traversing the target area. The majority of the main drainage features in the area follow north-south lineated features noticeable from satellite imagery. From the hydrological study reviewed the area is located on the watershed of three major river catchments, namely the Onilahy, the Ilinta and the Menarandra Rivers. Most of the target area falls within the Ilinta River catchment (catchment of about 7,538 km^2). All tributaries to the main rivers are classified as ephemeral (non-perennial). The Ilinta River flows in a south-westerly direction and terminates near the coastal town of Androka.

18.1.3 Tailings Characteristics

18.1.3.1 Grading Analysis

Tailings characteristics have been determined by SGS Mineral Services previously, on samples obtained from the site.

A grading analysis was conducted on the total tailings sample. The results illustrated in Graph 39 below indicates that approximately 34.7% of the material passes 75 microns. The material classifies as SM-ML, (Silty Sand with high silt fraction), in terms of the Unified Soil Classification System (USCS).



Graph 39: Grading Curve for Fine Tailings

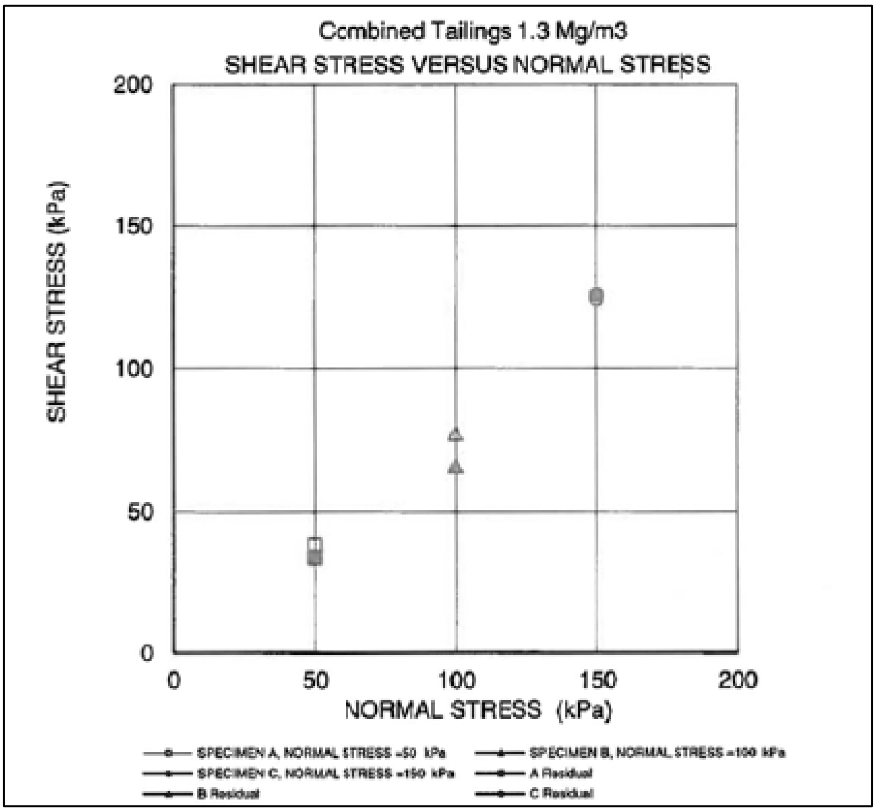
18.1.3.2 Material Strength

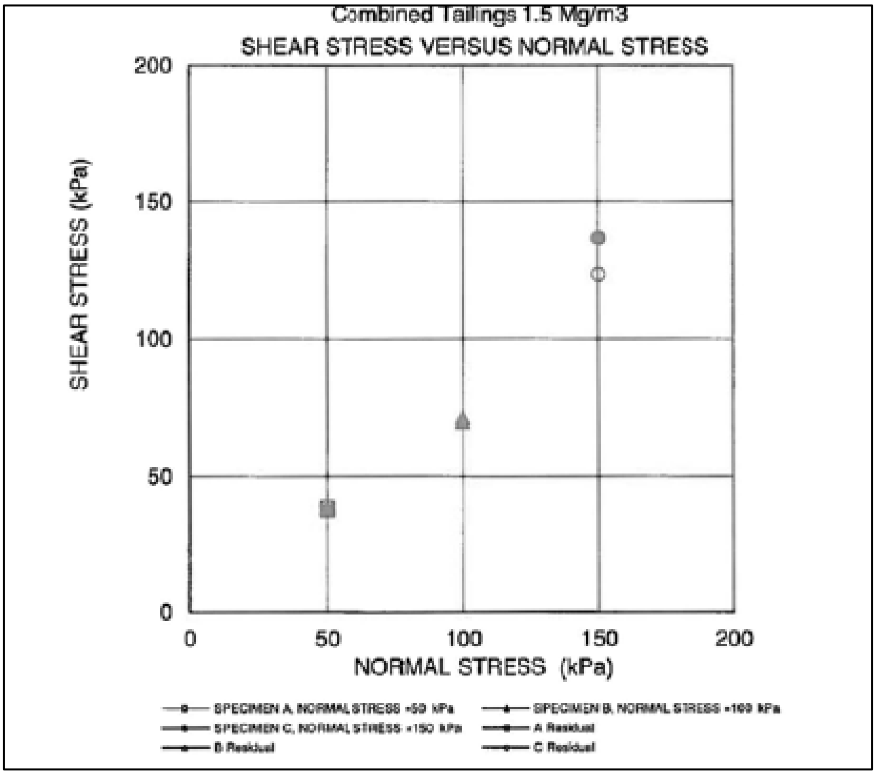
Consolidated drained direct shear test work was undertaken by SGS on the typical coarse split of the tailings' materials, remoulded to a 1.3 and 1.5 t/m^3 density respectively.

Two sets of three samples were tested at the respective densities. The residual shear stress versus normal stress of all three samples is presented below:

- The test results indicated that the average effective friction angle for the coarse tailings fraction is expected to vary between 34^o to 40˚, when consolidated to 1.3 t/m^3 and slightly increases to a maximum of 42˚ when reaching 1.5 t/m^3. No cohesion is expected for the material placed in dry tailings format.
- Permeability performed by constant head permeameter indicated that the expected average permeability could be ranging between 1 x 10^{-4} and 4 x 10^{-6} cm/s.

Refer to Graph 40 below.





Graph 40: Consolidated Drained Direct Shear Graphs

18.1.4 Geological, Hydrological and Geochemical Conditions

18.1.4.1 Local Geology

A site selection study was done by SRK, (Report Number 474874/1 TSF Site Selection Study Rev 0 (Final) and in general Site No.3 was identified as the most appropriate area for the TSF. The design discussed in this report focused on this area. Figure 76 below shows the sites investigated in the SRK Report.

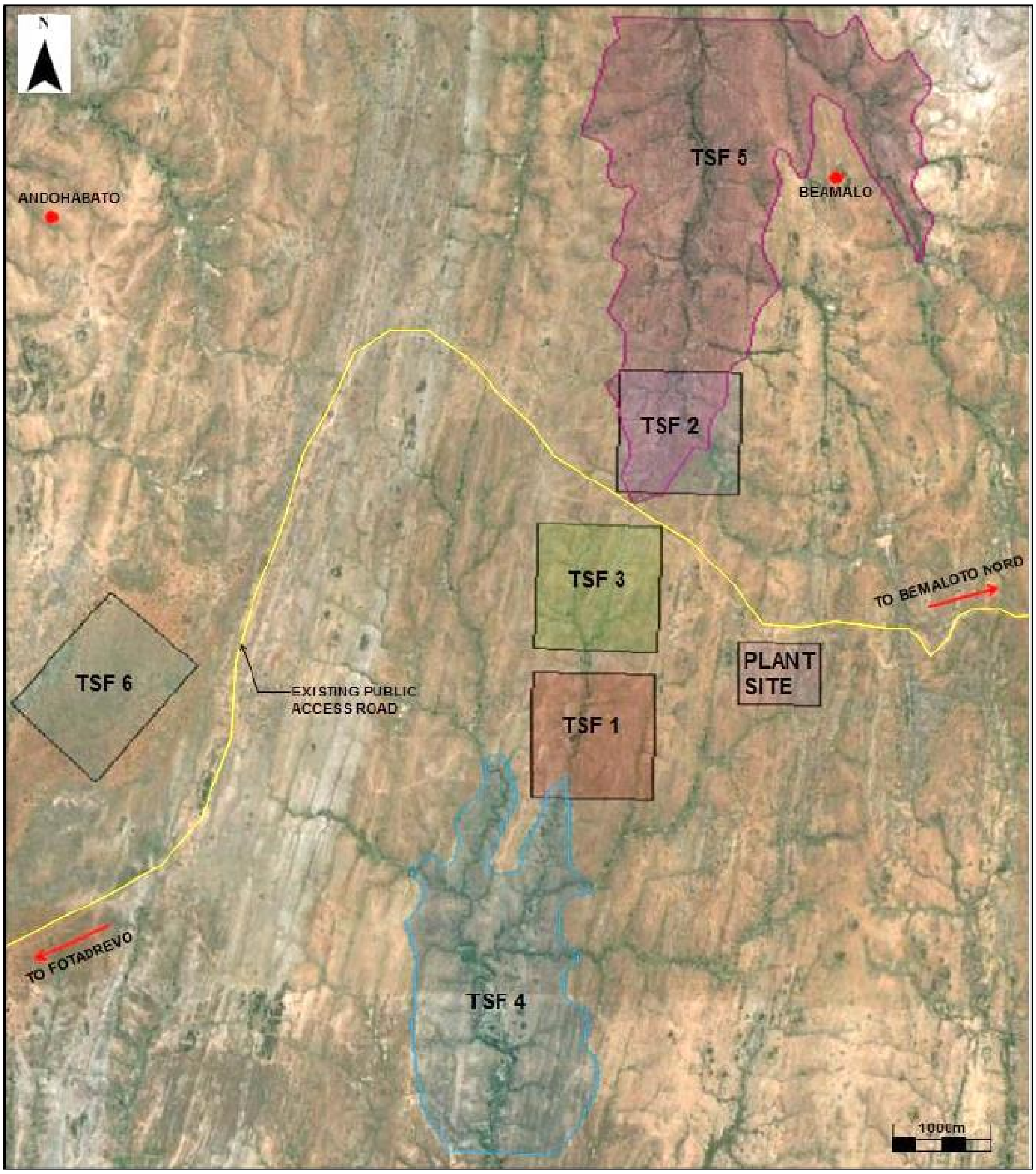


Figure 76: The Proposed TSF Sites

Geotechnical test work was undertaken in trial pits located inside the footprint of the tailings storage facility, (previous layout by SRK as TSF No.3 and is generally in same region as our planned footprint), and the results indicate that the site comprises shallow silty to clayey sand as the transported residual soil, overlying a weathered gneiss bedrock. Refer to Figure 77) for test pit layout and Table 69 for test pit results.

The average depth to the bedrock is approximately 300 mm, and ranges from 0.1 to 2.3m.

The bedrock is mostly covered by transported soil, (23 out of 24 test pits), with an average thickness of 200 mm and ranges from 0.1 to 0.5m. The material is indicated to be a silty sand that generally compacts well and classifies as a G7 material which occurs generally in small volume across the site.

Residual soil was only logged in 2 of the test pits, (TSF 3-12 and 13), and ranged in thickness from 0.3 to 1.1 m. The material is indicated as a silty to clayey sand.

The depth to bedrock within the TSF 3 footprint ranges from 0.1m to 2.4m, but is encountered on average at about 0.3m below surface. The very soft rock was excavatable down to final depths ranging from 0.3m to 2.8m (average 0.95m) before refusal was encountered in rock ranging from very soft, soft, medium hard to hard rock.

Reddish brown to off-white, red brown and black, highly weathered to medium weathered, very closely jointed with sub-vertical joints with sub-vertical north-south orientated foliation planes (at approximately 30 degrees), coarse grained, very soft to medium hard rock, Micaceous Gneiss.

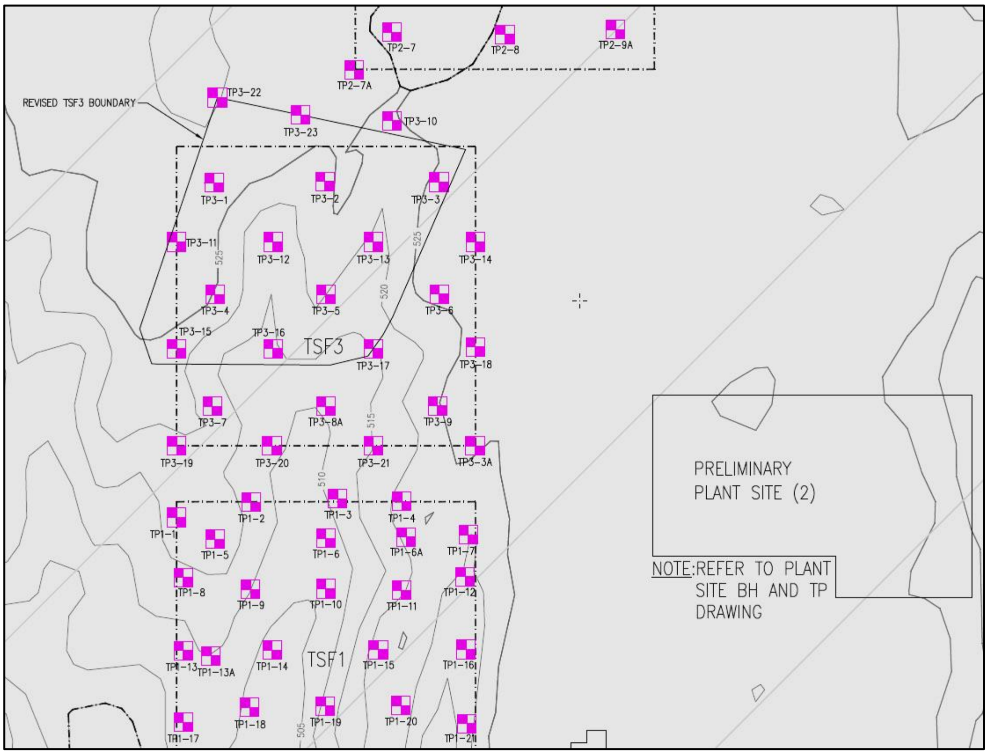


Figure 77: Test Pits at TSF3

Table 69: TSF 3 Test Pit Results

TSF	Test Pit Reference				Depth to Top of Strata in Metres (Thickness in Brackets)														Excavation Method	Refusal / Near Refusal Depth (m)	Ground Water Level / Inflow (m)	Excavation Class (SANS 1400D)
					Colluvium		Residual		Bedrock													
					Silty / Clayey Sand		Silty Sand		VSR		SR		SR-MHR		MHR		MHR-HR					Surface to Refusal Depth
TSF 3	TP	3	-	1	0.00	(0.20)			0.20	(0.90)	1.10+								TLB	1.1-1.8	None Observed	Soft to Intermediate
	TP	3	-	2	0.00	(0.20)			0.20	(0.10)					0.30+				TLB	0.3	None Observed	Intermediate to Hard Rock
	TP	3	-	3	0.00	(0.10)			0.10	(0.70)			0.80+						TLB	0.8	None Observed	Intermediate to Hard Rock
	TP	3	-	3A	0.00	(0.20)			0.20	(0.10)					0.30+				TLB	0.3	None Observed	Intermediate to Hard Rock
	TP	3	-	4	0.00	(0.10)			0.10	(0.60)	0.70+								TLB	0.4	None Observed	Intermediate to Hard Rock
	TP	3	-	5	0.00	(0.10)			0.10	(0.30)			0.40+						TLB	0.4	None Observed	Intermediate to Hard Rock
	TP	3	-	6	0.00	(0.10)			0.10	(0.70)	0.80+								TLB	0.8	None Observed	Soft to Intermediate
	TP	3	-	7	0.00	(0.20)			0.20	(0.60)	0.80+								TLB	0.8	None Observed	Soft to Intermediate
	TP	3	-	8A	0.00	(0.10)			0.10	(1.00)	1.10+								TLB	1.1	None Observed	Soft to Intermediate
	TP	3	-	9	0.00	(0.10)			0.10	(0.20)			0.30+						TLB	0.3	None Observed	Intermediate to Hard Rock
	TP	3	-	10	0.00	(0.50)			0.50	(1.40)+									TLB	1.9	None Observed	Intermediate to Hard Rock
	TP	3	-	11	0.00	(0.10)			0.10	(0.45)	0.55+								TLB	0.55	None Observed	Intermediate to Hard Rock
	TP	3	-	12			0.00	(0.30)	0.30	(0.50)+									TLB	0.8	None Observed	Soft to Intermediate
	TP	3	-	13	0.00	(0.20)	0.20	(1.10)	2.30	(0.50)+									TLB	2.8	None Observed	Soft to Intermediate
	TP	3	-	14	0.00	(0.45)			0.45	(1.35)	1.85+								TLB	1.85	None Observed	Soft to Intermediate
	TP	3	-	15	0.00	(0.30)			0.30	(0.50)	0.80+								TLB	0.8	None Observed	Soft to Intermediate
	TP	3	-	16	0.00	(0.20)			0.20	(1.70)+									TLB	1.9	None Observed	Soft to Intermediate
	TP	3	-	17	0.00	(0.15)			0.15	(1.55)+									TLB	1.7	None Observed	Soft to Intermediate
	TP	3	-	18	0.00	(0.20)			0.20	(0.85)+									TLB	1.05	None Observed	Soft to Intermediate
	TP	3	-	19	0.00	(0.10)			0.10	(0.75)+									TLB	0.85	None Observed	Soft to Intermediate
	TP	3	-	20	0.00	(0.15)			0.15	(0.90)+									TLB	1.05	None Observed	Soft to Intermediate
	TP	3	-	21	0.00	(0.10)			0.10	(0.95)+									TLB	1.05	None Observed	Soft to Intermediate
	TP	3	-	22	0.00	(0.20)					0.20	(0.40)+							TLB	0.6	None Observed	Intermediate to Hard Rock
	TP	3	-	23	0.00	(0.40)					0.40	(0.20)					0.60+		TLB	0.6	None Observed	Intermediate to Hard Rock
	Minimum				*0.00*	*(0.10)*	*0.00*	*(0.30)*	*0.10*	*(0.10)*	*0.20+*	*(0.20)*	*0.30+*		*0.30+*					*0.30+*		
	Maximum				*0.00*	*(0.50)*	*0.20*	*(1.10)*	*2.30*	*(1.70)*	*1.85+*	*(0.40)*	*0.80+*		*0.30+*		*0.60+*			*2.80+*		
	Average				*0.00*	*(0.19)*	*0.10*	*(0.70)*	*0.28*	*(0.75)*	*0.83+*	*(0.30)*	*0.50+*		*0.30+*					*0.95+*		

Table 70: Summary of Soil Properties

Soil Description	Cohesion (kPa)	Phi (deg)	Unit Weight (kN/m³)	Permeability k (m/s)
Transported silty Sand (SM-ML)	0	32	20	1×10^{-8}
Very soft rock Gneiss* (SM-ML)	0	36	24	1×10^{-6}

*Assuming joints and fractures are open to allow water seepage via the rock mass.

18.1.4.2 Borrow Area

Although TSF 6 site from the SRK investigation, Figure 76 above, shows the possibility of borrowing material, observations and further work indicated that the distance and material availability may be problematic. It was, therefore, decided to keep to a more conservative approach and accept that limited construction material will be available, and the required material is to be sourced predominantly from within the footprint of the selected area.

Structural material is viewed to be sourced from the overburden material and waste rock from mining operations.

18.1.4.2.1 Local Hydrology

A hydrological assessment was done by SRK previously and for this purpose the data was reviewed and corroborated with later material made available. For the expansion, this will be re-evaluated during detailed design.

In terms of the design of the TSF, the focus is to allow for storm water management around the structure, as well as for surface run-off on top of the TSF during operations. An operational plan will be developed as part of the design to ensuring the draining of all ponding water away from the TSF within a 48 hour period. This water will be directed to the 2 RWDs to minimise infiltration. The TSF is a dry stack, co-disposal facility.

The site falls within an arid region and therefore the run-off events are expected to be short, rapid, and run-off within the TSF catchment (dirty water). Water collected this way, will be held in the return water dams for potential re-use at the plant. The water from the return water dams is expected to be intermittently available during the rainy season and is not a constant source to the water balance.

The design rainfall depths for the 48 hour 1:50 and 1:100 recurrence intervals are 460 and 510 mm respectively with the area having an average annual rainfall of 830 mm and evaporation of 1,387 mm (Based on data for period 1942 to 1996).

The 2 return water dams have been modelled to ensure that the dams do not reach their maximum capacity more than once within a 1:100 year recurrence interval (98th percentile). The dam capacities are as follows:

- RWD 01 = 336,980m^3.
- RWD 02 = 265,810m^3.

The Figure 78 below shows the local catchment with the north-eastern side being the elevated region and following the water shed line from north-west to south-east.

The TSF stretches over 2 valleys and is to be developed in phases with the eastern return water dam becoming active first.

A storm water cut-off trench is located along the eastern and western flank, leading past the 2 return water dams. The northern flank is located adjacent to the water shed and, therefore, the expected run-off from this catchment is minimal.

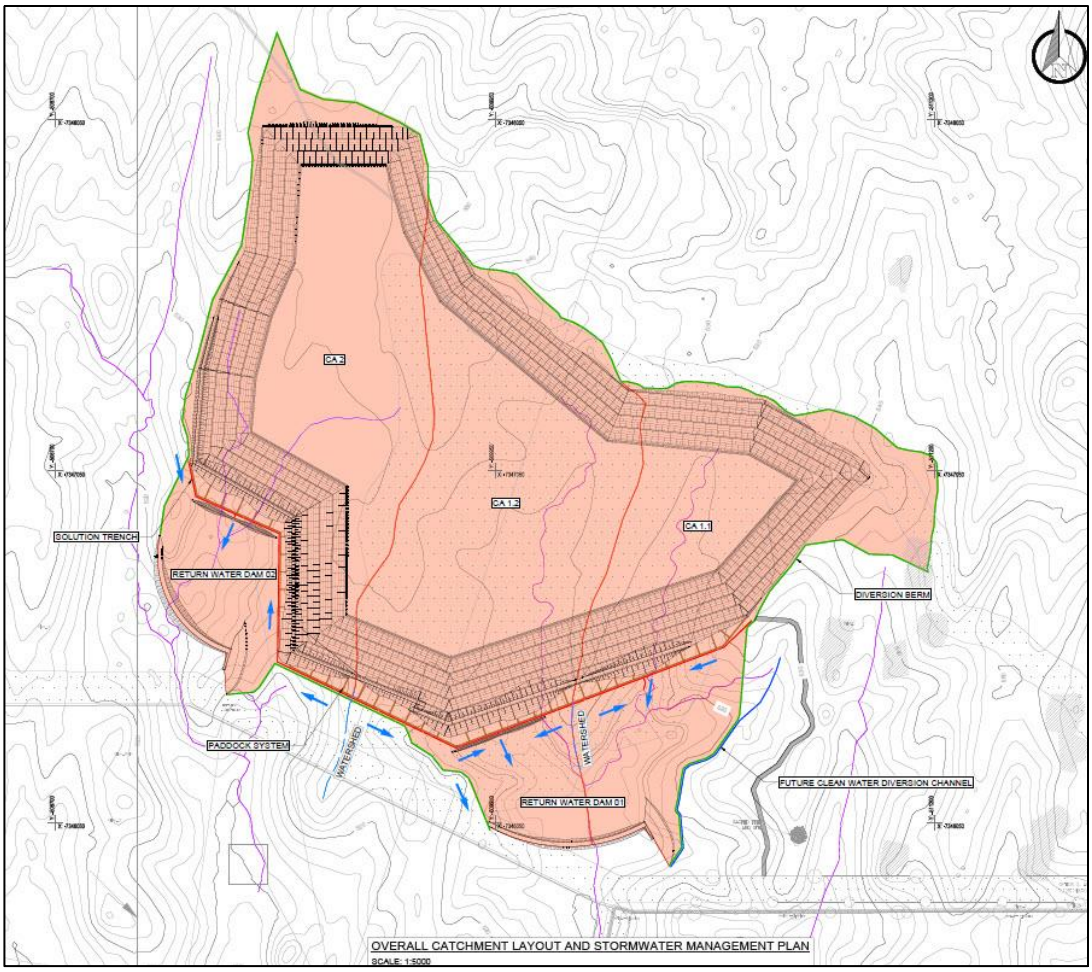


Figure 78: Local Catchment of TSF

18.1.4.3 Leachate and Geochemical

Feedback from the geochemical study by GCS (2015) indicates that the reaction rates of the secondary sulphate minerals in tailings material are generally slow. Acid drainage, or leachate containing metals in high concentrations is, therefore, not expected over the short to medium term. Slightly acidic leachate (up to a pH of 4.5) could emanate over the long term, with potential slightly elevated Al, Mn and Fe concentrations (< 5 mg/l). The sulphate concentration during the operational phase will vary between 200 and 1,200 mg/l (500 mg/l average).

Chloride is also likely to be dominant in tailings water, estimated to be between 250 and 1,300 mg/l during the operational life. Higher salt loads are expected in the long-term (post-closure), with sulphate concentration as high as 3,200 mg/l.

The 2015 geochemical tailings source term assessment will be updated using test results from the Phase 1 tailings and waste rock material and considering dry deposition.

Contaminant transport ground water modelling was undertaken for both liner and non-liner TSF scenarios (GCS, 2015). The contaminant plume modelling indicated that for a wet facility, a liner system would be required to prevent medium and long-term unacceptable impacts to the water environment. It is to be noted that the analysis was based on a wet facility. The mobilisation of contaminants for the dry stack facility is expected to be a fraction, since seepage is expected to be minimal with infiltration from rainfall water being the only source.

Infiltration into the TSF is to be managed by having:

- A stringent storm water management plan. The decision was, therefore, taken not to line the dry stack facility. The ground water contaminant risk, however, will be re-evaluated for the dry stack design and appropriate management measures will be assigned if required. The management plan remains an active document during the operational and closure phase and will need to be updated and adjusted as the TSF grows and its profile varies.
- Mechanical compaction of the tailings material will remain essential and is to the advantage in gaining overall material stability and low infiltration. Please note that the current performance on site shows that a relatively dense, low to very low permeable material is obtained through compaction and furthermore, there is a geochemical reaction, (that is to be studied and further understood), that hardens the tails to a cementation with a level of impermeability.
- Operating the TSF for closure as the end of life the TSF approaches. This generally starts with having a closure design and implementing the measures gradually for at least the last 5 years of the TSF operational life, thereby achieving a more effective final closure shape and having a capped or vegetated or the combination thereof for the top and slopes of the TSF. This further reduces infiltration and achieving a hydraulic disconnect with any phreatic zones within the TSF.

- The decision was, therefore, taken not to line the facility with the dry stack design in contrast to the liner requirement for a wet facility. The ground water contaminant risk, however, will have to be re-evaluated for the dry stack design and appropriate management measures will be assigned once more in-depth studies and material performance on site indicates significant variation.
- No ground water and surface users are expected to be adversely impacted, based on proposed management measures, and maintaining a dry TSF facility which limits the mobilisation of pollutants within the TSF.

18.1.5 Tailings Co-Disposal Design

18.1.5.1 General Overview

For this design, the material is to be transported to the TSF from the plant area by an overland conveyor system, then mechanically placed and compacted within the footprint of the TSF. The waste rock will form the outer containment wall region with the fines being spread and compacted behind this containment wall. The contact region between the coarse and fines will generally be a mix of material and is viewed as a transition zone between the course outer and the finer inner zone.

The TSF is to be developed in phases leading from the eastern side, where the fixed conveyor line and spreader is located. The phases are a gradual development, spreading from the east in layers and gradually filling the basin by loading and spreading of the material. The TSF is then to be raised in these layers until final height (approximately 60m) is achieved.

Development is to be a dry-stack facility with the upstream construction method followed. This decision is based on the TSF will consisting of dry stacked tailings, (indicated 16% to 20% moisture content), and the outer wall being a sturdy compacted waste rock wall. The expected factor of safety is above 1.8 with a ground acceleration factor of less than 0.8 m/s^2. The TSF is also located 2.2 km away from the vibrational activities from mining and from the magazine.

Internal drainage is to be minimal and located along the lower toe regions of the TSF, the expected function of these is for draw down of seepage that may occur from infiltration during rain events. Ground water is indicated to be more than 4.0m below the TSF basin. Storm water is to be stringently managed and infiltration of rain water minimised for the facility to remain inherently dry with minimal seepage.

Refer to Figure 79 below.

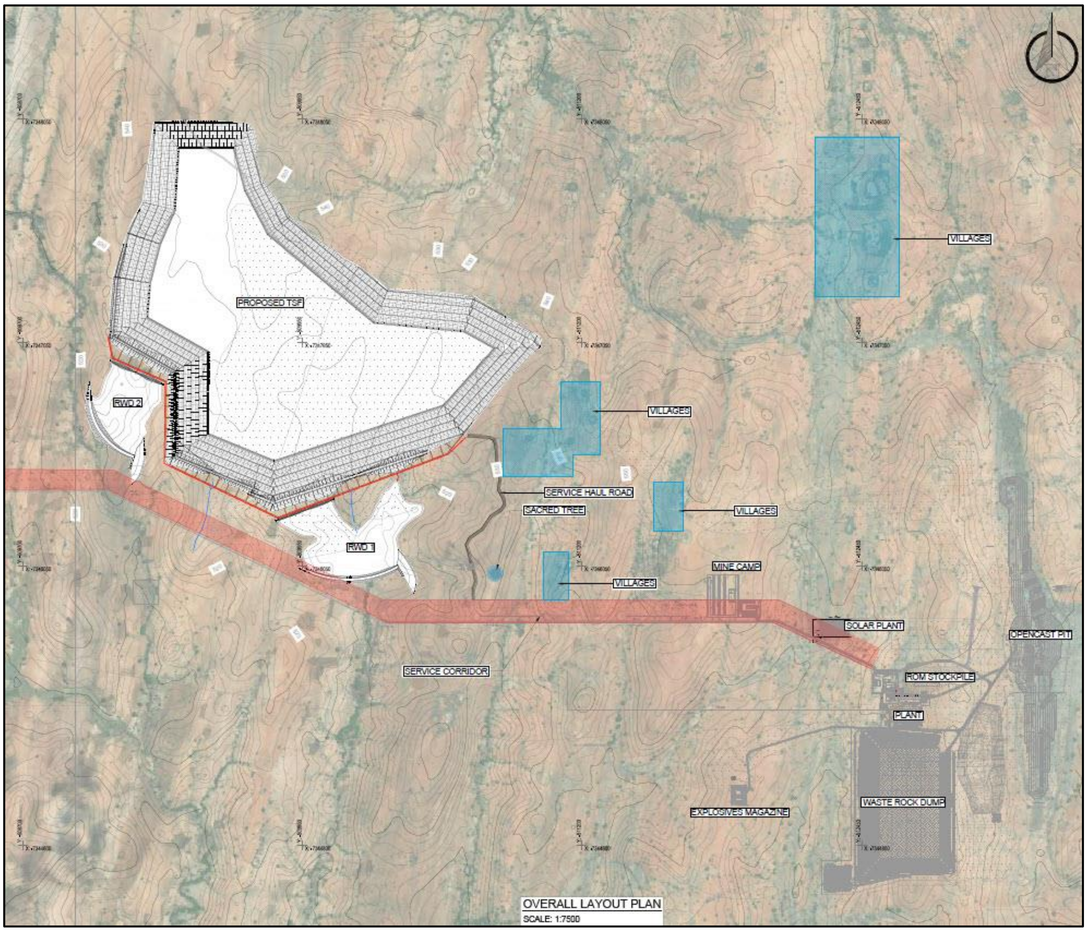


Figure 79: Layout of TSF

18.1.5.2 Wet Deposition vs Dry Stacking

In this case the generic terms used as wet deposition and dry stacking refers to:

- Wet deposition, being the more conventional slurry being pumped to the TSF and then deposited, typically either by spigot or hydro cyclone.
- Dry stacking, being the typical filter pressed tailings that is hauled and mechanically placed and compacted.

In brief the considerations for each are shown in Table 71 below.

Table 71: Wet Disposition vs Dry Stacking Considerations

Area	Wet Deposition	Dry Stacking
Construction / Retention Infrastructure	Significant starter walls are to be constructed that applies a resistant force to the saturated tailings that is contained and mitigate the low stability geomembrane. In the case of Molo TSF 3 the site has not enough source material (especially impervious) and an external burrow will be required.	Starter walls are to be constructed and requires a high level of source material. However, the material can be more pervious (waste rock) with the focus on structural material.
TSF Water Management	The facility has a high level of saturation and requires a significant level of pool control, internal drains, and deposition control to have a wide enough coarse zone for stability. In the case of Molo TSF 3 the facility will require a significant quantity of drainage pipes and filter sand material. Both will need to be commercially sourced with the Geosynthetics and HDPE being imported at a high cost and logistics.	The tails still have a level of moisture, but mostly not sufficient to contribute to seepage and internal saturation of the structure. Internal drains will be mostly to manage the limited infiltration emanating from rain events. In the case of Molo TSF 3 the storm water management plan and shaping of the TSF will be pertinent but less reliant on the importing and commercial sourcing of drainage material.
Water Consumption	A wet facility is heavily dependent on free water, with a significant volume being locked in as interstitial water (20 – 35%). A volume is then also consumed through losses and evaporation through the conveyance cycle, including the drainage to the solution trenches and storage in the RWD.	Through the filter process, moisture is removed with 16 to 20% remaining in the tailings. The material is further dried through the handling process. Most of the water is directly returned to the plant and has a significant less exposure to losses e.g., evaporation. In the case of Molo TSF 3, the area is viewed as a water scare area and water losses must be kept to a minimum.

Area	Wet Deposition	Dry Stacking
Pollution Mitigation	The high degree of saturation in a wet facility leads to a significant level of seepage and the migration of pollutants into the environment. To mitigate this a Geomembrane needs to be installed and an extensive drainage network is required.	With proper management of storm events and exposure to precipitation, the level of saturation is limited to the moisture of the tails and infiltration from storm events. The degree of seepage and mobilisation of pollutants is minimal in comparison to a wet facility and can be mitigated by a focussed and small network of internal drains.
Operational / Stability	Slurry is hydraulically deposited, and the system is dependent on the segregation / separation of the course and fines material. In the case of Molo TSF 3 this can be achieved by hydro-cycloning (Indicated). However, the strength gain is dependent on the natural consolidation of the material that results in a low annual rate of rise. To mitigate this constraint, a bigger footprint will be required which is challenging based on the site selection study.	Material is mechanically placed and compacted. Therefore, this process is plant intensive and requires proper planning in terms of stacking development and traffic management. A high annual rate of rise can be achieved.
Deposition Planning	Deposition is usually in cycles along the outer edge of the TSF. These cycles and layer thickness of each deposition is determined by the material properties. Access on top of the TSF is limited to the outer edge and using the walkway that leads to the penstock.	Deposition is managed through a deposition and TSF development plan that must be strictly adhered to. The entire surface on top of the TSF is utilised for access, which is challenging, especially during the rainy season. During this period, certain areas will not be accessible, and a vehicle access plan is pertinent to allow for sufficient deposition space.

18.1.5.3 Truck Haul vs Conveyor System

Tails generated from the Molo Mine operations, need to be transported to the TSF located approximately 2.2 km east from the plant operations. The elevation variation over this distance is approximately 10m vertical.

The method of transporting the dry bulk material to site needs to be considered. The 2 options for consideration are:

- Hauling with trucks.
- Overland conveyor and stacker system.

Refer to Figure 80 and Figure 81 below.

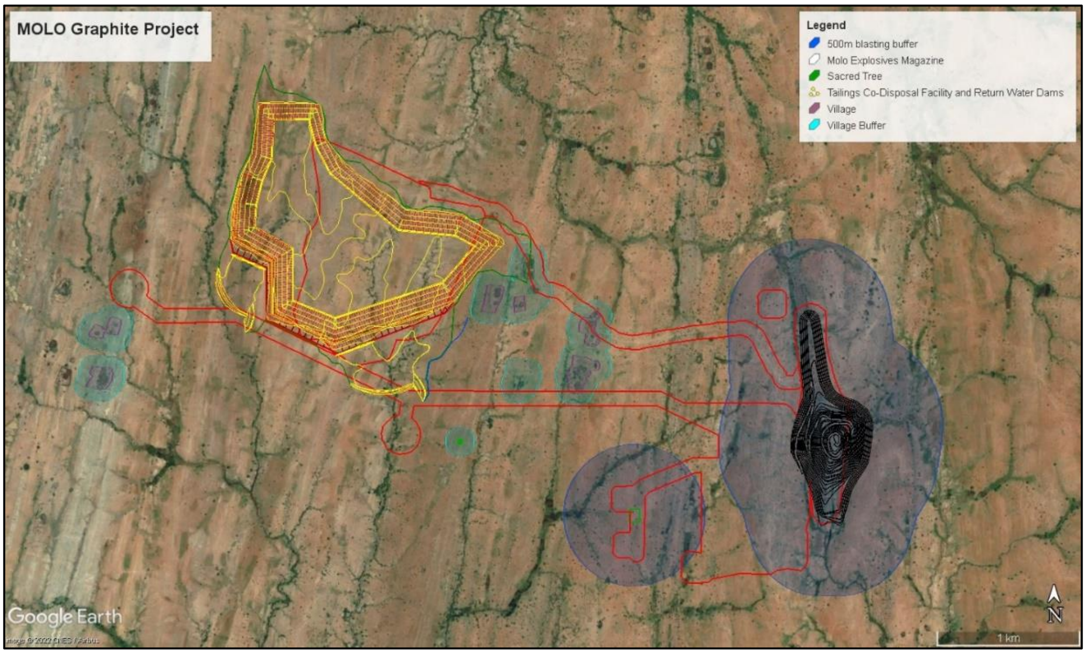


Figure 80: Molo Graphite Project Site Layout

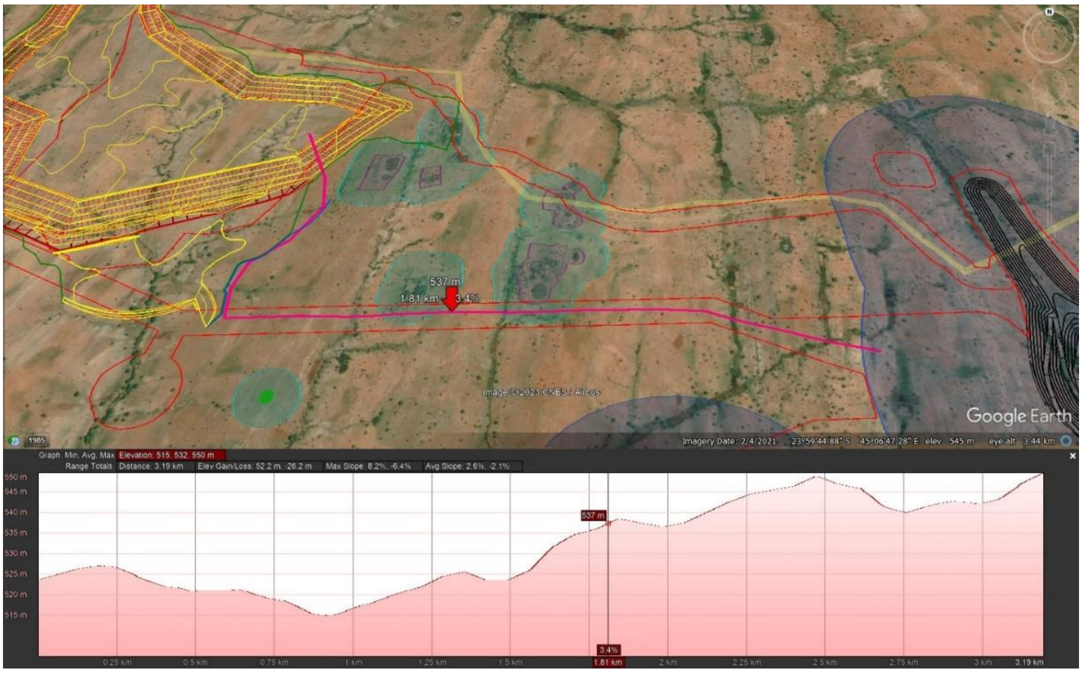


Figure 81: Haulage Profile

18.1.5.3.1 Considerations

The analysis was done as a brief overview for the expansion of the project and an in-depth analysis needs to be completed once more information is available. The items considered were as follows:

- Fuel and Power:

  Fuel availability for the trucks is a concern and needs to be transported from a source assumed to be at least 150 km away. Power towards the conveyor system could be generated by the planned solar system. The increase in power demand as base case for the mining activities should not be significant, assumed to be a maximum 8%.

- Availability, Maintenance:

  In general, haul trucks are expected to be available almost continually with proper maintenance and scheduling. The only expected down time can result from problems at the loading, or offloading facility, or from a major interruption on the haul road itself, (road deterioration, slopes, and safety barriers).

  The conveyors are likely to have a lower availability, based on the complexity of the system, including the stacker interface. The resulting lower capacity can be off-set by increasing the design capacity of the belts and allowing for larger stockpiles. Furthermore, planned belt downtime should be scheduled to coincide with plant maintenance.

  A large truck workshop is required to maintain the hauling fleet. The conveyor belt can be maintained by a smaller team with a smaller workshop.

- Labour:

  The truck hauling option is likely to require more staff, while the conveyor system will require a smaller crew. This reduces the strain on the employee facilities, as well as general overheads.

- Environmental:

  Belt conveyors operate relatively quietly, and dust is an issue only at loading and discharge points, where it can be contained and dealt with.

- Health and Safety preferred:

  Belts can be monitored and operated remotely from a central control room, removing operators from possible risk areas. There will be reduced risk of night time accidents as other users will not be able to utilize the conveyor route to the degree possible with a haul road.

  Pedestrians are at risk with the haul truck option to a greater degree than with the overland conveyors. The potential risks of unauthorized access are consequently lower.

- Closure:

  Overland conveyors can accommodate steeper gradients than road trains and thus the need for cut and fill is reduced.

18.1.5.3.2 Conclusion

Based on the information available, it is currently assumed that the overland conveyor is preferable to the truck haul system.

18.1.6 TSF Capacity

The structure is to contain the tails material generated by the mining activities and, in concept, to have the capacity to cater for a LoM of 25 years and a tailings feed that consists of:

- 51.6 Mt Fine Tailings.
- 1.63 Mt Waste Rock.

The facility will, therefore, have a storage capacity of 58 Mm^3 with a height of 60m along the highest slope. The footprint of the TSF will be approximately 185 ha.

The Figure 82 below shows the waste rock starter wall and overall sections with the layer developments in colour. The outer coarse region is indicated as a rock hash overlay with the starter wall development in a pink colour.

The expected volume to be deposited and, months expected to complete the lift, is shown below for the LoM:

- Lift 1 – 1.031 Mm^3, 5 months.
- Lift 2 – 6.668 Mm^3, 35 months.
- Lift 3 – 12.979 Mm^3, 70 months.
- Lift 4 – 14.423 Mm^3, 80 months.
- Lift 5 – 13.083 Mm^3, 74 months.
- Lift 6 – 9.683 Mm^3, 48 months.

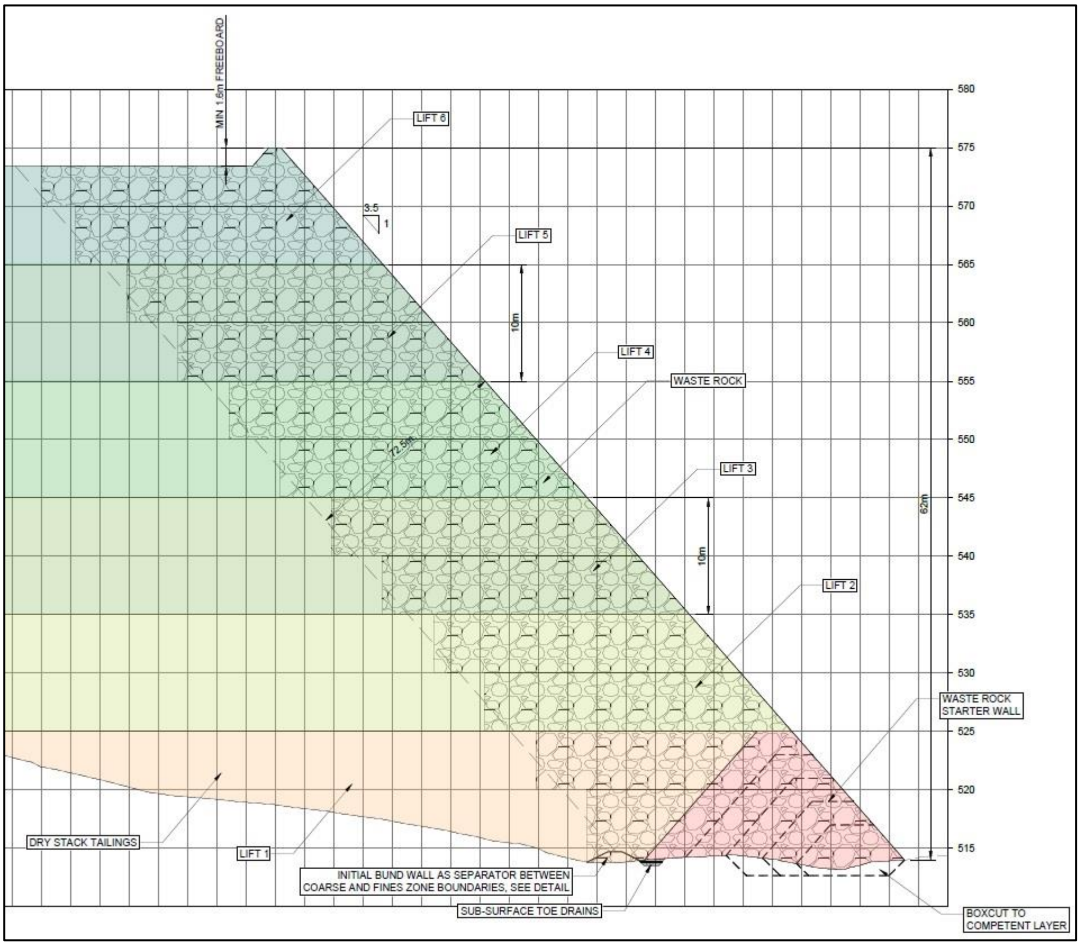


Figure 82: TSF Typical Section

18.1.7 TSF Structure and Development

18.1.7.1 Founding and structural

The geotechnical and site selection studies available gives a clear indication of shallow foundation conditions being predominantly along the starter wall and TSF basin. Available soil material for wall building is only available at TSF 6 position (Figure 83) which is a significant haul distance.

The design, therefore focussed on:

- Removal of spoil material down to competent base material, (expected as shallow excavations). This spoil material can be used as part of the wall building, especially at the return water dams if found suitable.
- Ensuring that the box cut region under the starter walls remains as rugged and free of loose material as far as possible, (improving interfacing and grip between TSF and in-situ material).

- Having a starter wall with a wide base and at least 10m in height. The weight and the wide base of such a structure assist in a better resistance to the forces exerted by the TSF.

Waste rock is identified as the primary source of structural wall building material, but needs to be limited to a maximum of 300 – 400 mm in size.

18.1.7.2 Internal Phreatic Management

The facility is to be a dry stack TSF with the tailings being mechanically deposited and compacted. Therefore, the TSF is expected not to have any internal saturated / phreatic zone with only localised, short-term saturation from rain events.

However, some water is likely to permeate through the outer coarse material zone. An internal toe drain is allowed for along the low laying regions to intercept any infiltration and convey this to the RWD. This flow is expected to be seasonal and limited.

No fountains, or water sources were observed from aerial imagery, however, if any are to be found within the basin they are to be isolated by constructing a localised filter region to collect the water and convey the water through an outfall line to downstream of the clean and dirty water diversion bunds.

18.1.7.3 TSF Development

The development of the TSF is done by loading material from the stockpiles at the spreader, (eastern corner), and then hauling the material to the developing layer where the material is end tipped, spread, and compacted in layers to the design requirements. The first layer development is downstream against the starter wall as indicated with the orange / grey zone in the first 2 templates below. This will fill the first 2 valleys with clean water run-off being diverted along the western and eastern sides.

This base layer is then further lifted to fill these valley formation as indicated as the green zone in Lift 2 slide. The lift will in essence develop all along the planned starter wall and grow upstream into the valleys within the footprint. The third lift will basically cover the entire footprint and start raising above the starter wall. The fourth to sixth (final) lift will cover the entire footprint and raise the slope downstream and above the starter wall.

It is important to note that this development is to be carried out in layers in such a way that storm water is not allowed to pond on the exposed top surface of the TSF and must be drained as quickly as possible through run-off channels. The development of each of the 6 lifts / layers is approximately 10m thick and will have an advancing face originating from the eastern side to the west.

The detail design will also need to consider the construction and maintenance of dedicated access corridors within the basin, (where competent coarse tails are used), to allow vehicular access during and after rain events.

This advancement is achieved by end tipping and spreading the material in layers not thicker than 0.4m. The layer is then compacted by a sheep foot roller. The coarse zone is always to be a minimum of 1.6m above the fines material layer to limit material run-off and to function as a barrier for the plant during construction.

The Figure 83 below shows the initial lift and final lift for the complete development, refer to the drawings in the appendices.

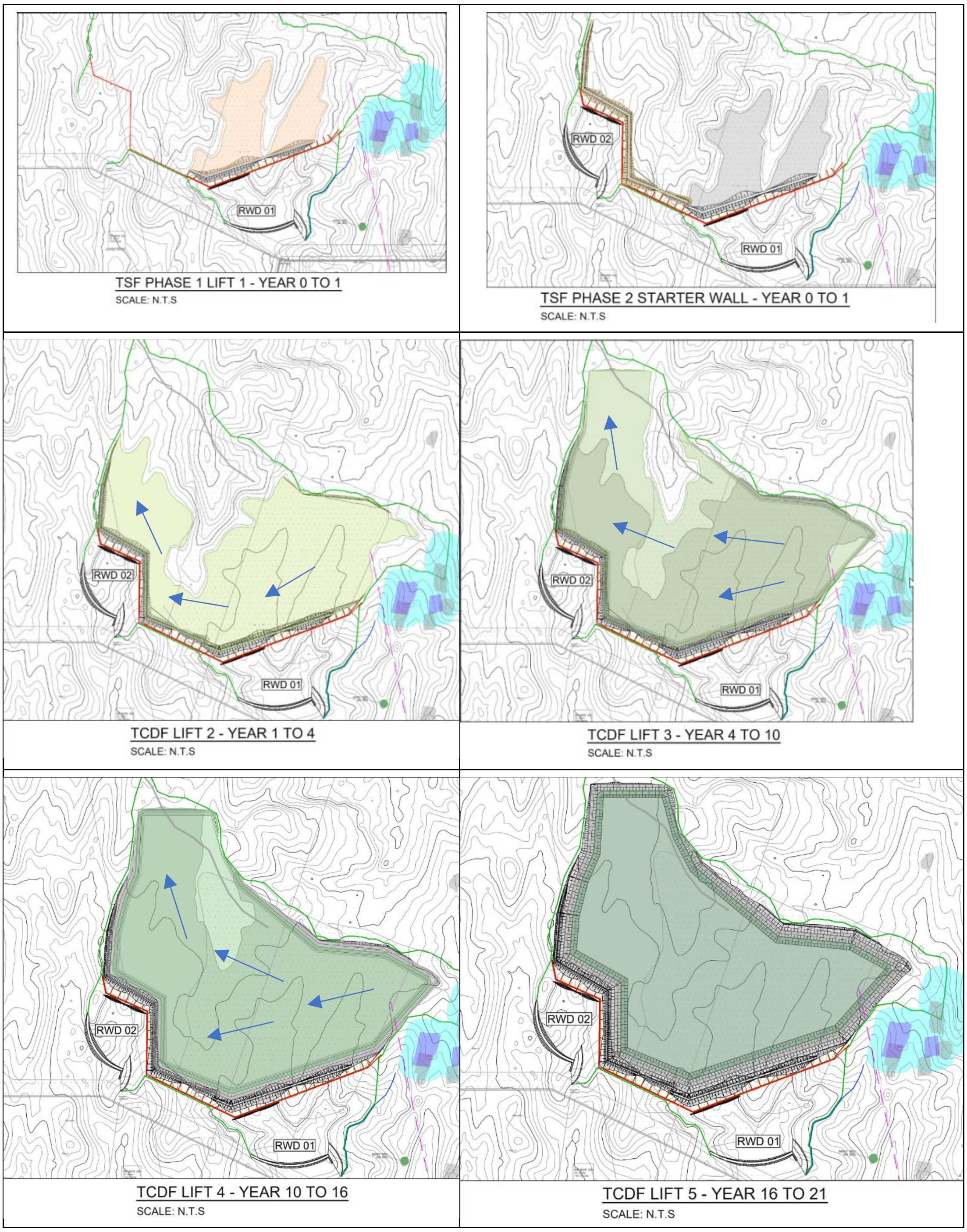


Figure 83: TSF lifts and basin development

18.1.7.4 TSF Shaping and Storm Water

Clean water run-off from upstream of the TSF is to be diverted through clean water channels around the TSF (Figure 86 outer green lines), thereby having a clean and dirty water separation. These channels also limit the dirty water catchment to a minimum and, therefore, reduce the size of the RWDs.

Dirty water along the toe region of the TSF is collected by lined, solution trenches (Figure 84, outer red lines) that leads into the RWD.

The top of the TSF is always to be shaped with a minimum of 2% slope towards the eastern and western edge that leads into constructed run-off channels, (constructed as part of operations and guided by the storm water management plan), that flows into the solution trenches.

18.1.7.5 Dust Generation and Mitigation

The current design is more prone to dust generation than a conventional TSF and requires careful planning during the operations. To mitigate dust generation, the design allows for a coarse outer wall zone that protects the slopes from generally dust generation. The material is further compacted in layers as the TSF develops that ensures that the material is properly interlocked and less prone to be dislodged and lifted by windy conditions.

An observed condition on the active site is the geochemical interaction of the material that creates a hardened material through a cementation process. The understand of this chemical interaction is imperative and is viewed as an opportunity to assist in dust suppression that can be used concurrently with tailings compaction process.

The surface exposed during the lift development on the TSF must also be managed and optimised to ensure the minimum surface is exposed to the elements and risk of dust generation.

18.1.7.6 Operating for Closure

It is good practice and must be adhered to in terms of the TSF that an updated TSF closure plan be in place at least 5 years before end of life of the TSF. This allows for the planning and operating of the TSF for closure which minimised the sudden “rush” at the end of TSF life to shape and prepare the TSF for the closure. Closure costs, in part, is then also absorbed as operating cost of the TSF.

The facility will have a course waste rock outer surface that is less prone to erosion and instability over time. The slopes of the TSF are to be rehabilitated as the TSF develops by encouraging growth on the slopes. During the decommission phase the drains must remain operational and maintained. The final closure would involve the shaping of the crest surface, vegetation of the surface and introducing measures to avoid infiltration into the TSF.

Plants and grass will be guided by the environmental studies and recommendations thereof.

18.1.8 Risks and Mitigations

The following risks are considered and need to be mitigated and managed:

- Dry Stack Facility:

  The structure is designed to be a dry stack facility and needs to be operated as such. Storm water must be drained as soon as reasonably possible from the structure and no standing water is encouraged on the TSF. Since the structure does not have a supernatant pool, the risk of over topping is purely from not having storm water management in place and/or not following the stage development design of the TSF. Water infiltration will also affect the overall stability of the TSF and must be kept to a minimum.

- Material Balance:

  It is imperative that the outer containment wall is always higher than the minimum required freeboard to keep the TSF stable and ensure that over topping does not become a risk. Therefore, the material feed must be sufficiently planned so that the development of the outer wall always leads to constructing the TSF.

- Material Sources:

  The design is based on the geological information available. Variability is likely to occur in geological terms and any of these variations must be reported to the responsible TSF engineer, to be considered in terms of the design and construction of the outer wall.

- TSF Operator and Engineer:

  The scale and understanding of the development of the TSF is relatively complex and will require constant adjustment to onsite conditions, (e.g., rainy season, material source, production rates). It is, therefore, imperative that a competent team is identified and tasked with the operation and closure of the TSF. The team will consist of the client representative, TSF engineer and an operator and their team members.

18.2 Mine Infrastructure

18.2.1 Explosives Magazine

The explosives magazine consists of fully double fenced containerised storage facilities, fully enclosed by a retaining earth wall on an engineered terrace. The facility is sized to house 4 x 40 ft containers. During Phase 1 only 2 storage containers were purchased, for the expansion an additional 2 storage containers will be procured and installed.

Access to the explosives magazine will be controlled by the Blasting Contractor who is appointed to undertake the blasting on site, and will also be managed in conjunction with the local police services, (Gendarmerie Nationale) in the area.

18.2.2 Access Road to Explosives Magazine

The access road to the explosives magazine was constructed in Phase 1 of the Project. No additional work in this regard is envisaged for the expansion.

18.2.3 Main Haul Roads (Ex-Pit)

The main haul roads between the plant and mine terrace were mostly constructed during Phase 1 of the Project, however, additional sections will be required to accommodate for the crusher feed of the project expansion. The main haul roads have been designed to accommodate the mining fleet.

The haul roads are a minimum of 15m wide and accommodate a 2 way traffic with no requirement for a berm in the middle of the haul road. A berm is included on the outer edge of the haul road, minimum of 1.5m high.

18.2.4 ROM Tip Ramp

A second ROM tip ramp will be required for the expansion and has been included close to the Phase 1 ROM tip ramp to reduce the haul distance and additional hauling road. Only the tip wall will be constructed by the Contractor and the tip ramp will be constructed from overburden stripped by operations.

The ROM tip wall will be constructed form reinforced earth. The trucks will dump directly into the primary crusher feed bin as opposed to Phase 1 where the crusher feed bin is fed by a FEL.

18.3 Process Plant Infrastructure

The infrastructure to support mining and processing plants will be augmented and increased in size to accommodate the increased capacity and requirements brought on by the expansion.

18.3.1 Power Distribution

The plant sub-stations shall be prefabricated and installed on concrete bases with plinths, and with steel access stairways and landings. Access will be via the ends of the sub-station with one side having a service door and pedestrian access on the other side.

All guard railing to the landing on the access side will be removable to allow for equipment to be installed and removed, as required, throughout the duration of the mine’s life.

The plant sub-stations shall be powered from the Consumer Sub-station (IPP), via an 11 kV ring feed. Typically, these MCC’s shall be rated at minimum of 50 kA, 3-ph, 380 VAC, and be of a Form 3B isolation and construction format.

It is envisaged that 6 x 380 VAC MCCs, as well as 2 x 11 kV Sub-stations are placed strategically in order to be closest to the loading, reducing cable runs and installation costs. Distribution to these loads shall respectively be on 600/1000-XLPE/PVC/SWA/PVC and 6.35/11-XLPE/PVC copper power cables via 11 kV Ring-main units and step-down transformers refer to Figure 84 to Figure 89 below.

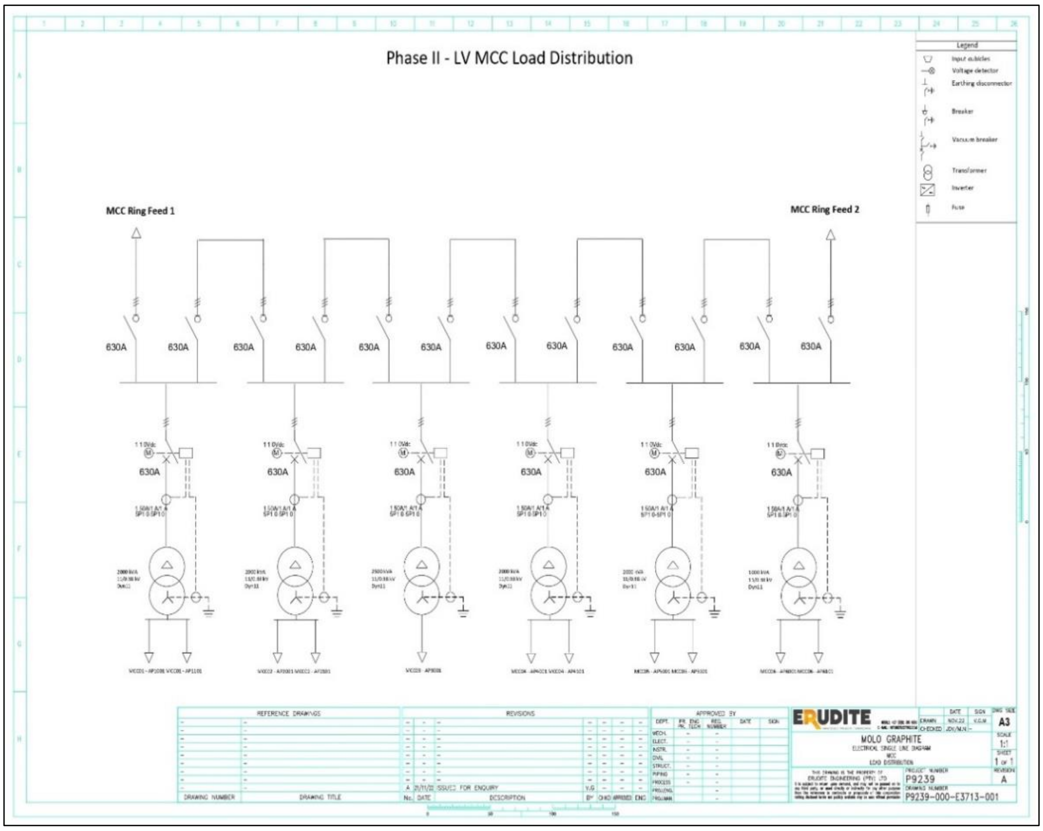


Figure 84: Plant Area: Main Power Reticulation

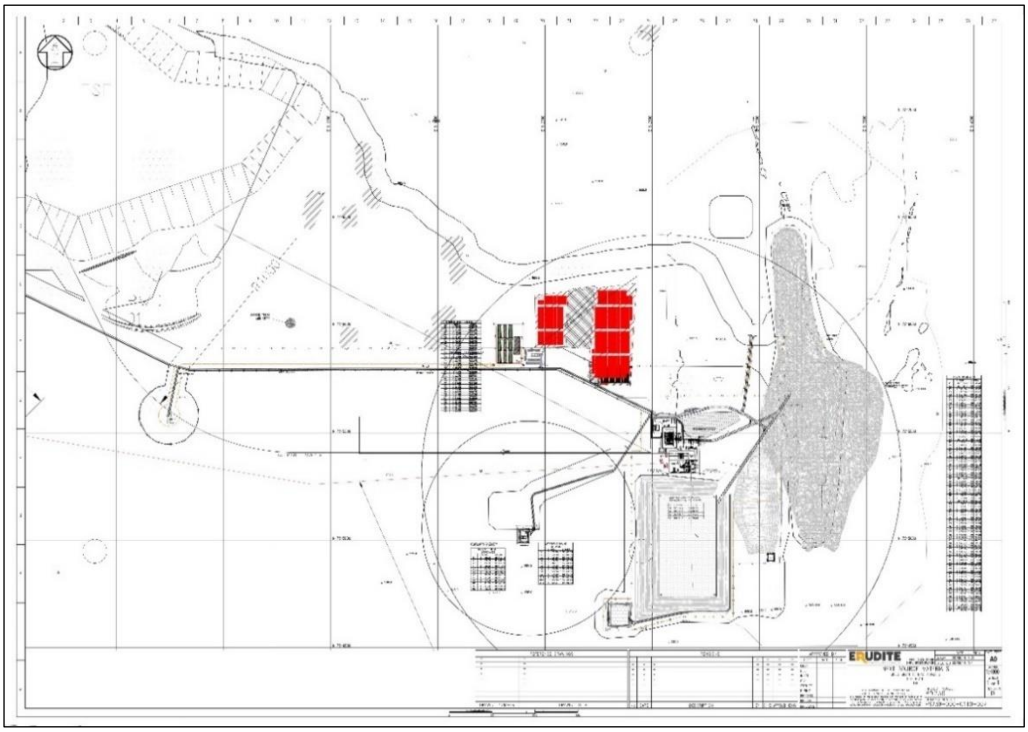


Figure 85: Site Plan Plant

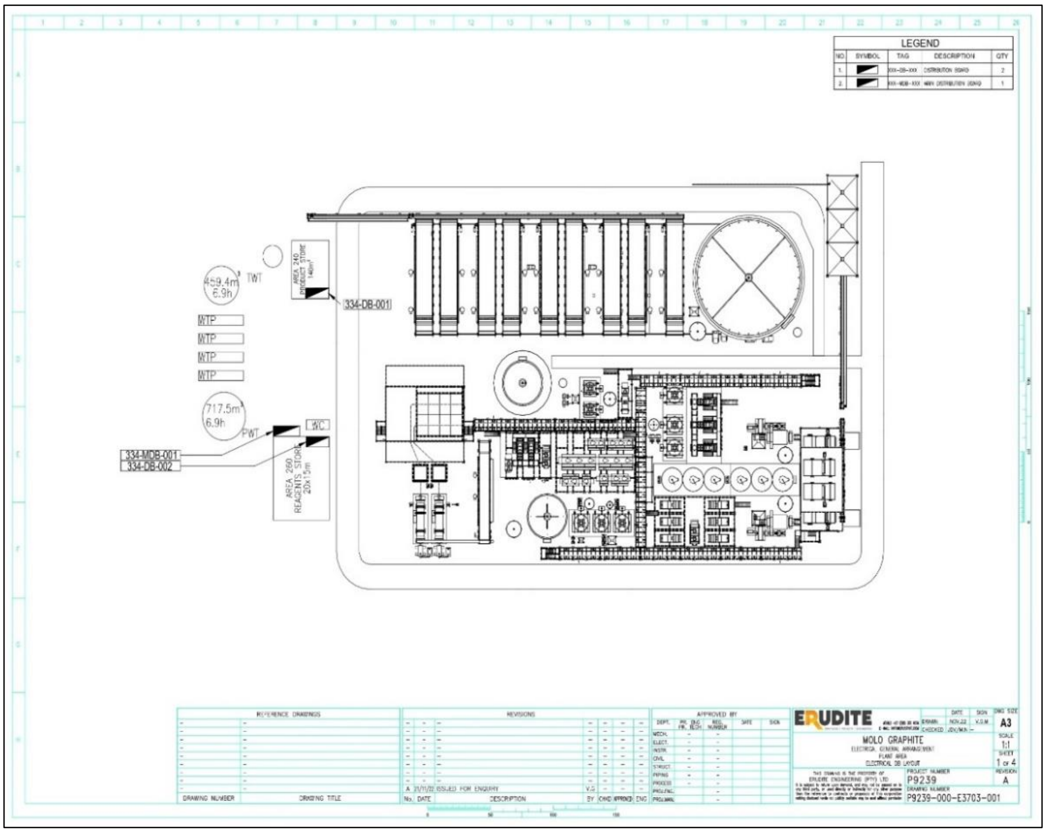


Figure 86: Plant Area - Power Distribution

18.3.2 Lighting

The plant and infrastructure general area lighting shall consist of two parts, namely solar powered, or mains powered.

The Solar Powered Pole Mounted Luminaire shall be the LED type and shall be used alongside the roads.

SSLXPRO 80 solar street light with 210W/18V mono crystalline solar panel, MPPT controller and LiFePO4 batteries.

SOL/SP080 8m mounting height surface mounted heavy duty HDG steel pole complete with SOL/BC/SOL/SP080 HDG bolt cage for above.

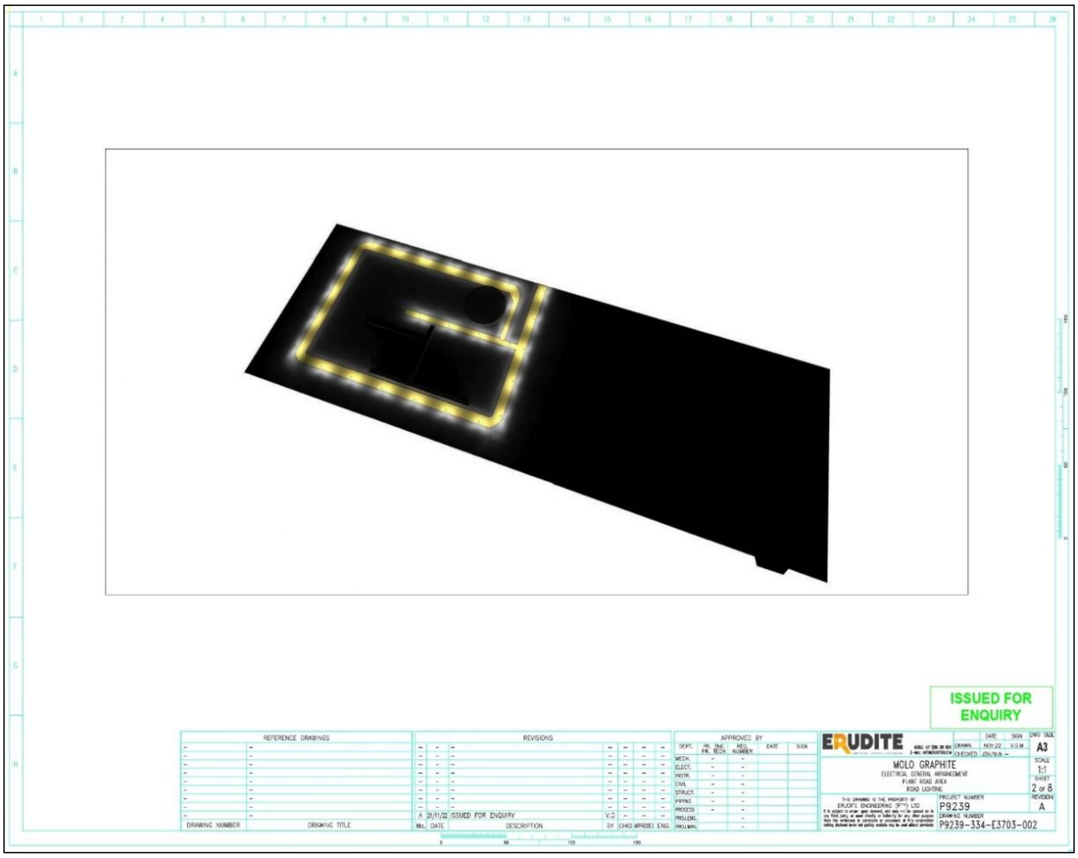


Figure 87: Access Roads - Solar Lighting

The high mast pole mounted luminaire (powered from the mains) shall provide general lighting to the plant areas and be of the LED type, HP30 250W Flood. These lights shall be mounted in groups of 4 on 12m high scissor type masts.

Luminaires and control gear shall be mounted for ease of removal, re-lamping, and servicing.

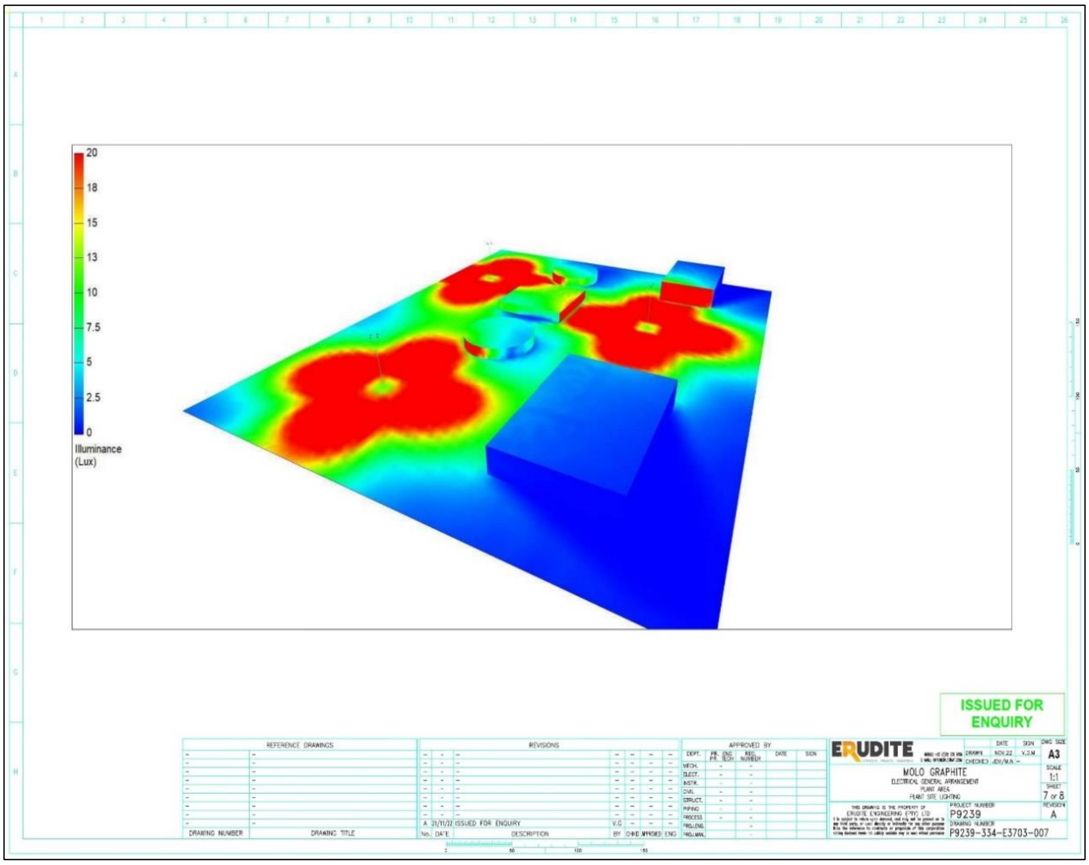


Figure 88: Plant Area - High Mast Lighting

18.3.2.1 Plant and Infrastructure Earthing

The earthing of the plant and the infrastructure is based on the internal equipotential bonding to the internal steel work, as well as to the trench earth. Buildings shall be protected from lightning using lightning masts. Protection shall be accomplished in accordance with IEC 62305: All Parts: Protection against lightning: Physical damage to structures and life hazard. The Lightning Protection System shall be designed so that it affords maximum protection and reduces the risk of damage being caused by lightning.

Designs may incorporate rolling sphere, mesh, or angle of protection principles, but full design data shall, in any event, be furnished by the Contractor as part of the deliverable detailed design. Refer to Figure 89 below.

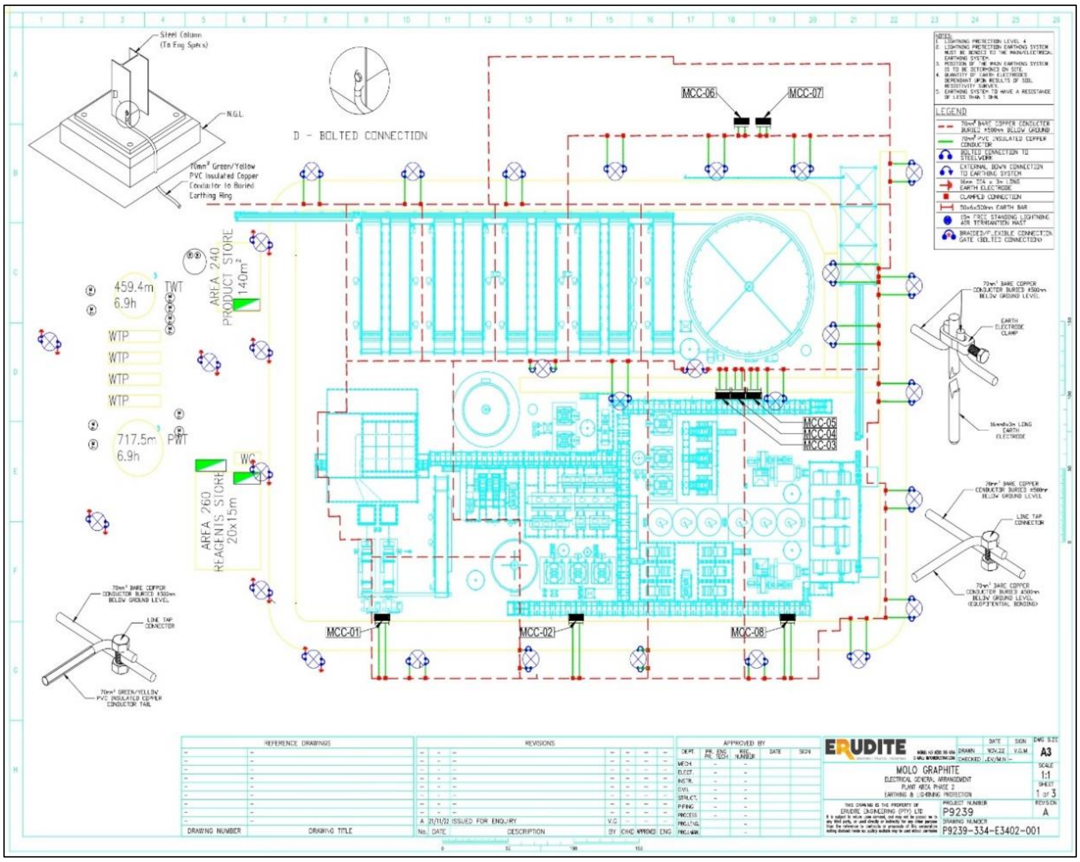


Figure 89: Plant Area - Earthing

18.3.3 Plant Office

The plant offices will be expanded to make allowance for an additional 90 desks and this will result in a facility 5 times the current size.

18.3.4 Plant and LDV Workshop

Plant and LDV workshops will be expanded and increased to 3 times the size of the current facility totalling 1,255m^2.

18.3.5 Laboratory

The laboratory as constructed for Phase 1 will be doubled, by adding an additional 40 ft laboratory container, as well as an additional 40 ft sample storage container.

18.3.6 Change House

The Change House provided for Phase 1 of the Project will be extended to accommodate the increased personnel loading. Separate ablution facilities will be provided within the expansion. All effluent from these facilities will be pumped to the new expanded, sewerage treatment plants.

18.3.7 Reagent Stores

The reagents building is located adjacent to the new plant. The building will serve as storage for the solid reagents required for a period of 3 months. The required storage for the bags, when double stacked, is 200m^2 area, a store of 300m^2 has been allowed for to accommodate the movement of the forklift.

The liquid reagents, (frother and collector), will be delivered by a tanker, and storage has been allowed in the plant.

18.3.8 Product Store

The product building is located adjacent to the new plant. This building will serve as storage for 5 days of product which will require a total stored area of 1,200m^2 (when double stacked). A total floor area of 1,500m^2 has been allowed to accommodate the storage requirements and the movement of the forklift.

18.4 Shared Infrastructure and Services

The shared infrastructure and services to support the processing plant operation and the mining operation was developed by Erudite and positioned to optimise the overall mining and processing operation.

18.4.1 Sitewide Water Management

A sitewide water management assessment has been undertaken as part of the FS which includes a Storm Water Management Plan (SWMP) that has been developed in accordance with the IFC Environmental Health and Safety Guidelines for Mining (IFC, 2007) to mitigate potential water contamination and to ensure a safe working environment. The SWMP was set up to minimize and manage the loss of the water resource. The dirty water footprint is minimized and the use of dirty water in the ore beneficiation process is optimized.

The proposed SWMP measures includes the implementation of Pollution Control Dams and Return Water Dam (RWD) at the proposed TSF Facility. Additional measures proposed includes sediment traps where the drains flow into the Pollution Control Dams to reduce the amount of silt that deposits in these dams.

The operational philosophy of mining and processing operations will be to operate the water system as a closed system which would not discharge into the environment. PCD water will, as far as possible, be used as a make-up water for dust suppression. The plant PCD will provide buffer storage capacity for this. Discharge from the PCDs will, however, occur and will need to be carefully controlled and monitored. This will be re-evaluated for the expansion by re-modelling the salt balance model for the Life of Mine, based on new information from geochemical tests and modelling. PCD water under discharge conditions will comply with at a minimum the Madagascar criteria.

The ranking of water sources to be used in the ore beneficiation process, in order of priority, are likely to be:

- PCD water from storm water / seepage from plant.
- TSF , reject dump, ore stockpiles and open pit water.
- Raw water sources (Wellfield). The raw water tanks will receive water from the wellfield system.

Both the process and storm water dam at the plant will be lined with a 2 mm HDPE liner to prevent water losses into the underlying ground. (Figure 90 below).

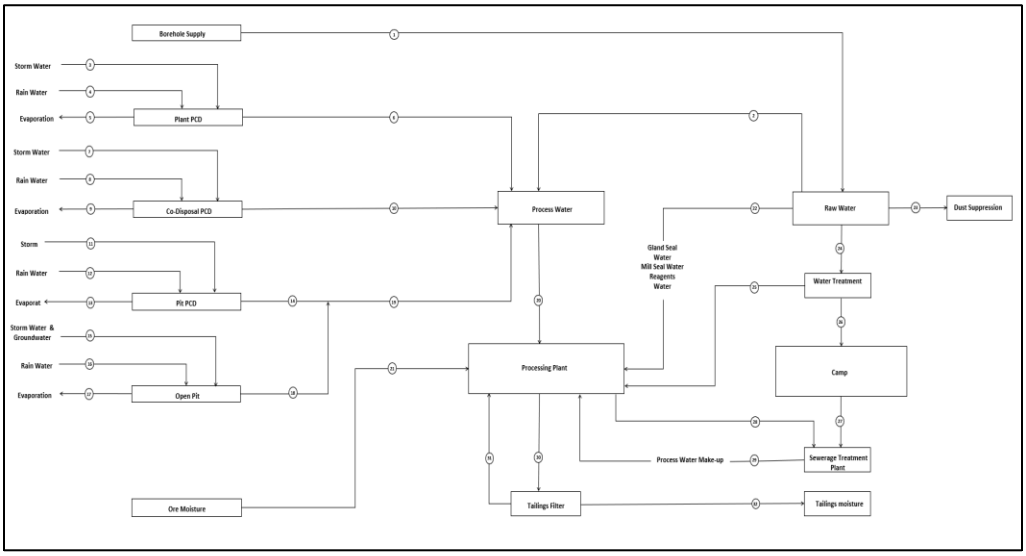


Figure 90: Overall Site Water Balance

The wellfield supplies make-up water via borehole pumps to the wellfield collection tank. The collection tank distributes the raw water to the process water tank, plant water tank and the water treatment plant. The water treatment plant distributed treated water to the camp and processing plant potable water storage tanks. The wellfield water to the processing plant supplies to the gland service water, reagents, mill seal water, process water tank and dust suppression.

18.4.1.1 Raw Water Supply (Wellfield)

The operation will require 180m^3/hour of raw water. The raw water will be sourced from a well field to be developed.

To optimise the mine water management a dynamic water balance model will be evaluated as part of the detailed design for the expansion. The re-use of dirty water in the mining processes will be maximised to reduce raw water make-up demand and well field abstraction.

The development and design of the well field will consider hydrogeological and social aspects of the surrounding area. The hydrogeological component will include borehole siting (geophysical), borehole drilling, aquifer testing, sustainable ground water abstraction modelling and permitting. Sustainable aquifer abstraction will be obtained by spreading the boreholes in an area with a ±10 km radius around the mine. The number of wellfield boreholes is conservatively estimated to be 22 with individual borehole yields expected to be between 8 and 27 m^3/hour as determined from Phase 1 hydrogeological drilling. Provision will, furthermore, be made for spare abstraction capacity to allow for potential maintenance losses.

A central wellfields tank which is positioned west of the plant will be used to store water from different borehole pumps. Two pumps situated at the central tank, will then pump water to the plant and camp.

An environmental impact assessment, as well as a possible relocation action plan will have to be undertaken for the wellfield, to assess the impact and ensure that the necessary permit can be obtained.

18.4.1.1.1 Integrated Water and Renewable Energy Model

The model objective is to ensure that the mine is able to receive 180m^3/h from the buffer storage reservoir.

The outcome of the borehole solar sizing model yielded a solar installation size of 20 kWp per borehole.

The overall installation recommendation is a 460 kWp combined solar plant, distributed to 22 borehole sites. (Refer Figure 91).

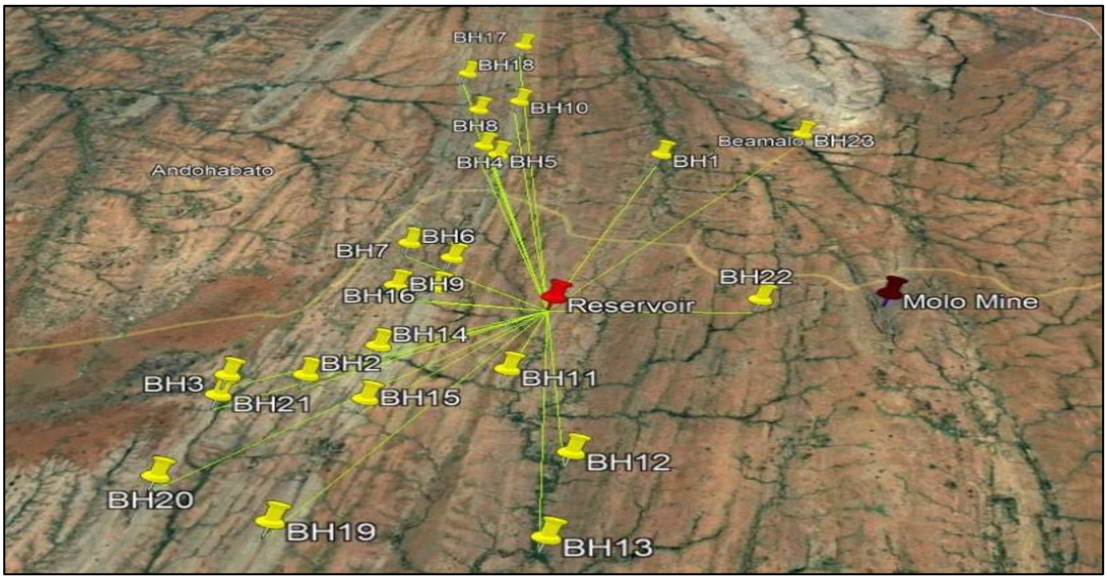


Figure 91: Distributed wellfields

18.4.1.1.2 Wellfield Power Supply

It is envisaged providing free standing 20 kWp off-grid Solar / Solar-Diesel hybrid VSD systems to power the various boreholes for process and potable water supply.

The process requirement is that a minimum of 3 of these boreholes pump 24 hours per day, 7 days a week. These boreholes will be powered by a hybrid Thermal and Solar system.

The following design parameters were applied to size the borehole power plant:

- Hourly Solar Yield as per supplied data and assuming a fixed East-West System with 10° tilt.
- Maximum design pumping rate of $27m^3/h$ over 7 solar hours for each borehole pump.
- Borehole pumps are 7.5 kW each.
- All boreholes are pumped to a centralized reservoir sized for a week's buffer.

18.4.1.1.3 Power Distribution

The power to the boreholes and water tank shall be distributed from a starter panel, installed on concrete bases by means of 600/1000-XLPE/PVC/SWA/PVC cables, buried, or placed on racks.

The borehole starter panel shall be rated at a minimum of 40A, 50 kA, 3-ph, 380 VAC and be of a Form 1B isolation and construction format.

The water tank utility starter panel shall be rated at a minimum of 300A, 50 kA, 3-ph, 380 VAC, and be of a Form 1B isolation and construction format.

Start and stop control to the boreholes shall be either done by hand, (manually), or hard-wired pressure / level and/or flow switch.

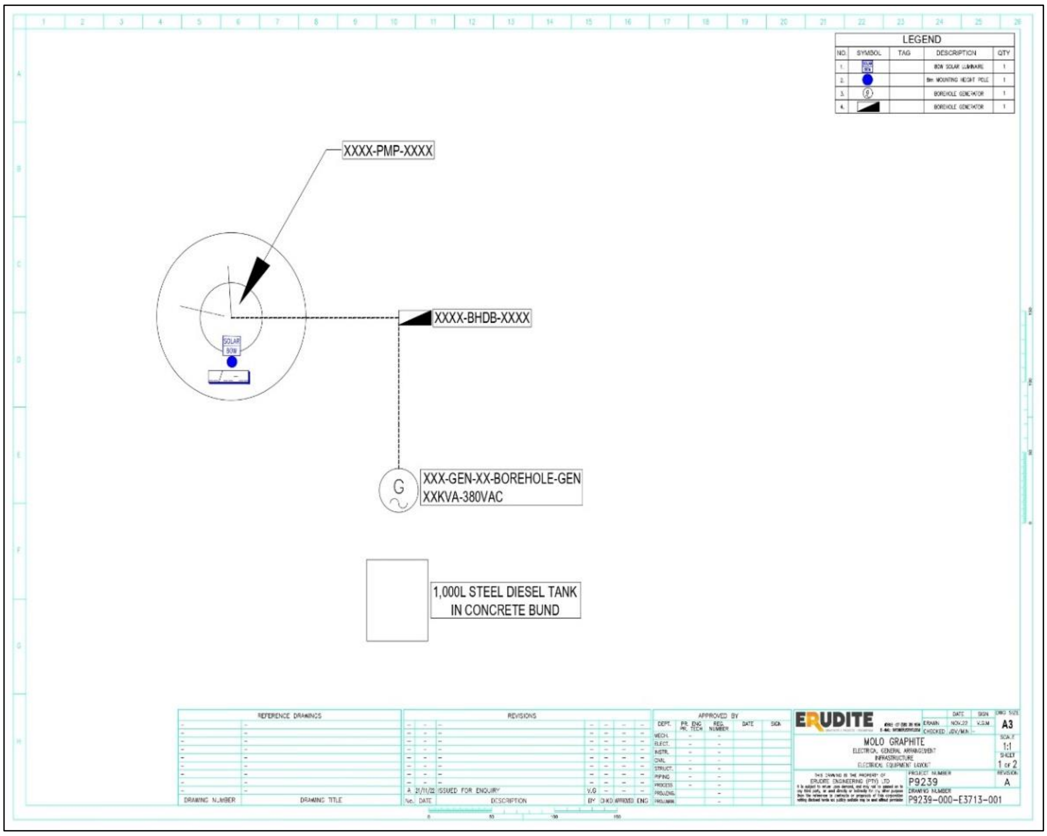


Figure 92: Boreholes - Power Distribution

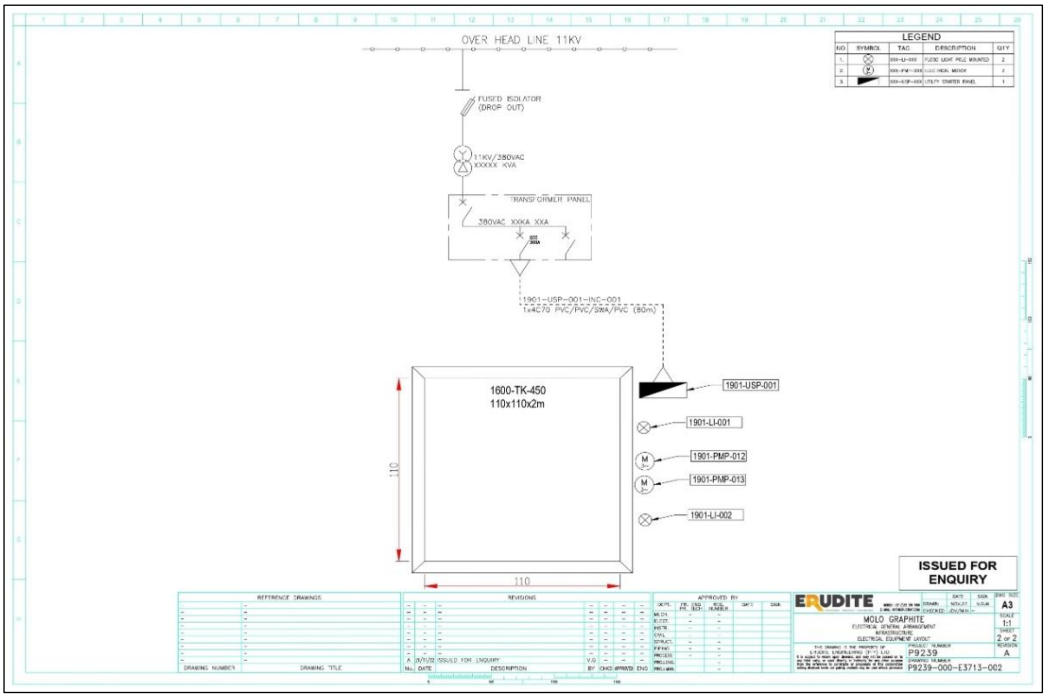


Figure 93: Water Tank - Power Distribution

18.4.1.1.4 Lighting

The Solar Powered Pole Mounted Luminaire shall provide lighting to borehole areas and be the LED type that is identified to be used alongside the roads.

SSLXPRO 80 solar street light with 210 W/18V mono crystalline solar panel, MPPT controller and LiFePO4 batteries

SOL/SP080 8 m mounting height surface mounted heavy duty HDG steel pole complete with SOL/BC/SOL/SP080 HDG bolt cage for above.

Luminaires and control gear shall be mounted for ease of removal, re-lamping, and servicing.

Power to the lighting shall be of the same type of luminaire, but be powered from the mains at 220 VAC.

18.4.1.1.5 Boreholes and Water Tanks Earthing

The earthing design of this system shall consist of bare copper conductors buried ± 500 mm below ground around the electrical equipment, earth electrodes, down conductors, and insulated copper conductor for bonding of equipment. All earth connections shall be thermite welded, brazed, clamped, or attached by solderless pressure type connectors, the latter type being most suitable for earth connections to electrical equipment. Where an earthing conductor is exposed to possible mechanical damage, it shall be protected by means of a suitable heavy duty PVC conduit, or pipe.

Refer to Figure 94 below.

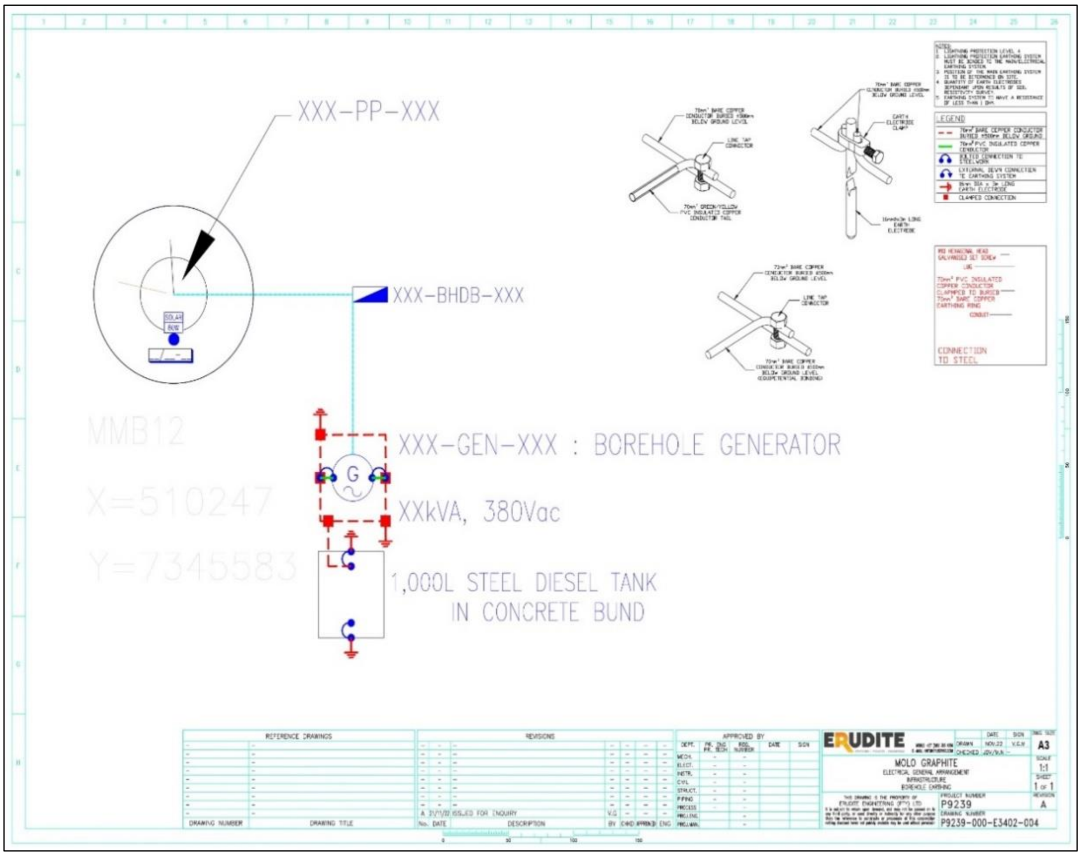


Figure 94: Boreholes and Water Tanks - Earthing

18.4.1.2 Storm Water Drainage

Storm water run-off will be diverted around the plant's mine terrace, the construction camp terrace, the open pit mine, ore stockpile areas, material reject dump and the co-disposal facility by a combination of unlined channels and berms which daylight downstream of the project infrastructure. Material excavated from the unlined channels will be re-used to construct the berms adjacent to the channels.

Storm water falling on the plant's mine terrace, open pit, stockpile areas, material reject dump and the TSF is considered dirty water and as such, is collected in the Return Water Dams (RWD), or Pollution Control Dams (PCDs) located on site and discharge to the environment is limited.

All PCDs on site have been sized to accommodate a minimum of a 1:10 year storm event for the plant PCD and, 1:100 year for the TSF PCD. Discharge from the PCDs will, however, occur and will need to be carefully controlled and monitored. This will be re-evaluated for the expansion by re-modelling the salt balance model for the LoM, based on new information from geochemical tests and modelling. PCD water under discharge conditions will comply at minimum with Madagascar legislation.

18.4.1.3 Potable Water Reticulation

Potable water for the Project will be supplied via the potable water treatment plant which is fed from the raw tanks at the accommodation camp site.

The potable water reticulation system will include a network of buried uPVC pipes installed at the permanent camp, processing plant and mine's complex. This network of pipes will be fed from the potable water supply storage tanks at the accommodation camp. Satellite storage tanks will be placed where required with local reticulation via pressure pumping.

18.4.1.4 Potable Water Treatment Plant

A provision has been included for the expansion potable water treatment plant installed during Phase 1. The Phase 1 water treatment plant was specifically selected as a modularised plant to ensure that it could be easily expanded should the need arise. The same approach was used for the expansion, to include a modularised plant for the treatment of raw water to portable water.

The augmented facility will be capable of generating an additional 165m^3/hr of potable water, which will be used in the plant, as well as for human consumption. The potable water storage will also be increased to 530m^3.

18.4.1.5 Waste Water Reticulation

The waste water reticulation will comprise a buried network of uPVC pipes, man holes and waste water treatment plants which will collect all waste water from showers, urinals, kitchens, mess, basins, sinks, etc. and gravity feed this waste water to the waste water treatment plant. The waste water generated at the mine site will be pumped to the accommodation camp for treatment.

18.4.1.6 Waste Water Treatment Plant

A provision has been included for the expansion of the waste water treatment plant installed during Phase 1. The Phase 1 waste water treatment plant was specifically selected as a modularised plant to ensure that it could be easily expanded should the need arise.

The waste water facility will remain at the accommodation camp, as it remains the biggest generator of waste water. Waste water generated at the plant will be pumped to the accommodation camp where the water will be treated.

The existing facility will be augmented by an additional 20m^3/day sewerage treatment plant. Drying beds have been included for the discharge and drying of the sludge from the units on the roofs of these units. These drying beds will be periodically cleaned.

18.4.1.7 Fire Water System, Reticulation and Storage

The fire water system for the plant, camp and mine area complex will be fed from the raw water dam located adjacent to the plant terrace. Sufficient capacity is included in the raw water tank to accommodate the fire water system requirements.

Pump suctions will be installed so that the fire water requirements are not reduced / utilized except for fire-fighting purposes.

Fire hydrants and hose reels are proposed to be installed at the plant area only, with fire extinguishers (9 kg), being utilised at all the buildings / offices and the mining complex.

The final layout of the fire reticulation system will be finalised once the preferred supplier is appointed during the execution phase of the Project.

18.4.2 Power

18.4.2.1 Electrical, Control and Instrumentation Design

Where International Electrotechnical Commission (IEC) Standards exist, they shall be used as appropriate, as a first preference. Where these are not applicable, or available, then SANS standards, (as applicable), or relevant national standards shall be used.

However, to allow flexibility in design and sourcing of equipment, standards may be re-considered by the Engineer, however, compliance with the latest revision of P9239-000-E510-001-A EC&I Design Criteria shall take preference.

18.4.2.1.1 Standards and Codes

The design and installation shall comply with all statutory regulations and will be subject to approval by the regulatory authorities, where appropriate. All equipment shall bear CE marking where applicable for the type.

Compliance with the latest amendments of the following codes and standards shall be considered a minimum requirement. In the event of differing requirements between codes and standards, the most precedence code, or standard shall apply as approved by the Engineer.

Standards to be followed shall include:

- Obligatory, International (SANS, IEC, ISO, and ISA), whereas SANS shall take preference if such standards are duplicated.
- Madagascar National Standards, where applicable.

18.4.2.1.2 Obligatory Standards and Regulations

Mine Health and Safety Act (Act 29 of 1996).

Occupational Health and Safety Act (Act 85 of 1993).

Minerals Act (Act 50 of 1991).

18.4.2.1.3 Project Electrical Specifications

The following standard specifications provide the necessary detail for equipment and expand on the contents of this Electrical Design Criteria.

Table 72: Project Electrical Specifications

Document Nr	Rev	Description
P9239-000-E510-001	A	Electrical, Control and Instrumentation Design Criteria
P9239-000-E520-001	A	Electrical, Control and Instrumentation Design Criteria for Vendor Packages
P9239-000-E520-002	A	11 kV Medium Voltage Switchgear Specification
P9239-000-E520-003	A	Distribution Transformers Standard Specification
P9239-000-E520-004	A	Testing And Commissioning of E&I Systems Specification
P9239-000-E520-005	A	Mini-Sub-station Specification
P9239-000-E520-006	A	Low Voltage Cable Specification
P9239-000-E520-007	-	Not Used
P9239-000-E520-008	A	Motor Control Centres Specification
P9239-000-E520-009	A	Low Voltage Variable Speed Drives Specification
P9239-000-E520-010	A	Containerised Sub-station Specification
P9239-000-E520-011	A	UPS - Engineering Specification
P9239-000-E520-012	A	Medium Voltage Cable Specification
P9239-000-E520-013	A	Lighting Luminaires Specification
P9239-000-E520-014	A	Earthing and Lightning Protection Specification
P9239-000-E520-015	A	Electrical Installation Specification
P9239-000-E520-016	-	Unused
P9239-000-E520-017	A	Battery Tripping Unit Specification
P9239-000-E520-018	-	Not Used
P9239-000-E520-019	-	Not Used
P9239-000-E520-020	A	Vendor Drawing and Data Requirement Standard

18.4.2.2 Power Demand

A bottom up estimating methodology was used to arrive at a Notified Maximum Demand (NMD), and electrical consumption (excluding standby loads), for the proposed installation. This was done by taking into consideration motor efficiencies, circuit types, actual load duties and utilisation factors.

Typically, the plant energy requirement is calculated in combination with the plant process philosophy, start-up procedure and what is presented in the mechanical equipment list.

The infrastructure loading includes power to the Water Purification Plant, Sewerage Plant, Stores, Laboratory, Change House and Plant General Lighting.

Table 73 below indicates the breakdown of the major values pertaining to the plant and infrastructure electrical loads.

Table 73: Power Demand Summary

Detail	Plant & Plant Infrastructure	Camp	Boreholes & Water Tank Pumps
Installed kW	14,041.84	615.00	315.00
Standby kW	1,996.23	0.00	75.00
Required kW	12,045.61	615.00	240.00

18.4.2.3 Power Generation

Power is provided to the Phase 1 operation through an independent power provider agreement with CrossBoundary Energy. CrossBoundary Energy use a mixture of solar and thermal generation to meet the Phase 1 demand.

The power generation solution is a hybrid power solution which consists of a combination of the following:

- Various prime rated diesel power plants.
- Fixed tilt east-west solar PV plant.
- BESS to match load profile and operating philosophy.
- Hybrid integrated energy control system
- Containerized control room at 11,000/380 VAC from where it is distributed to the various infrastructure loads such as the Plant, Camp and water-wells. This is done by means of 6.35/11-XLPE/PVC cables, as well as 11 kV overhead lines.

Figure 95 below refers to the typical layout as indicated below.

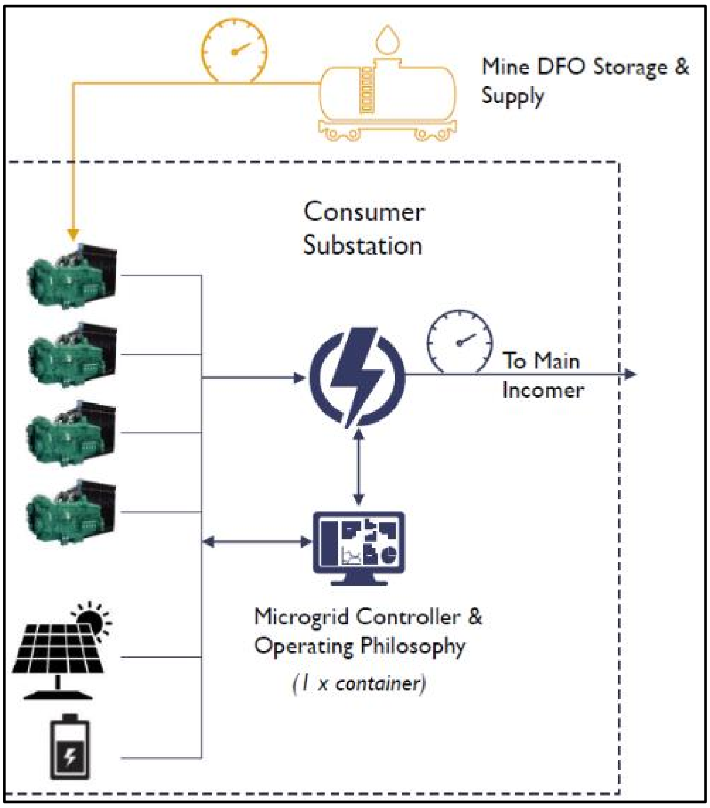


Figure 95: Typical Alternative Power Arrangement

With this solution, the generation demands are met through a combination of sources. When starting larger loads, such as the Mill, the thermal generators will run in parallel to match the startup load. Once the plant has reached equilibrium the thermal generators will be stepped back. During continuous normal operation of the plant, power shall be provided as a combination of Solar, BESS and Thermal.

This power generation solution will be expanded to match the increased power demand for full production.

The bulk power supply is rated at 13.35 MW at 380 VAC and 11 kV.

This includes the bulk power supply to the camp of 0.615 MW at 11 kV/380 VAC, as well as the bulk power supply to the water tank pump station of 0.075 MW at 11 kV/380 VAC.

Off-grid borehole water pumping supply is not included in the above, as these pumps will be powered by stand-alone solar power plants, or a combination of solar and diesel generators.

18.4.2.3.1 Generation Modeling

The expansion load list and operating parameters were used to provide an estimated load profile with average load of 10.5 MW and a max load of 13.2 MWe. A stable load profile of 75% (assumed) of installed loads, with 12 hours and 24 hours maintenance downtime every alternative 14 days was considered. Solcast data was used as the solar resource in PVSyst simulation of the yield.

Average load profiles for accommodation, ablution and workshops, as well as the operation of the well-fields for the expanded mine capacity, were estimated based on experience on similar projects and information for Phase 1.

The water tank booster pumps will be supplied from the consumer sub-station via an 11 kV overhead line and will be operated 24/7.

The thermal plant operating philosophy, combined with the processing plant's expected power quality and controls, will require generator minimum load factor of 40% and a minimum of 2 generators in operation.

- The mine will consume at least 92,200 MWh per annum .
- Ground conditions assumed to be suitable for PV installation, to be confirmed following geotechnical investigation.
- Connection point assumed to be capable of evacuating all the power from the hybrid power facility.
- The terrain is assumed to be relatively flat and accessible.

18.4.2.3.2 Thermal Power Generation

The thermal capacity shall be sized for 14 MWe firm capacity, which includes the following hardware:

- Phase 1: 6 x Scania DC16-93A (Existing)
- Expansion Project: 8 x MTU HTW2200T5 (New).

The above shall relate to 18.16 MWe total installed capacity, (Prime Rated Power), at an N+2 redundancy which guarantees availability in the event of an unexpected engine failure, (of any engine type), and major engine maintenance / overhaul.

It is considered that with two of the larger engines out of service, the facility will be capable of supplying the full site demand of 13.25 MWe.

Refer to Table 74 below.

Table 74: Thermal Operations Scenarios

Scenario	Operational Units	Total Firm Capacity
All engines available (Normal Operation)	6 x 518 kW Scania 8 x 1,881 kW MTU	18.156 MWe
1 x Engine unavailable	6 x 518 kW Scania 7 x 1,881 kW MTU	16.275 MWe
2 x Engines unavailable	6 x 518 kW Scania 6 x 1,881 kW MTU	14.394 MWe

The expansion Thermal Power Plant will be located at a centralized location, reticulating power to loads via an MV distribution network.

Typically, the configuration will be the following:

- The Expansion Project Thermal Plant will comprise:
  - 8 x HTW2200T5 11 kV 50 Hz Diesel Generators.
  - 4 x 10,000 litres Double Walled Diesel Day Tanks.
  - 1 x 11 kV NECRT.
  - 1 x 11 kV 40 ft Containerised Sub-station.
  - 2 x 40 ft Workshop and Spares Containers.
  - 1 x 20 ft Site Office Control Room.
  - To accommodate the preceding, additional land is required for the installation of the expanded Thermal Plant. (Typically, 50m x 70m).

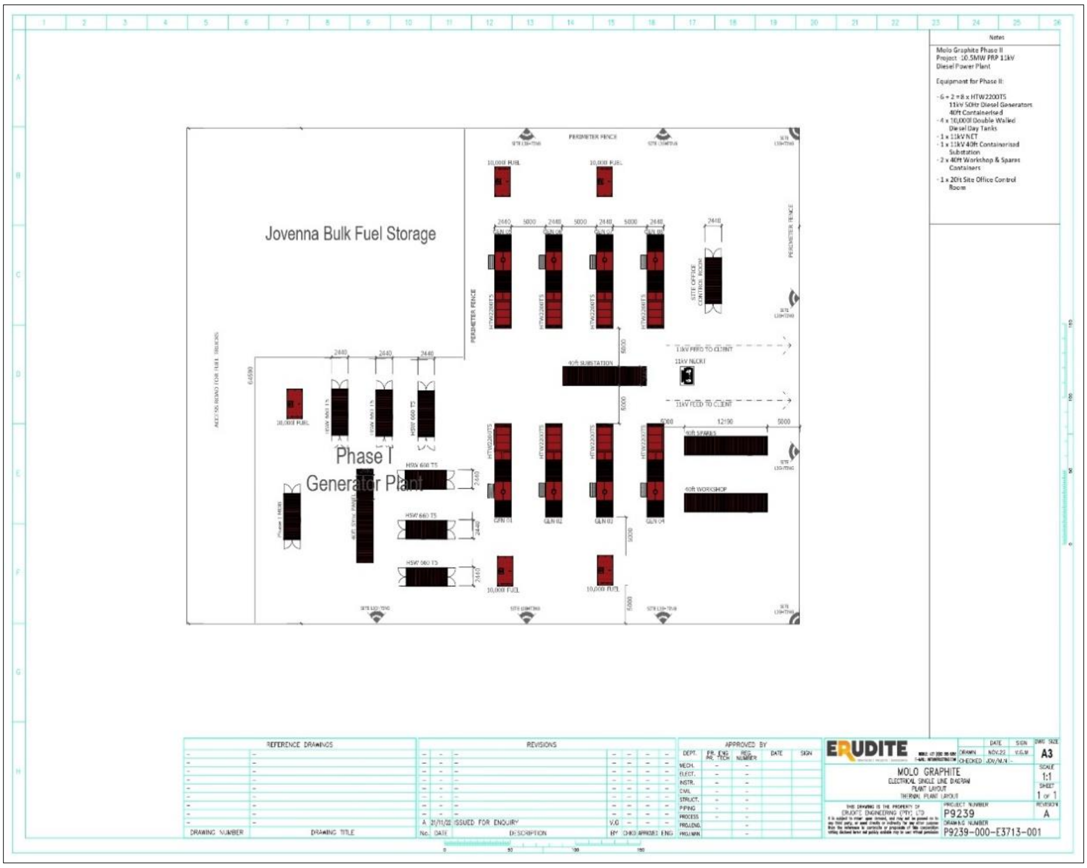


Figure 96: Envisaged Thermal Plant Layout

18.4.2.3.3 Solar Photovoltaic (PV) and Battery Energy Storage System (BESS)

Solar Photovoltaic (PV) systems convert energy from solar irradiation to electrical energy. This power source is considered a renewable energy resource. The 550 Wp, P type Mono-crystalline, Mono-Facial PV modules were selected for the basis of the design for the FS, totalling 18 MWp DC. Market rates are a key driver on the technology selection.

PV modules are considered a commodity, and their prices can fluctuate rapidly as supply and demand vary. The environmental conditions are also considered to ensure the chosen technology will be able to function efficiently in the specific site conditions, such as high temperatures and humidity.

The selected string inverters have a nominal capacity of 350 kVA at 30°C, and a total derated plant capacity of 14.7 MW AC at 40°C. A BESS is used to store electrical energy and can benefit the power system in many ways.

The BESS technology is very well suited for:

- Renewable energy shifting.
- Flexible ramping.
- Transmission and Distribution referral.
- Behind the meter power management for Time-of-use energy tariff arbitrage and maximum demand reduction.

It is envisaged that the 18 MWp solar PV and 12 MVA BESS solution will result in ±30% renewable energy contribution to the Expansion of Molo Graphite. However, this requires further optimization based on a detailed load analysis and dynamic systems studies.

As solar is the most economical source of power, it shall be maximised.

Technical restrictions considered in configuring the solar PV facility are as follows:

- The solar PV operates in isolation, is not grid forming and necessitates synchronizing with another grid forming device, (e.g., grid, generators, or batteries).
- The solar PV power is not dispatchable and is only produced when the sun is shining.
- The generators will have to run at a minimum load factor requiring control and potential limitation of solar PV output.
- In addition to the 2.7 MWp Solar PV and 1.5 MVA/1.37 MWh BESS of Phase 1, the expansion of the Renewable Energy Facility will install a further 15.3 MWp of Solar PV and 10.5 MVA/9.7 MWh BESS.
- The total Renewable Energy Facility for the Expansion Project will consist of 18 MWp Solar PV and 12 MVA/11 MWh BESS.
- Integrating the BESS is required to stabilize the fluctuations of PV power and expand the solar PV facility to 18 MWp, offsetting more fuel with cheaper solar power.

The expansion Solar Power Plant will be located north of the Phase 1 plant, which shall include multiple string inverter stations and step-up transformers. The transformers shall be used to step up the 0.8 kV to 11 kV and be combined at the MV collector sub-station. Depending on the results of the geotechnical investigation, as well as land use restrictions imposed by the presence of sensitive habitats, the expansion could be divided into 2 distinct locations.

Additional land is required, to cover an additional 15 Ha (500 metre x 300 metre) of land to the north of the Phase 1 Solar PV Facility refer Figure 97 below.

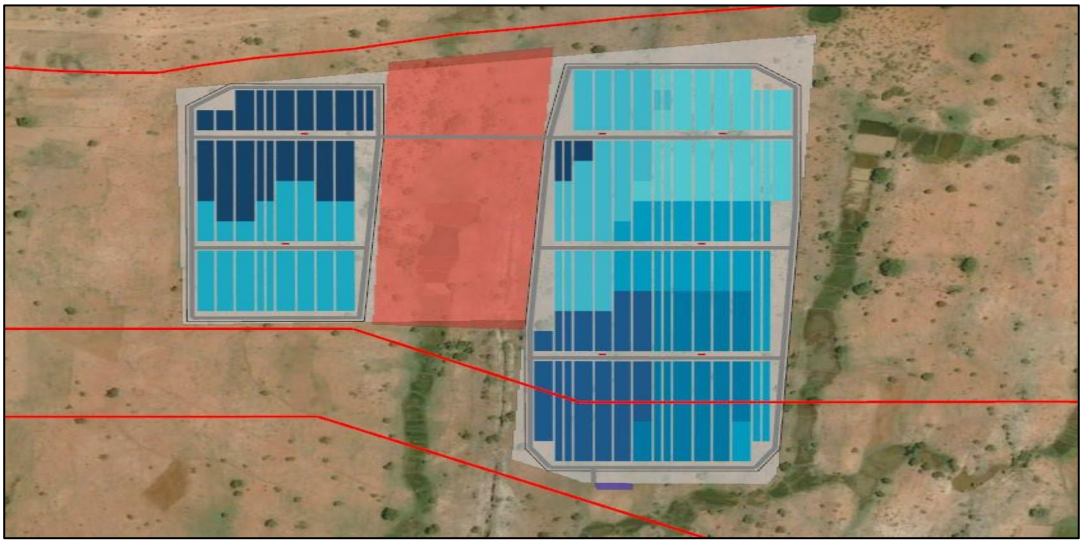


Figure 97: Envisaged PV and BESS Plant Layout

18.4.2.4 Consumer / Incomer Sub-station

The Consumer sub-station will be a weather proof prefabricated modular unit, installed on a concrete bases and plinths, with steel access stairways and landings. Access will be via the ends of the sub-station with one side having double door openings and pedestrian access on the other side. All guard railing to the landing on the access side will be removable to allow for equipment to be installed and removed as required throughout the duration of the mine's life.

Power distribution via this 11 kV sub-station, rated at 1,250A, 31.5kA to the respective loads. This consumer sub-station shall be the Point of Common Coupling (PCC) for the thermal, solar and battery energy storage system.

The 2 x 1.4 MW, 3-ph, 380 VAC diesel generators shall be controlled and connected to this sub-station, rated at 4,500 A, 35 kA either being stand-alone, or part of the generator panels.

From this Consumer sub-station, the power shall be distributed at 380 VAC, via copper cables, to the 6 respective plants 380 VAC-MCCs, as well as 2 x 11 kV Sub-stations located strategically in the plant, from where the plant loads and some infrastructure shall be supplied.

18.4.2.5 Plant Sub-stations

The plant sub-stations shall be powered form the Consumer Sub-station, via an 11 kV distribution network utilising a combination of ring main units using 6.35/11-XLPE/PVC cables and overhead lines, as well as 11,000/380VAC stepdown transformers. These plant sub-stations shall be of Form 3b construction separation and isolation format rated at 50 kA, 3-ph, 380 VAC.

The plant sub-stations shall be prefabricated and installed on concrete bases and plinths with steel access stairways and landings. Access will be via the ends of the sub-station with one side having a service door and pedestrian access on the other side.

All guard railing to the landing on the access side will be removable to allow for equipment to be installed and removed as required throughout the duration of the mine's life.

It is envisaged that all MCCs will be placed strategically to be closest to the loading, reducing cable runs and installation costs.

18.4.2.6 Transformers

As the generators excite at 380 VAC + N + E, no distribution transformers are required on this plant, other than that required by control and instrumentation to reticulate. It shall be noted that the power generation provider shall require multiple 11kV / 380 VAC transformers.

18.4.2.7 Earthing and Lightning Protection

The earthing and lightning protection includes the plant, infrastructure, and camp areas. The earthing system design shall provide an equipotential system with all equipment being effectively at a single earth potential. This shall be achieved by tying together all earthing systems including structures, electrical equipment and instrumentation earth systems. The objectives of earthing design are as follows:

- To provide a current return path to the electrical power source in the event of a phase to earth fault.
- To provide an equipotential between conductive metallic structures in the event of a lightning strike.
- To provide low step and touch potentials in the event of phase to earth faults.
- To provide sufficient electric charge dissipation in the event of a lightning strike, static electricity, and stray currents.
- To minimize the effect of induced voltages in the event of a lightning strike, static electricity, and stray currents.
- The Earthing and Lightning Protection installation shall be in accordance with SANS Codes of Practice 10199.(2016) and 10313 (2018) in conjunction with SANS 62305-1-2-3-4: 2011, and SANS 62561 parts 1-7.

18.4.3 Access Roads

The access road that was developed as part of Phase 1 will require no further upgrading of the access road. The road will require more frequent maintenance in line with the increased usage.

18.4.4 Terraces and Bulk Earthworks

The main terraces for the works include:

- Permanent Camp.
- Plant.

Terraces will be constructed as part of the works and will be installed at a minimum of 300 mm above natural ground level to reduce the risk of flooding in the rainy season.

18.4.5 Gate House and Turnstile Access Control

No additional access control is envisaged for the plant. However, additional access control will be introduced between the change house and the process plant.

18.4.6 Accommodation Camp

The accommodation camp will be augmented with an additional 9 units similar to the units supplied for Phase 1. Each unit will be capable of housing 24 people, with a central ablution facility for each unit. These units will be serviced by a central kitchen complex consisting of a kitchen, mess outdoor entertainment area as well as a laundry in a similar configuration as the current facility.

In addition, separate accommodation and ablution facilities for 32 security personnel has been allowed for.

18.4.6.1 Power Distribution

The power supply to the camp infrastructure MCC shall be supplied by the consumer sub-station. This shall be done at 11 kV on an overhead line from where it is stepped down to 380 VAC via 2 x 315 kVA transformer / mini-subs. (Refer Figure 98 below).

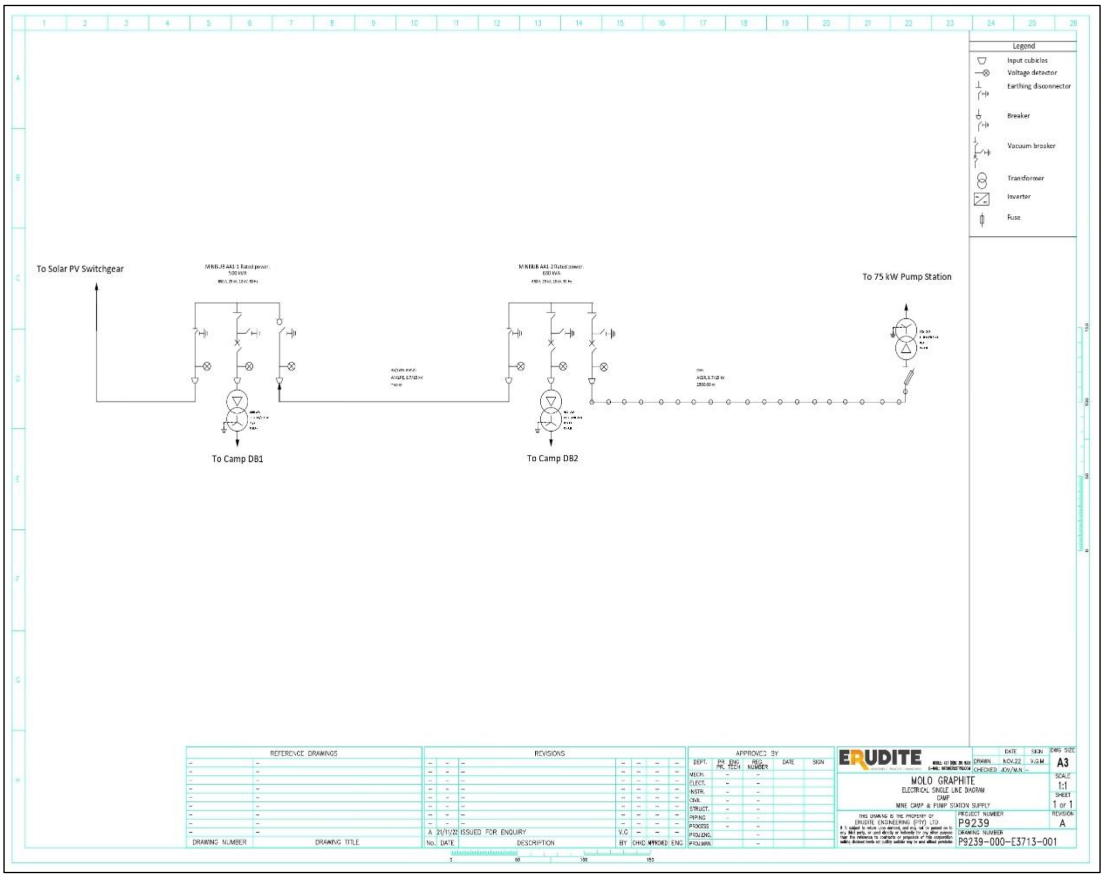


Figure 98: Camp Area - Main Power Reticulation

The camp power shall be distributed from a main containerised sub-station, installed on a concrete base and plinths with steel access stairways and landings. Access will be via the ends of the sub-station with one side having a service door and pedestrian access on the other side.

All guard railings to the landing on the access side will be removable to allow for equipment to be installed and removed, as required, throughout the duration of the mine's life.

The camp infrastructure MCC shall be rated at minimum of 1000 A, 50 kA, 3-ph, 380 VAC and be of a Form 3B isolation and construction format.

It is envisaged that the camp infrastructure MCC shall provide power to the respective DB's and starters of the respective water and sewerage treatment plants. Distribution to the respective loads shall be by buried 600/1000-XLPE/PVC/SWA/PVC copper power cables.

These distribution boards and starter panels are placed strategically to be closest to the loading, reducing cable runs and installation costs.

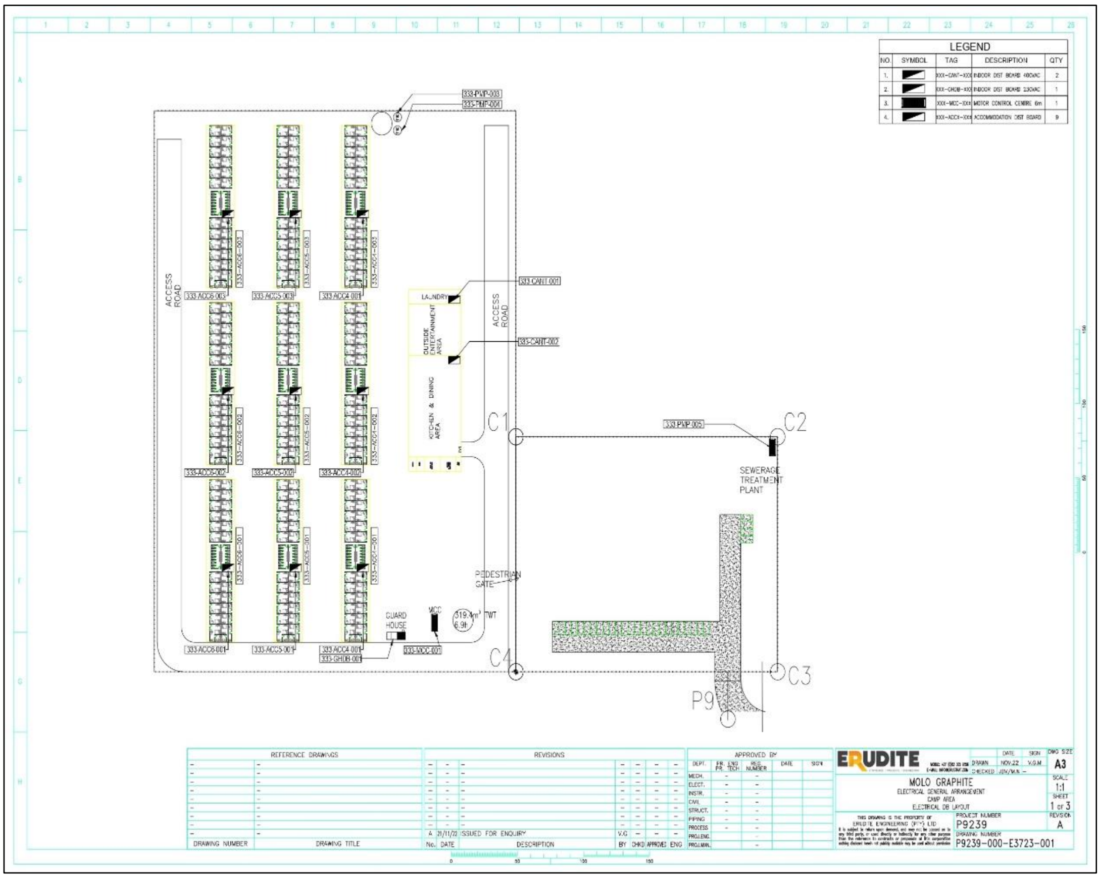


Figure 99: Camp Area - Power Distribution

18.4.6.2 Lighting

The high mast pole mounted luminaire, (powered from the mains), shall provide general lighting to the camp area and be of the LED type, HP30 250W Flood. These lights shall be mounted in groups of 2 on 12m high scissor type masts.

Luminaires and control gear shall be mounted for ease of removal, re-lamping, and servicing. Refer Figure 100 below.

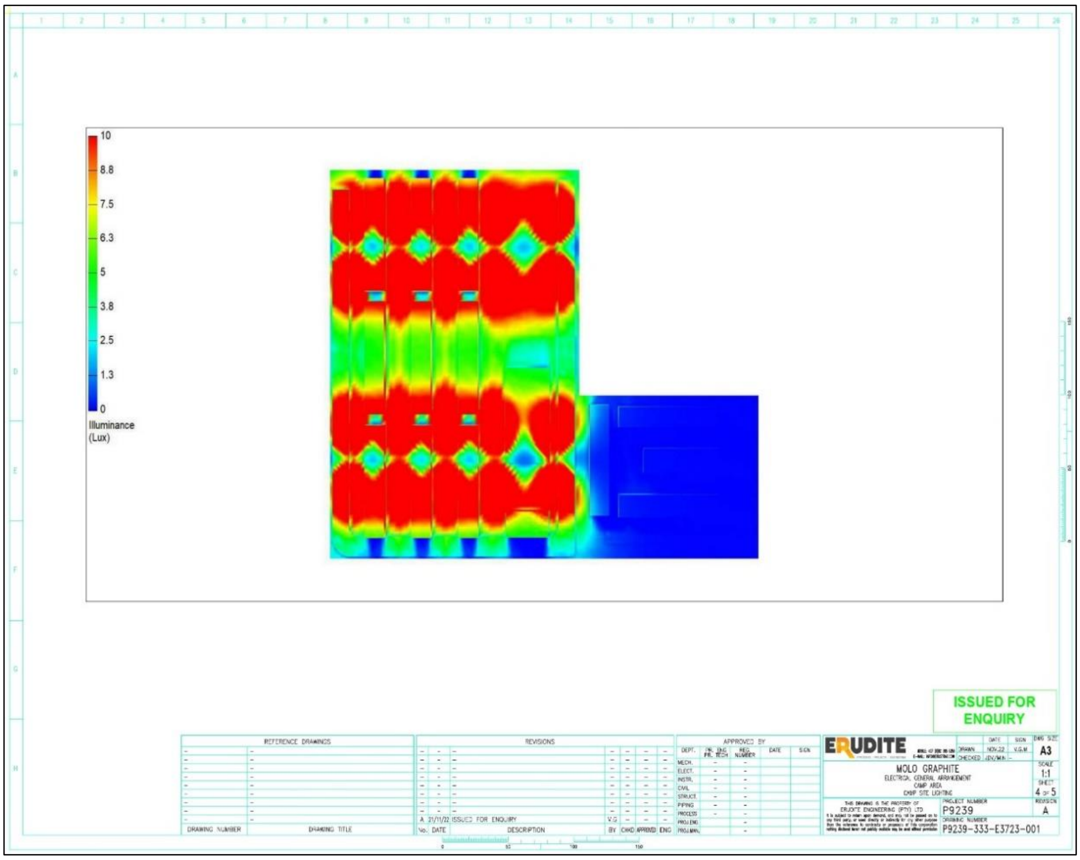


Figure 100: Camp Area - High Mast Lighting

18.4.6.3 *Camp Earthing*

The earthing design is based on a containerised accommodation constructed of metal clad roofs and side walls which act as a natural down conductor, with the lightning protection connected to the metallic structure at the base and then to the earth rods, including the fence around the camp. The lightning protection system shall comprise lightning earthing terminals, roof conductors, down conductors, test links and earth electrodes. Each down conductor shall have a separate earth electrode. (Refer Figure 101 below).

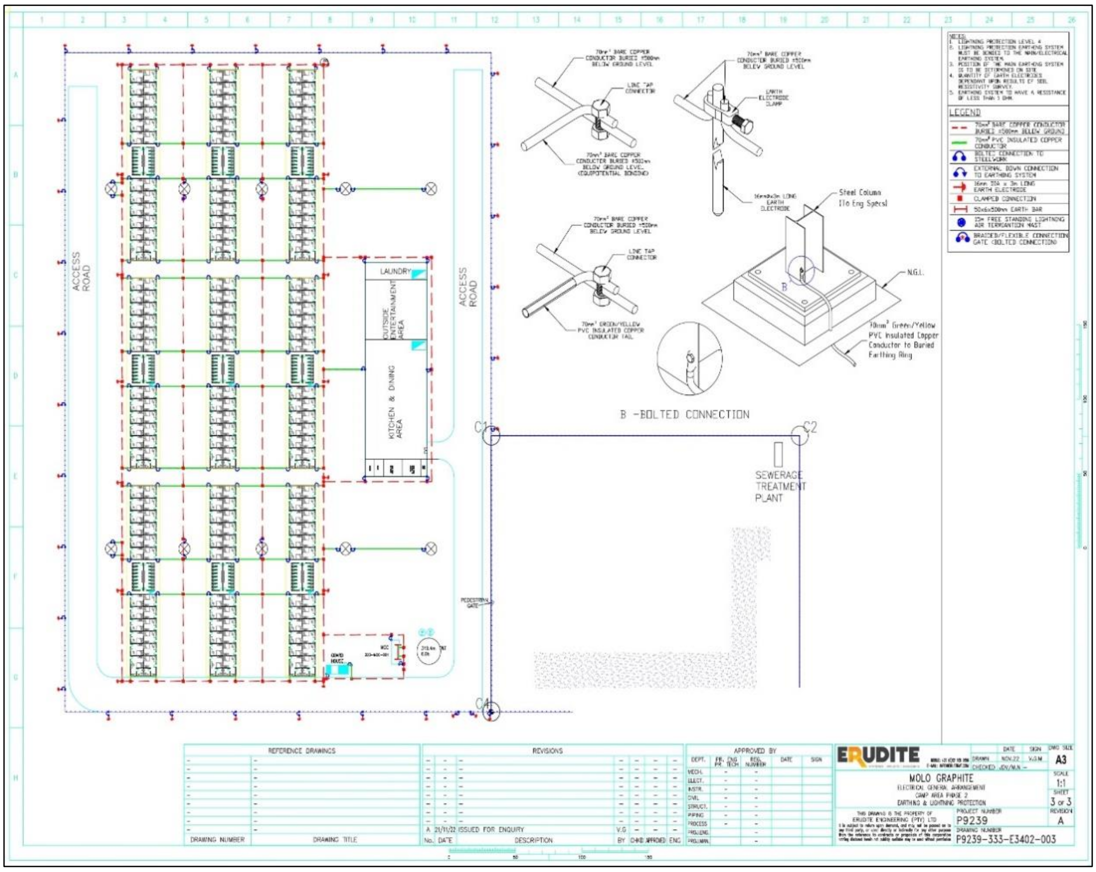


Figure 101: Camp Area - Earthing

18.4.7 Fencing

The plant and mine complex, camp and construction camp will be fenced. The security fence is intended to keep both unauthorised people and local livestock out of the areas detailed above.

The security at the plant and mine complex, the permanent and construction camp areas will be 2.4m high diamond mesh security fencing with 3 strands of galvanised barbed wire to the top of the fence and 6m wide access gates, together with pedestrian access gates which will be located at all the above-mentioned areas.

Access control by means of booms, turnstiles and security check points will be installed at the various locations as required.

18.4.8 Spares Storage

A spares storage area has been provided for, which will support both the mining operation and the process plant operations. The storage area includes 2 shipping containers which will be refurbished and used as offices and storage.

18.4.9 Tyre Storage Yard

An open area has been identified on the plant terrace for the storage of spare tyres for the plant and mine LDV fleet and the mining fleet. A provision is made for security fencing at this location.

18.4.10 General Storage Yard

An open area has been identified on the plant terrace for a general storage yard which will be utilised to store oversize equipment and spares for the plant and mine operations that are not stored at the strategic spares storage area. A provision is made for security fencing at this location.

18.4.11 Bulk Fuel Storage Facility

During the operational phase of the project, the bulk fuel storage system will consist of 5 containerised, double-walled fuel storage tanks, with sufficient storage capacity for 2 weeks operation, (approx. 300,000 litres. Each storage tank has 60,000 litres capacity. The advantage of such a system is that very little is required in terms of civil and structural work to construct such a system, which in turn reduces capital cost.

During the construction phase of the Project, the fuel storage on site will have the capacity to store two weeks supply, (approx. 120,000 L), in 2 containerised, double-walled fuel storage tanks. The tanks required at operation phase will be procured as part of the early works and mobilization of the project and will be utilized during the construction phase of the Project.

18.4.12 Re-fuelling Station

The re-fuelling station will be centralised on site and will be utilised by the plant and mine LDV fleet, the haul truck fleet, and the mining haul truck fleet. The re-fuelling station is located on the plant terrace central to the site.

The re-fuelling station will have a direct feed from the diesel storage facility on site and no provision has been included for any storage of diesel at the re-fuelling station.

18.4.13 Information and Communication Technology

18.4.13.1 Physical Network Connectivity

Currently a fibre optic ring network is in place connecting the plant with the office buildings located on site. An expansion of this will be required to service new buildings that are due to be deployed onsite in the office and plant area to facilitate the extra space required for operations. Additionally, a dual redundant run should be considered so that if there are multiple breaks in the fibre the connectivity will continue to work. Core services which are segregated using VLAN technology run across this network and allow Corporate, Operational, CCTV and other such services to run across the site.

Quality of Service measures will be fine-tuned to ensure that the right traffic is being prioritised and enables the network to function efficiently. The camp location is located some distance from the office and plant area and is currently connected via a wireless point to point link, however, fibre should be installed to provide a more stable connection as well as support the higher bandwidth requirements that additional staff will require as part of Expansion Project. The core network requires an upgrade from 1 GbE connectivity to 10 GbE connectivity to support future requirements.

Environmental Monitoring and Control.

Currently basic environmental monitoring is in place for temperate in the server room. However, the plant control room has no temperature control nor monitoring of it. Monitoring devices should be installed in the server room and plant control room, to ensure that temperature controls are working efficiently and to detect when cooling systems breakdown, but also to monitor that the humidity level is sufficient.

18.4.13.2 Systems Servers and Software

Due to the nature of the remote location and high amount of data that will be generated, leveraging cloud technologies is a challenge that cannot currently be solved. Sufficient levels of local compute, storage and networking need to be in place to support daily operation requirements, which includes the need to run local application servers for control of local equipment on site. All core server, network and storage infrastructure will be housed in a dedicated server room and replicated nodes will be in place off-site such as Mauritius and/or Antananarivo. Consideration should be made for utilising the core server infrastructure room for secondary plant (OT) systems and backup nodes, as well as potentially a 2nd server room that is physically separated within the overall site.

18.4.13.3 Telecommunications

Mobile services are currently weak on site across the mine, plant, offices and camp and once the density of personnel onsite increases the current ability to utilise what is available may prove challenging. There should be consideration around installation of a mobile services tower and engagement with local telcos to allow installation of repeater equipment to provide either better coverage for the current strongest provider (Orange), or to enable additional / multiple providers to supply mobile services to site which will also enable additional options for data services provision.

Wireless network connectivity across site enables the use of Voice over IP (VOIP) services for outbound / inbound calls from the site with Microsoft Teams continuing to be the preferred communication method within the company across all global locations.

18.4.13.3.1 Support Services

To support 24 x 7 plant operations, as well as the increased footprint of users on site, additional Technical Support staff will be required. Due to the remote location of the site, staff should be upskilled with basic knowledge regarding how to troubleshoot and fix any SCADA issues that may arise and have a more intimate knowledge of how operational control systems work both at a functional level, but also at an architectural level which includes the ability to backup, recover and restore key systems in cases where there is no redundancy, or resiliency built in, or in case of equipment failure of single nodes.

18.5 Product Transport

As briefly discussed in Section 5.1, Access; the main route for transportation of product is to the port of Tulear. Tulear is viewed as the primary option on the basis of its proximity to Molo, more economical transport cost, port and warehousing costs, regional political requirements, etc.

Tulear port was selected on the following points:

- Closer by 100 km to the Molo mine site than to any other Madagascan port.
- Lowest warehousing and port handling cost.
- Lowest road transport costs.
- Situated in the same province as Molo mine (Tulear Province), which is important for political reasons, (flow of material maintained within Tulear Province, ease of issuance of road transport permits, use of government port infrastructure rather than a privately operated port, taxes for export and logistics are held in the same province.
- Due to the location of the Molo Graphite deposit and the general infrastructure in Madagascar, product can only be transported in bulk bags with hopper trucks to the nearest ports for exportation. Options for rail, based on the existing infrastructure within Madagascar are not suitable for the project.

The existing road from Molo to Tulear Port has been effectively used for transportation of product and goods during the commissioning and ramp up of production and proved to be viable for Phase I volumes of cargo movement.

Aside from the existing all-road route from Molo mine site to Tulear Port, (which has limitations due to bridge restrictions and periodic road surface degradation, due to inclement weather and traffic volumes), there is, for the expansion project, an option to transport the bagged product via truck along the south bank of the Onilahy River, to a loading point near Soalara.

This should also minimise the impact of the rainy season on movements, increasing the route pass ability per annum, as well as permitting increased truck payloads.

This option is still under assessment, and a study should be finalised end of March 2024. Nonetheless, it is possible that, based on the first approximate indications, deployment of a landing craft should not impact the overall transport price and, this development has the support of the Governor of Tulear Province as well as the Madagascan Government.

Note: Whilst the port of Fort Dauphin has been evaluated as a port of despatch from Madagascar for graphite, it will remain, for the time being, a second option.

18.5.1 Logistics Management

A logistics company, AsstrA, currently managing Phase 1 of the Molo project, has been selected to manage the global logistics requirements of the expansion project.

With the support of the strategic partnership of COMATO, (Tulear port operator and stevedore), and AsstrA's management team, (under the guidance of NextSource logistic director), all parties will work closely with site operations and management, as well as with sub-contractors, partners, and customers to deliver product in a safe and timely manner. The management team will oversee the in-country trucking, warehousing and loading, as well as export. The logistic team's preferred port of export is the Port of Tulear on the west coast of Madagascar. When required, or requested, ASSTRA will handle the international ocean freight movement.

18.5.2 Bagging

Graphite requires appropriate bagging for ease of transportation via road and shipping. This is necessary in order to avoid the loss of product in transit that could breach environmental laws, ease transshipment between road and ocean movement, comply with port facility and ocean carrier regulations.

The product is transported in 1,000 kg bulk bags and will be loaded onto hopper trucks at the Molo processing plant. Following the commissioning of the expansion, the combined concentrate production to be transported is 150 ktpa. This equates to an estimated total of 150,000 bags of graphite per annum. (Basis 1,000 kgs Per Bags)

A product store, with the capacity of holding approximately 2,400t product production, is included in the expansion project to allow for the surge capacity during truck loading, as well as to accommodate for rainy season delays when road conditions become impassable.

18.5.3 All-Road Transport

In the Lebedev Consulting report, (Dynamic Simulation of Molo Phase 2 Graphite Project, 2023) a transportation simulation was done from the site to the Port d'Ehoala. Since the simulation study, the alternative port at Tuliara was reviewed and selected for the project. The simulation parameters shown Table 75 below to the port at Fort Dauphine suggested a daily convoy of 23 trucks and trailers to transport the 152,970 tpa calculated estimate at the time of study.

Table 75: Product Convoy Parameters (Simulation Model)

Description	UoM	Value	Note
Product Target	tpa	152,970	Simulation
Graphite per bulk bag	kg	750	Rounded calc 25,000 kg/33 bags
Bulk bag trailer 1	-	18	Rogers Shipping
Bulk bag trailer 2	-	15	Rogers Shipping
Bulk bags per truck	-	33	Calculation
Graphite pay load	t	24.75	Calculation
Trucks per convoy	-	23	Calculation
Product per convoy	t	569.28	Calculation
Convoys per annum	-	269	Calculation

During the establishment and commissioning of Phase 1 the road from Molo site to Tulear Port was identified as being the preferred export route. The truck payload restriction for movement from Molo to Tulear using the all-road route via the Tongabory Bridge for Phase 1 is 18,000 kgs (18 Bags) as shown in Figure 102.

The total distance from the mine to the port on this route is approximately 300 kms one way. The total trucking distance as per AsstrA UK Limited for the contract is 600 kms round trip.

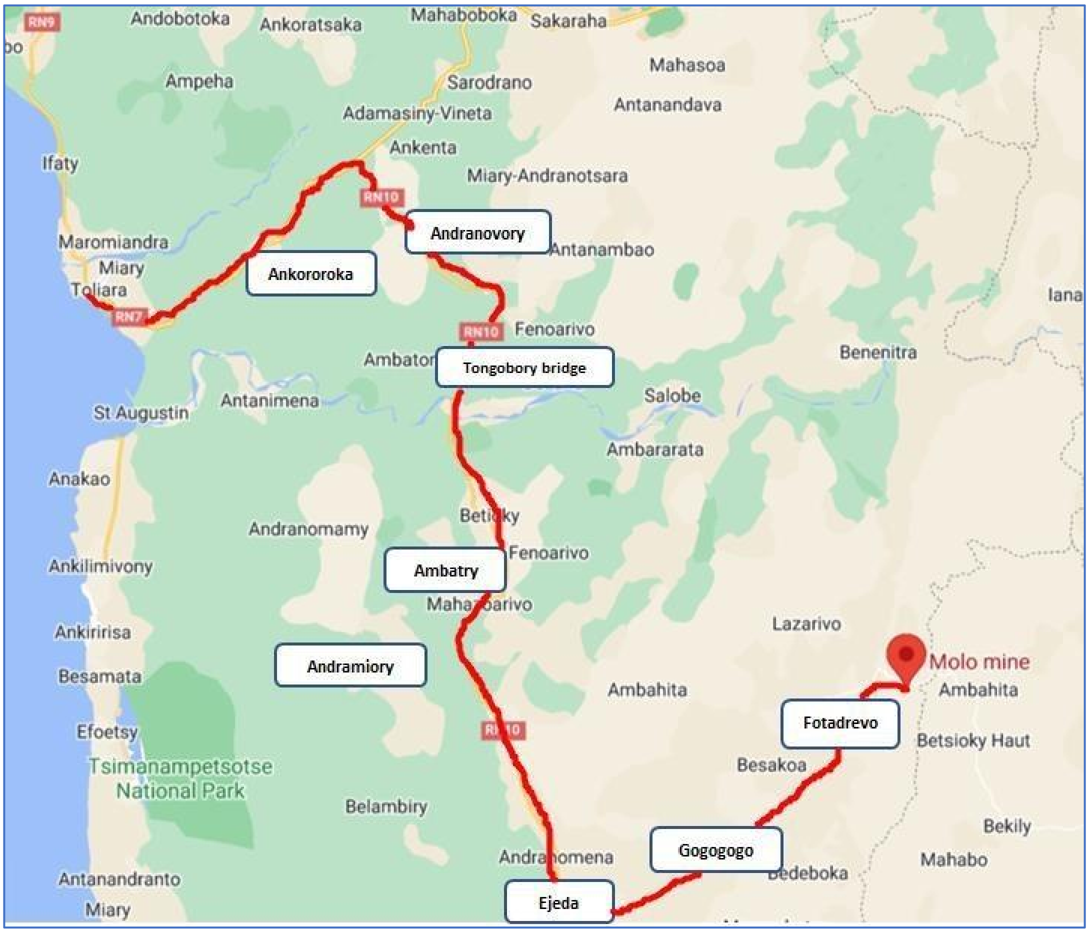


Figure 102: Current transport route from site to port

18.5.4 Alternative Transport (Landing Craft – Soalara to Tulear)

To eliminate the bridge restrictions at Tongabory the option to barge loaded trucks from Soalara to Tulear Port was identified. This will increase the loading capacity of trucks and reduce the total trucking distance. Map below of the alternative route with landing craft on barge.

Figure 103: Alternative barging route

The operation of the landing craft for movement of laden trucks from Soalara to Tulear, will be operated by a Madagascan majority owned private company. The operator will employ Steyr 6 x 6 hopper trucks for transportation of the product. The Madagascan consortium will invest in a fleet of new Steyr 28t capacity 6 x 6 hopper trucks of sufficient quantity to meet the movement requirements for the expansion project. Illustration of trucks in Photo 8 below.

An additional advantage, trucks delivering graphite to Tulear can collect diesel oil using pillow tanks specially designed to fit the Steyr trailers when full and capable of folding flat and moving with the graphite laden trucks.

This would have the effect of reducing overall road transport costs by avoiding the use of separate tankers for the delivery of fuel.

Photo 8 below.

Photo 8: Photo of typical Steyr Truck 306

The barging operation is envisaged to be implemented for the expansion project or, even before during Phase 1 of the project, hence, carrying a maximum load of 28t (28 bulk bags), and will haul the bags of graphite from the mine to Tulear port via the Soalara route. This route, at 210 kms (to Soalara), is significantly shorter than any other route, thus yielding lower fuel costs and shorter on road transit times. From Soalara the trucks will be moved on the landing craft as shown in Figure 104 below.

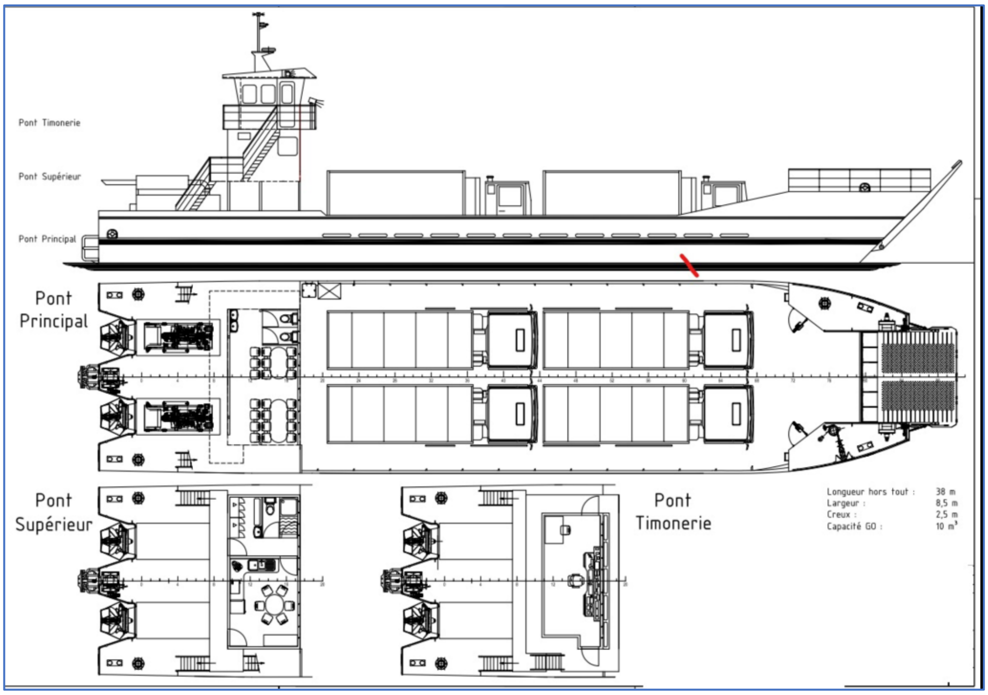


Figure 104: Barge with loaded trucks

By applying the same principles as per the simulation study Table 76 was created for the transportation of product from the site to Tulear port via Soalara.

Table 76: Calculated Product Convoy Parameters to Tuliara

Description	UoM	Value	Note
Product Target	tpa	150,000	Planned production
Graphite per bulk bag	kg	1,000	Estimated t per bag
Bulk bag trailer 1	-	28	AsstrA
Bulk bag trailer 2	-	0	
Bulk bags per truck	-	28	AsstrA
Graphite pay load	t	28	Calculation
Trucks per convoy	-	20	Estimated Calculation per day {26 days per month)
Product per convoy	t	560	Estimated Calculation
Convoys per annum	-	268	Estimated Calculation

To ensure an effective operation based on simulation model calculations applied to the new calculation, and with a fleet for 5 convoys, it suggests a total fleet of 170 trucks. Multiple camps and support facility requirements will be needed enroute for vehicles and driver accommodation between Molo site and the port allowing for truck overnight stops with refuelling services. These are averages and figures do not consider production availability, truck availability, road conditions, effective loading and unloading time and shipping schedules.

18.5.5 Port Facilities

Receipt of cargo at Tulear warehouse, (a facility with a minimum holding capacity of 2,400 tonne), will involve product being offloaded into store. Direct loading from truck trailer into shipping containers will be done if possible to increase total handling capacity.

The capacity of a 20 ft GP container is 24 Bulk Bags (24 tonne).

The capacity of a 40 ft GP container is 32 Bulk Bags (32 tonne).

For the initial export shipment (to Qingdao) during Phase 1, all bags were packed into the 20 ft containers.

Movement from warehouse facilities to port for vessel loading is by truck, with a maximum trip distance of 6 kms.

18.5.6 Risks and Mitigations

A risk analysis has been conducted as part of the FS that included the routes to Tulear port, as well as to the port at Fort Dauphin.

The main risks identified during for the completion of the study are:

- Truck availability.
- For Phase 1, due to the general road conditions from site to Tulear, truck availability as indicated in the simulation model of 90% could be much lower.
- Waiting time for loading and offloading of trucks.
- Long periods of heavy rain which may render parts of the route impassable.
- Ensure road maintenance are kept to high standards.
- Bridge and river crossings.
- Shipping schedules and container availability.
- Environmental and social impact along the road because of the increased traffic.
- Safety risk factor for local people in villages and living beside the road.

Table 77: Product Transport Risk Mitigation

Risk	Area Identified	Mitigation / Action Required
1	Truck Availability	Contract negotiation to ensure the hauling fleet will be capable of moving the product on time with spare capacity to cater for breakdowns. In addition, vehicle service and maintenance station will be required along the road to improve availability of fleet. Due to change in routes and general hauling conditions a full transport study will be completed in 2024 and improve the current simulation study for all routs.
2	Truck Turnaround	Truck loading and offloading currently has been budgeted for using of forklift trucks. This process could be slower than anticipated as indicated in the simulation study. The installation of overhead cranes in store will be evaluated to improve loading time and similar installation at loading docks. With an improved simulation study the total fleet will be better assessed.
3	Weather Conditions	During severe rains, (predominantly months of January and February), that create impassable conditions on the road for heavy traffic, additional storage of graphite concentrate will be required at site. To eliminate this bottleneck additional storage capacity will be reviewed during detail design for the project.
4	Road Maintenance	Road degradation will affect the quantity of material that will reach port for export. This is likely to be an annual, (particularly during the months of January / February). A road maintenance plan agreed between all stakeholders needs to be implemented.
5	River Crossings	There are multiple water courses along the routes whose capacity will increase during the rainy season and where the crossings are fords, their condition will deteriorate without remedial action. During the rainy season, bridges along the route too may, or may not have issues, but some of the bridges will require maintenance and repair work that will require project management from Molo site personnel.

Risk	Area Identified	Mitigation / Action Required
6	Shipping	Tulear Port is served by CMA CGM container line approximately every 17 days. This is a reliable, yet expensive service for the export of graphite. If this service were to cease, export cargo movement could prove difficult until expansion volumes are flowing. CMA CGM export freight rates are high due to their monopoly. On both of these points, new options need to be developed with other carriers (MCCL, PIL, Maersk etc).
7	Environmental	Possible impact on the environment could emanate from project related logistics i.e. diesel fumes, noise, product spills, road dust, light pollution etc. Preventative measures need to be put in place and their key components added to the final trucking contract.
8	Safety	Safety of personnel, equipment and product is paramount to the project. A purpose drafted HSE plan needs to be created, published to all stakeholders, and implemented. The policy needs to cover the HSE norms of NextSource Materials Inc. and all other stakeholders. As a supplier to NextSource Materials Inc., the transport contractor would have to agree to create the internal procedures and capacity to adhere to these requirements.

19 MARKET STUDIES AND CONTRACTS

Independent market studies were conducted by Benchmark Mineral Intelligence (Benchmark) and Fast Markets and summarised into this chapter. Both Benchmark and Fast Markets are independent credible specialist organisations that assess, analyse and report on the world supply and demand for graphite products including flake graphite to generate a forecast over a 20 year period commencing in 2023. The following is a summary of those market studies and update of the contracts Molo has in place.

19.1 Introduction

The flake graphite market continues to change and develop as demand for graphite increases. Graphite prices are regularly priced on an FOB China basis as this is the largest current market base. As the demand in North America and Europe increases, potential premiums could be achieved for non-Chinese-based materials. Flake sizing is broadly classified into four ranges:

- Small (-100 mesh, or <75 µm).
- Medium (-80 to 100 mesh, or 75 µm to 180 µm).
- Large (-50 to 80 mesh, or 180 µm to 300 µm).
- Extra-large, or jumbo (+50 mesh, or >300 µm).

These flake sizes are in turn classified by carbon content (“C”) and are typically sold in ranges of 88% C, 93% C, 94%-95% C, and 95%-97% C. The specific technical attributes of the flakes are then defined by end-user parameters such as expansion coefficient, thermal and electrical conductivity, and charge-discharge stability and efficiency. As the technical parameters sought by end-users are proprietary to their processes, pricing is not publicly available. Benchmark and Fast Markets both provide monthly graphite pricing for various flake sizes and carbon purities based on input from graphite suppliers and purchasers.

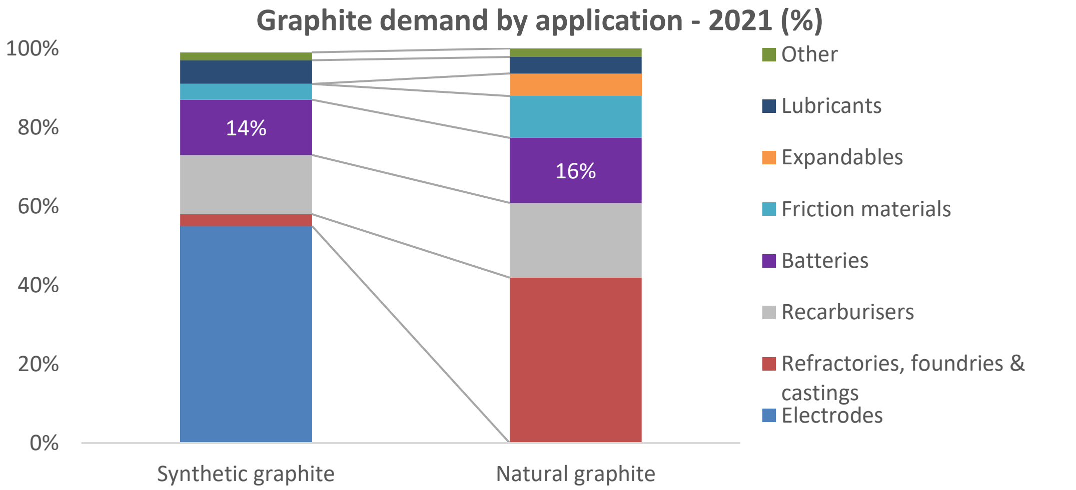
The key factors that drive flake graphite pricing are the graphite size and purity. Currently, the market gives a price premium for larger mesh-size (+80 mesh) products due to restricted supply. Premiums are also achieved for higher carbon purity graphite (+94% C) as it has fewer impurities and is less widely produced. The price for grades higher than 94% C increases at approximately 4.1% per C percentage.

19.2 Graphite Demand

19.2.1 Demand by Graphite Source

Graphite fines are used in several key sectors. However, it is graphite’s importance to the eMobility sector that is seeing its demand rise rapidly. Demand has seen above long-term trend growth over the last 5 years, primarily due to its application in the anodes of EV batteries.

It is worth noting the relationship between natural graphite and synthetic graphite. While most applications are agnostic to the source of graphite, with the choice coming down to availability and cost differential, some applications favour one of the two sources. Graph 41 below shows the split of the applications by source.



Source: Fastmarkets

Graph 41: Graphite Demand by Application – 2021 (%)

Overall, demand for all graphite rose from 2.18 Mt to 3.42 Mt between 2016 and 2021 an increase of 9.4% compound average growth rate (CAGR). Over the same period, natural, mined graphite demand rose at 4.1% CAGR, while demand for synthetic graphite rose at 13.1% CAGR. Synthetic graphite's stronger growth can be attributed to not only increasing demand for graphite electrodes from electric arc furnace steelmakers and strengthening aluminium production, but also a strong ramp-up in consumption in the battery sector.

Unless specified, the rest of this market summary considers only natural graphite.

19.2.2 Sectorial Demand

When looking at graphite, it is beneficial to split graphite demand into traditional applications, batteries, and novel applications. The applications have different drivers, and so are seeing different rates of growth.

As an example, demand for graphite from traditional end-use sectors rose at a steady, measured pace of around 3% CAGR over the period 2016-2021. This trend is expected to be maintained. Meanwhile, demand for graphite from the battery sector rose at 46% CAGR over the same period. This reflects the sharp increase in the production of electric vehicles (EVs). Other novel uses for natural graphite, such as expandable graphite, are also experiencing strong demand rises, albeit from low starting points. Refer to Table 78 below.

Table 78: Summarises the Expected Growth Rates for Each Application Area

Expected Growth Rates of Natural Graphite Demand by Application		
Classification	**Application**	**5 Year Growth Rate (% CAGR pa)**
Batteries	Batteries	>20% pa
Novel	Expandable Graphite	>15% pa
Novel	Graphene	>30% pa
Traditional	Refractories, Foundries and Crucibles	1-5% pa
Traditional	Re-carburisers for Steel	1-5% pa
Traditional	Lubricants	1-5% pa
Traditional	Friction Products	1-5% pa
Traditional	Pencils	1-5% pa
Traditional	Nuclear Industry	1-5% pa
Traditional	Paintings and Coatings	1-5% pa
Traditional	Brake Pads for Vehicles	-2-0% pa

Source: Fast Markets

Graphite is a necessary, significant input into several applications. Though, it is the surge in lithium-ion battery-powered EVs that has brought intense scrutiny of graphite availability, for the immediate future and in the coming years.

This ramp-up in EV production is already evident: In 2016, only 35 GWh of batteries were used in electric vehicles, but by 2022 this had risen to 581 GWh. To advance the energy transition, the trend will accelerate in the coming years. Especially, as the use of engines in passenger vehicles is legislated against in most jurisdictions. Fast Markets forecasts that battery demand in EVs will continue to see double-digit growth, with a compound average growth rate of 21.0% per annum over the coming decade.

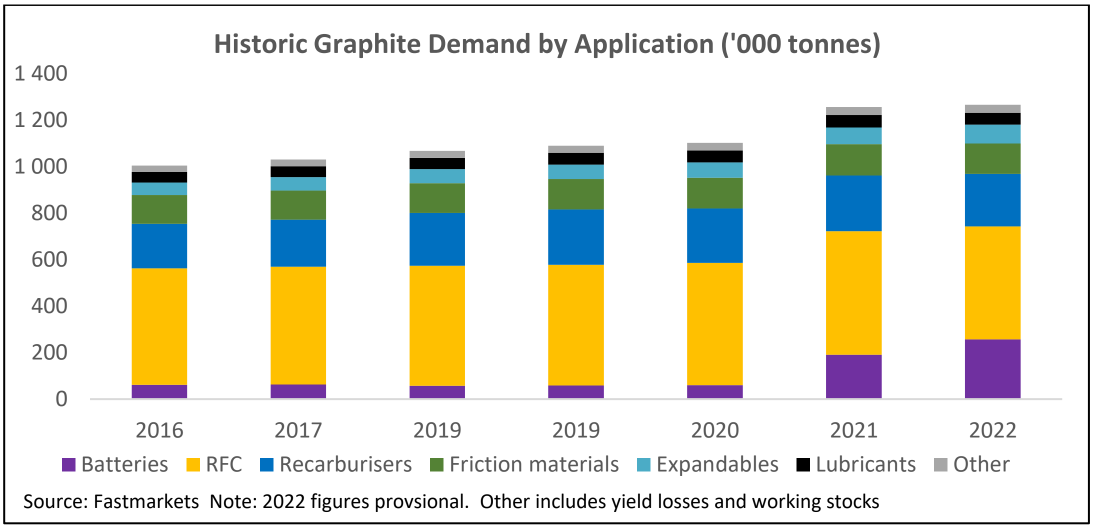
With an average of 800 kg – 850 kg of graphite needed in each gigawatt-hour, the implications for graphite are profound. Graphite demand from the battery sector is forecast to rise from 0.59 Mt in 2021 to 3.83 Mt in 2032. Besides EVs, energy storage will bring additional demand for batteries, to support the energy transition.

Not all batteries will be based on lithium-ion batteries or use natural graphite exclusively, and there are challenges for the graphite industry, as it develops and adapts to the needs of the rapidly growing EV sector. Despite this, Fast Markets forecasts exponential growth from battery applications, seeing graphite demand rise from 0.11 TWh in 2016 to 0.71 TWh in 2022 and reaching 4.96 TWh in 2032.

19.2.3 Historic Demand

Over the 5 years to 2022, natural graphite demand has increased at 4.2% compound average growth rate (CAGR); much in line with the growth of global industrial production. Demand may have stalled in 2022, despite a clear uptick in the battery sector, but this was due to the weak steel industry, caused by macro-economic headwinds.

The rapid rise of the battery sector is evident in Graph 42 below. Between 2016 and 2020, the sector accounted for 5% - 7% of natural graphite demand. However, in 2021 this jumped to 15%, as the EV sector started to gain momentum, and rose to over 20% in 2022.



Graph 42: Historic Graphite Demand by Application ('000 tonnes)

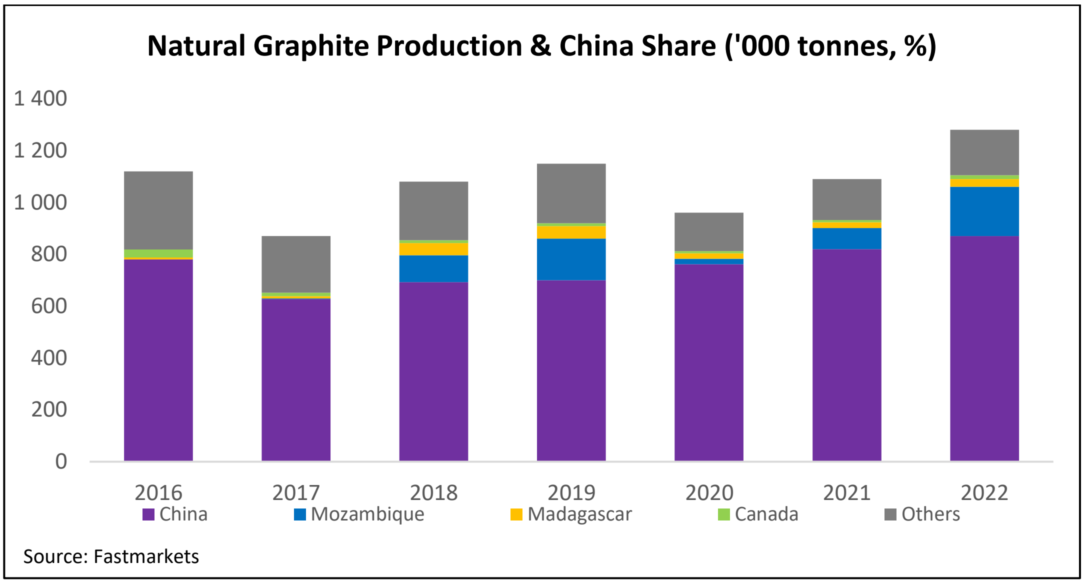
Between 2022 and 2032, Fast Markets expects graphite demand to rise at 13.1% CAGR each year, as EVs continue to gain market share and energy storage becomes more widespread. With the sector accounting for 20% of graphite demand in 2022 will increase to 69% by 2032.

Such strong annual growth in a commodity over a sustained period is unusual. There are few examples of where a single application, especially one associated with a relatively new technological focus, like the energy transition, has ramped up so quickly. This requires some pause for concern about the ability for production and the associated supply chains to adapt to the increased load. Satisfying demand of less than 1.3 Mt in 2022 to an estimated 4.4 Mt by 2032 will be difficult to achieve. However, realizing that global supply chains can move over 100 Mt of iron ore monthly, it is expected that supply chains will rise to the challenge, albeit with challenges and the need for considerable investment.

19.3 Natural Graphite Supply

19.3.1 Production

China has accounted for more than half of the natural graphite supply since at least 2000. Having reached 72% of global supply in 2017, the country's share has been easing as new production sources, such as those in Madagascar and Mozambique, have been developed. However, the effects of the COVID-19 pandemic on graphite demand, industrial supply chains and, consequently, low prices slowed output from sources outside China; allowing China's market share to rebounded to 79% in 2020 and 75% in 2021, before dropping to 68% in 2022. Refer Graph 43 below.



Graph 43: Natural Graphite Production and China Share ('000 tonnes %)

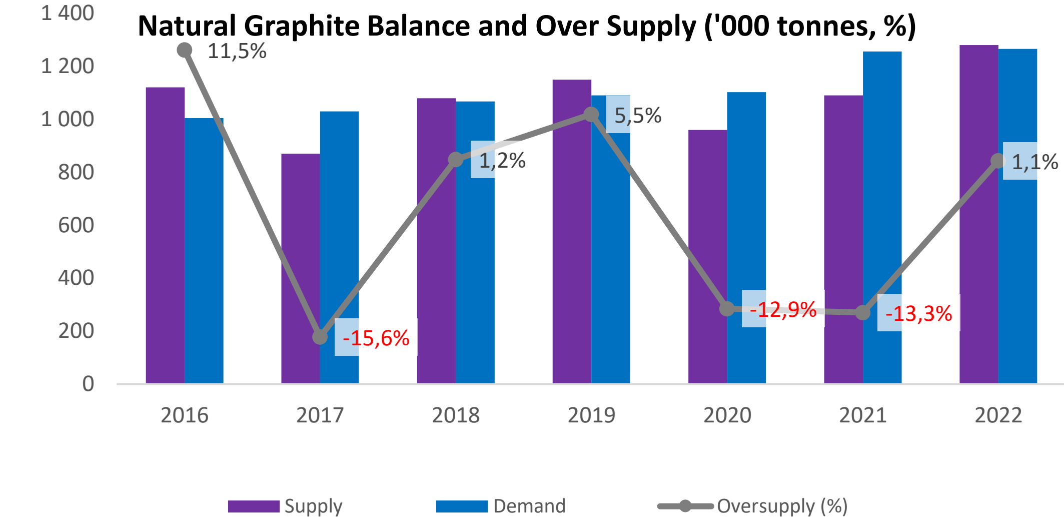
19.3.2 Mine Capacity

When considering graphite resources, there are no perceived, physical shortages. Estimated global deposits are 800 Mt, which is sufficient to satisfy demand for centuries to come, even if actual demand is several times higher than forecast. Therefore, the issue for graphite supply is one of economic extraction. With the mined product's characteristics and cost being of most importance, but logistics and emissions' intensity also of concern.

Benchmark Market Intelligence and Fast Markets both maintain databases of graphite mining projects. Both confirm that there are over 100 projects spanning more than 12 countries at differing stages of development. Not all provide capacity, or production plans, but of those that do the mean capacity of each is 51,000 tpa. So, approximately 2.5 Mt of capacity may be expected, if all the projects were completed.

19.4 Graphite Supply and Demand Balance

Natural graphite supply and demand has been in reasonable balance for many years, and has moved into a mild deficit recently. Supply is responsive to market fundamentals, which helps stop the market moving too far out of balance. Anticipation of increased demand and tight supply in 2017 prompted the arrival of some additional graphite capacity. In 2020 and 2021, when the COVID-19 pandemic undermined demand, supply from Africa was reduced to prevent excessive inventory building and depressing future prices. As demand lifted, so did supply. Refer Graph 44 below.



Source: Fastmarkets

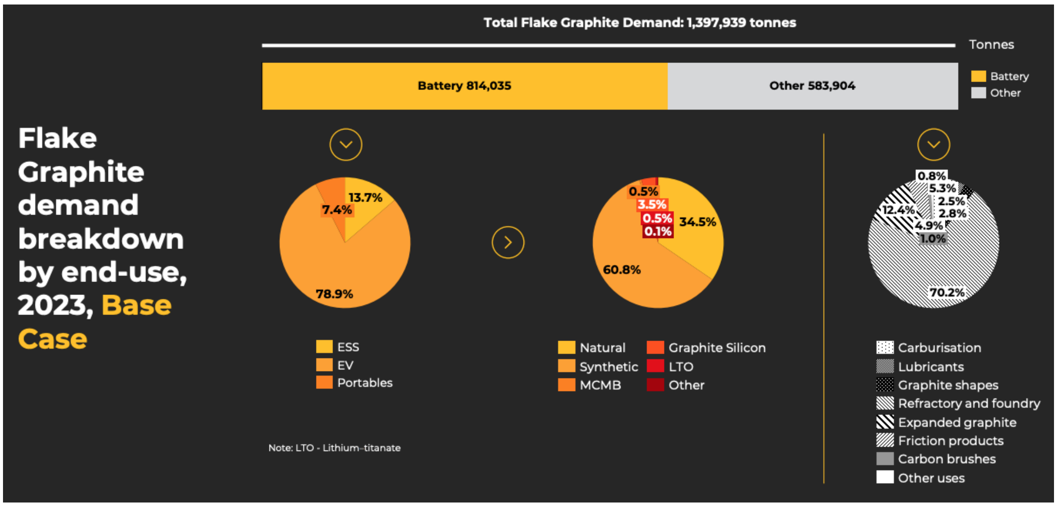
Graph 44: Natural Graphite Balance and Over Supply ('000 tonnes, %)

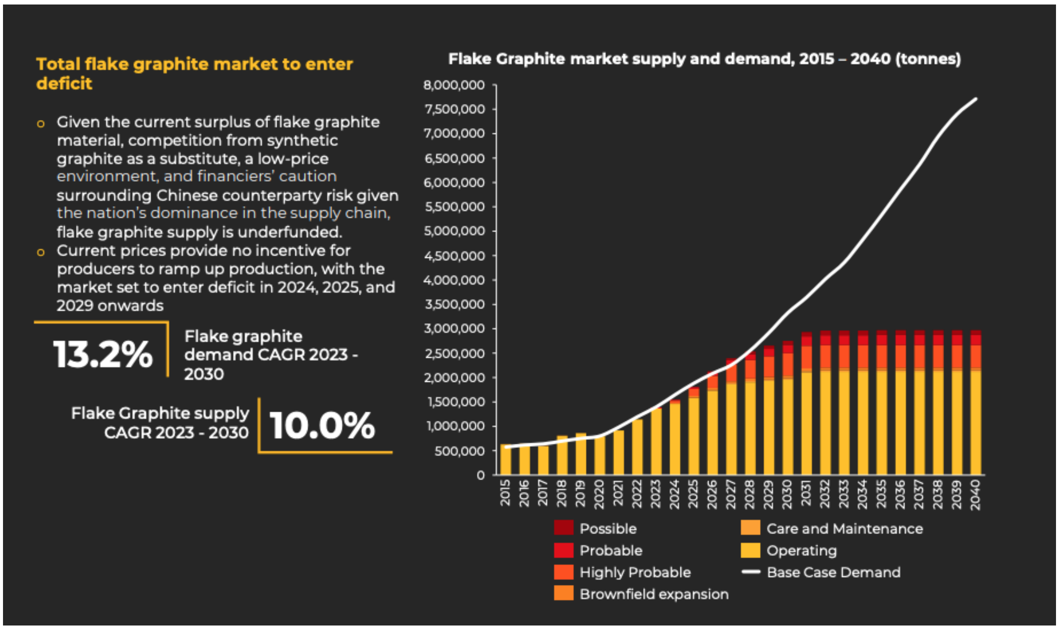
Fast Markets' view is that the rapid growth in the EV sector will propel demand and prices for graphite higher in the coming years, bringing a tight market. Refer to Table 79 below.

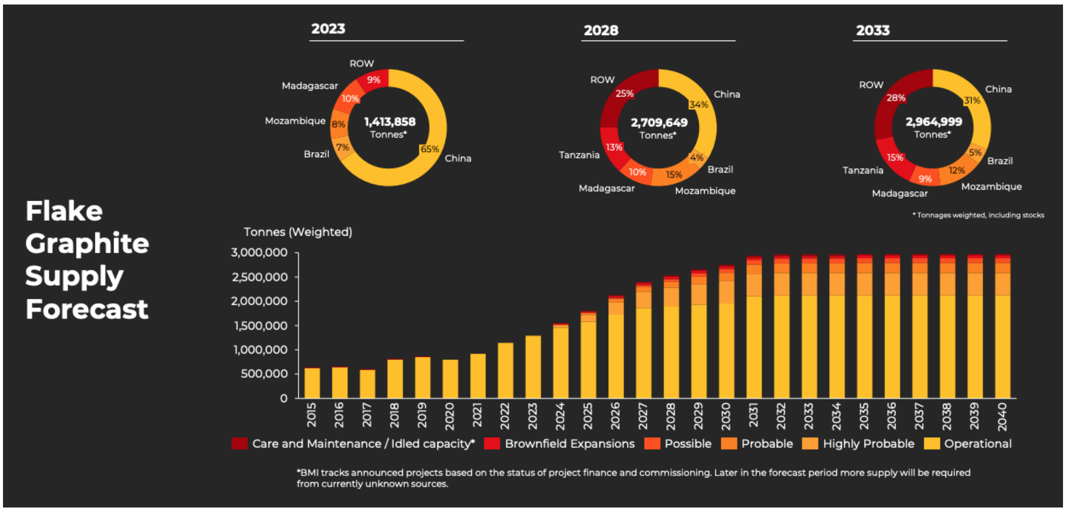
Table 79: Graphite Supply – Demand Balance and Market Over Supply ('000 tonnes, %)

Graphite Supply-Demand Balance and Market Over Supply ('000 tonnes, %)							
Item	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**
Demand	1,004	1,030	1,067	1,090	1,102	1,257	1,266
Supply	1,120	870	1,080	1,150	960	1,090	1,280
Balance	116	-160	13	60	-142	-167	14
Over Supply (%)	11.5%	-15.6%	1.2%	5.5%	-12.9%	-13.3%	1.1%
Source Fast Markets							

In the coming years, supply will increase as new mines start production and existing ones are expanded. This will see supply getting ahead of demand and looser market fundamentals developed. However, as EV demand continues to grow, surpluses will be absorbed bringing the market into balance again by 2032. Refer to Graph 45 below.







Graph 45: Total Flake Graphite Demand 1,397,939 t

19.5 Natural Graphite Pricing

Graphite is wrongly classified as a commodity. Unlike notable commodities, especially homogenous energy commodities and traded base metals, graphite has a number of characteristics that affect which applications it is suitable for. These are reflected in demand and prices. Physical characteristics are the most important to customers, but there is a growing focus on surety of supply and providence has seen origin becoming of importance. This scrutiny will intensify as regional governments consider graphite a critical resource and recognize its importance in supporting industrial policies.

19.5.1 Flake Size

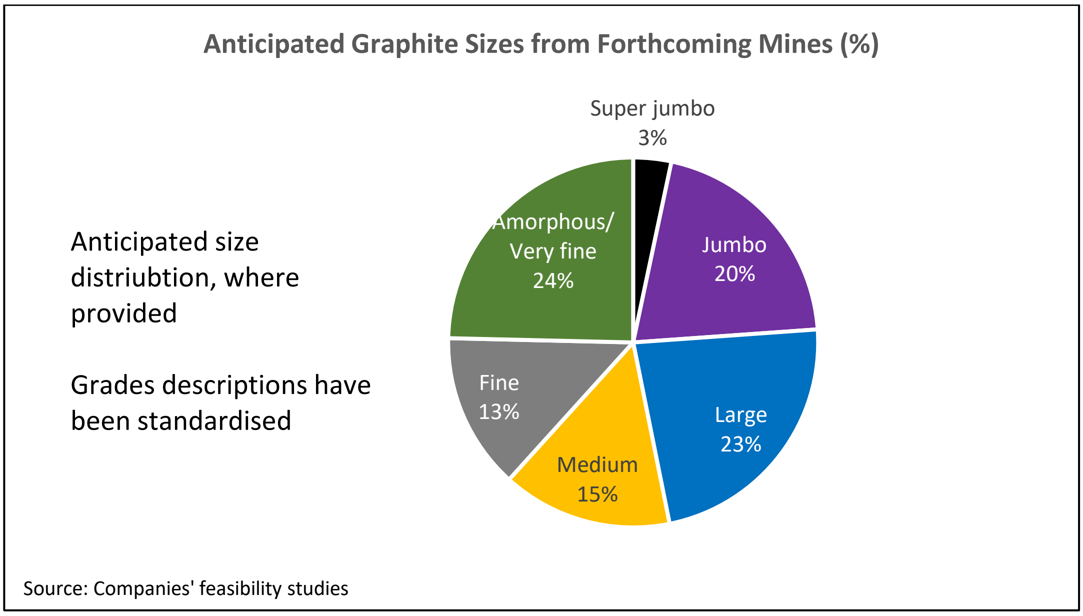
Besides the particle shape, the size of the flake is an important characteristic, as this can dictate where the material can be used. For expandable graphite, larger flakes are needed. For batteries, fine flakes are the best starting material, though larger flakes are also used and downsized. Table 80 below shows a generalized breakdown of different flake sizes, applications, and indicative, relative prices. The classification of flake size is not fixed, and different mines will use slightly varying values. Graphite flakes are sized by passing through meshed screens. The size is quoted as the smallest square hole the material can pass through, expressed as a mesh number, or in micrometres (µm).

Table 80: Natural Graphite Flake Classified and Example Characteristics

Natural Graphite Flake Classification and Example Characteristics					
Classification	**Flake size (µm)**	**Mesh (%)**	**TGC (%)**	**Typical Applications**	**Relative Price**
Amorphous / Very Fine	<75	-200	80-85	Industrial uses.	1
Fine	75-150	-100/+200	>99	Battery applications.	1-2
Medium	150-180	-80/+100	94-97	Industrial uses.	4-6
Large	180-300	-50/+80	94-97	Refractories, crucibles.	6-8
Jumbo	300-500	-30/+50	97-99	Expandable graphite, composites, electronics.	6-10
Super Jumbo	>500	+30	97-99	Specialised applications, e.g. nuclear reactors, aerospace, graphene.	10-15
TGC: Total Graphitic Content					
Source: Fast Markets					

Companies looking to develop a new natural graphite mine will often publish the proportions of the flake sizes their mines are expected to deliver as illustrated in Graph 46 below. This information indicates the potential for the mine to satisfy the higher-value battery and novel applications, and so can contribute to a higher valuation of each mine.

The focus on flake graphite exploration has already brought a slow, steady shift towards satisfying the needs of the battery sector and expandable graphite markets. This trend is expected to accelerate.



Graph 46: Anticipated Graphite Sizes from Forthcoming Mines (%)

19.5.2 *Total Graphitic Content*

Mined flake graphite is usually beneficiated at the mine site, where its carbon purity, (Total graphitic content, TGC) is raised to 94-95% C by simple, low-technology processes, such as flotation. This TGC is acceptable for many applications or as a feedstock for further processing. For battery and specialist applications higher purities are needed. The process of raising the purity further uses acids at elevated temperatures to reach 99% C and higher.

19.5.3 *Carbon Particle Shape / Carbon Morphology*

Carbon is extracted either as flat, flake-like platelets or as sand-like, amorphous material. Amorphous graphite is more plentiful, easier to extract, and requires less processing prior to selling. It made up 42% of production in 2021, with over 90% of this material coming from China. Flake graphite, 58% of 2021 production, requires additional care in handling to avoid breaking the flakes, but has a wider range of applications and so can attract a greater margin than amorphous graphite. A third, minor type, called vein graphite, which accounts for 0.4% of production, is found only in Sri Lanka. Here underground mining is used to extract graphite from veins, whereas other graphite deposits use surface mining. The higher costs of these operations are offset by the price premium for the high-purity material.

The growth of the new applications has incentivised the search for flake deposits globally. This is partly due to the mines of China being richer in amorphous graphite and having been somewhat depleted of flake graphite, but mostly from the forecast surge in demand for material suitable for battery applications.

19.5.4 Origin

A growing differentiator is the origin of the material. This is due to logistics costs and, increasingly the threat of resource nationalism.

Several geographies, including the European Union and the United States of America, consider access to natural graphite as critical to major industries, like automotive manufacturing and renewable energy. Consequently, there is growing concern about guaranteeing high-quality flake graphite supply for countries with insufficient graphite mines. Shipping graphite also adds to the Green House Gas (GHG) footprint of the material, which companies are trying to minimise.

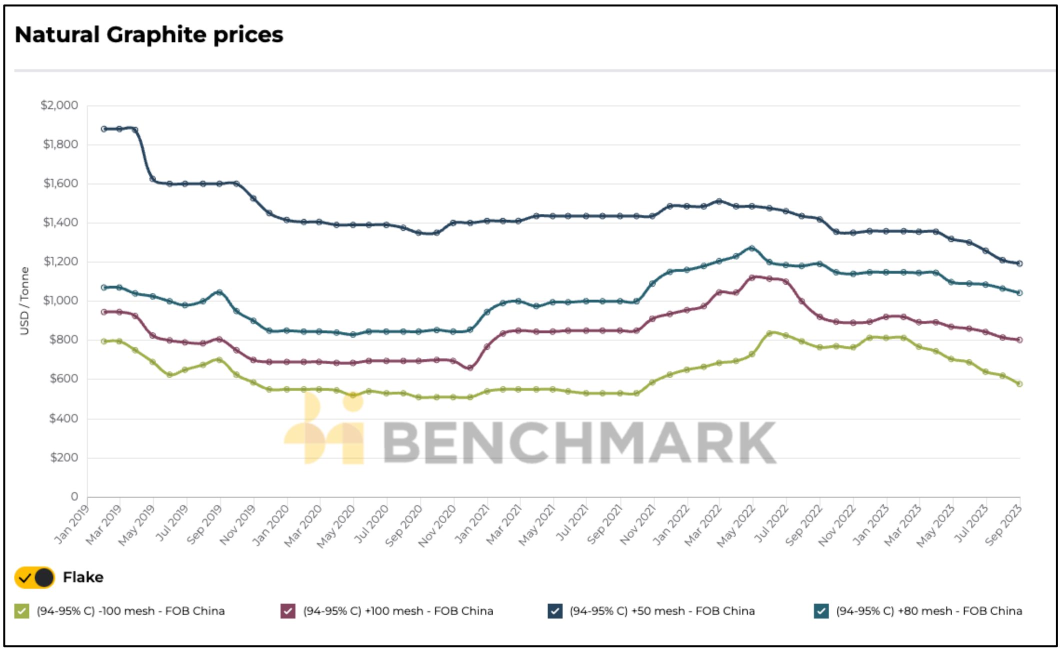
19.6 Price Development

19.6.1 Natural Graphite Historic Prices

Natural graphite prices vary for several reasons above. The graph below shows the historic prices collected by Benchmark for Chinese flake. The variation in prices across different flake sizes can be readily seen.

Historically, Chinese prices have traded at a discount to European prices, as greater availability and lower costs of production kept prices under pressure. This is less the case now, with China having to increasingly import graphite to satisfy its industries' needs, which has brought prices more into alignment globally. However, it is worth noting that Chinese prices are quoted as FOB, so transport costs are not included.

Historical and current natural graphite prices are shown in Graph 47 below.



Graph 47: Historical and Current Natural Graphite Prices

Throughout 2023, all flake graphite prices assessed by Benchmark continued to tick downwards, with the Benchmark Flake Graphite Price Index falling by 2.9% month-on-month, attributable to the peak seasonal production at Chinese graphite mines in Heilongjiang Province during the summer months.

In addition to the growing supply-side developments in China, demand for flake graphite from the EV industry also improved during the assessment period, with flake graphite producers experiencing increased order book and shipment volumes.

Growing downstream demand was reflected in June EV sales figures in China, which experienced an increase of 12.5% month-on-month to 806,000 vehicles, and Chinese EV production statistics which rose by 9.9% month-on-month to reach 784,000 units. Additionally, in June, global EV sales experienced an increase of 18.2% month-on-month, to reach over 1,250,000 vehicles, reflecting gradual rising demand from the EV sector worldwide. In contrast, traditional end markets, which to date remain the largest consumption driver for flake graphite, remained subdued throughout July, against a wider backdrop of macro-economic uncertainty in the Chinese domestic market. This was reflected in June's crude steel production figures, which reached 91.1 Mt, up just 0.4% year-on-year.

Table 81: September 2023 Spot Prices for Natural Flake Graphite per Tonne

Flake	September 2023	August 2023	Change
+50 Mesh, 94-95% C, FOB China	US$1,185	US$1,193	-0.60%
+80 Mesh, 94-95% C, FOB China	US$1,035	US$1,043	-0.70%
+100 Mesh, 90-93% C, FOB China	US$733	US$740	-1.00%
+100 Mesh, 94-95% C, FOB China	US$788	US$803	-1.90%
+100 Mesh, 94-95% C, DDP China	RMB 5,750	RMB 5,750	0.00%
-100 Mesh, 90-93% C, FOB China	US$445	US$450	-1.10%
-100 Mesh, 94-95% C, FOB China	US$563	US$578	-2.60%
-100 Mesh, 94-95% C, DDP China	RMB 4,050	RMB 4,200	-3.60%

19.6.2 Effect of Total Graphitic Content on Price

-100 mesh flake, which is the most favoured flake size by the battery sector, has been converging in recent years in price to the larger flake sizes due to demand as feedstock for producing graphite anode material. Not only has the -100 mesh gained relative to other sizes, but the premium for higher total graphitic content has also increased noticeably. This reflects the battery's sector demand for high-purity flake.

For carbon grades above 95% C, Fast Markets calculates a quarterly "Value in Use" (VIU) premium. This quantifies how higher carbon purities attract varying premia over the nominal, 94% C price as shown below. The VIU multiplier of 4.1% can be applied to each percentage from the base of the 94 - 95% C price. For example, the July 2023 FOB China price of +50 mesh at 97% C would be US$1,309/t. The premia have trended down due to greater availability of higher purity natural graphite, weakening synthetic graphite prices in China, and, recently, problems in the steel sector.

19.7 Flake Graphite Forecast

This feasibility study is based on short, medium, and long-term forecasts underpinned by Benchmark and Fast Markets across the product sizes Molo is expected to produce. The FOB China pricing forecasts have been modified with the following factors:

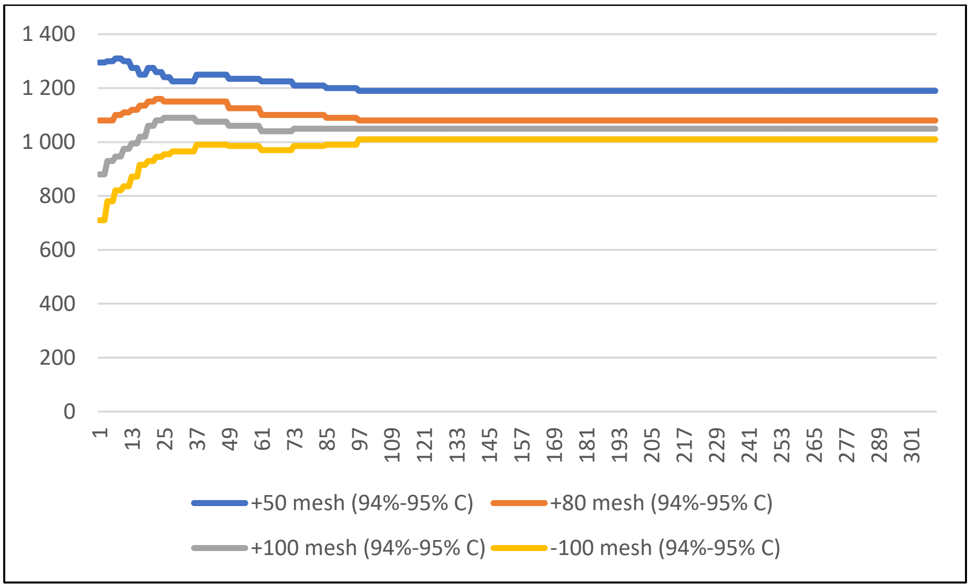
- FOB Madagascar reduces shipping and logistics costs to North American and European markets when compared to FOB China.
- Flake size distribution premiums per forecast data.
- Higher carbon content attracts a 4.1% premium per % C, over the published 94-95%C forecasts.

As per Table 82 below, the 5 year average "basket price" used for the Feasibility Study is US$1,088.36/t. The selling price was derived using the above flake size distribution arrived at from metallurgical testing, and the pricing information from Benchmark Mineral Intelligence and Fast Markets, on a weighted average of each size fraction and adjusted for increased carbon content using a "Value in Use" premium, on an FOB Madagascar basis.

Table 82: 5 Year Average Weighted Basket Price based on Molo Flake Distribution, FOB Madagascar

Microns	Mesh Size	C%	Yield (%)	Sale Price US$	US$/t
>300 µm	+50 mesh	96.9%	23.6%	US$1,604	US$378.50
180 µm – 180 µm	+80 mesh	97.0%	22.8%	US$1,215	US$277.09
150 µm – 180 µm	+100 mesh	97.2%	6.9%	US$981	US$67.69
<75 µm	-100 mesh	97.6%	46.7%	US$782	US$365.08
		97.2%	100.0%		US$1,088.36

Graph 48 below shows the LoM forecast price by mesh size.



Graph 48: Life of Mine forecast price by mesh size

Potential premiums which may become evident in the future which form part of the price sensitivity analysis include:

- A premium for non-Chinese supplied graphite in North America and Europe.
- An additional premium in North American markets due to the Inflation Reduction Act.
- A premium for sustainably mined and processed graphite.

19.8 Contracts and Agreements

Independent testing by various third-party end-users of flake graphite was announced by the Company in 2015, that confirmed that flake graphite concentrates from the Molo graphite mine meet, or exceed quality requirements for all major end-markets of natural flake graphite. The major end-markets for flake graphite include refractories, graphite anode materials used in lithium-ion batteries, specialty graphite foils used as essential components in the chemical, aeronautical and fire-retardant industries and graphene used in high-end ink and substrate applications.

The Company expects to sell most of the flake graphite produced at the Molo graphite mine through off-takes with several key customers. All of Phase 1 production is spoken for by off-take sales agreements by two main parties.

On October 16, 2018, the Company announced a binding off-take agreement for the supply of SuperFlake® graphite concentrate with a prominent Japanese Trading Company ("Japanese partner") that is a primary supplier of flake graphite to a major Japanese electric vehicle anode producer. To protect certain confidential aspects of the agreement, the Japanese Trading Company and the Japanese electric vehicle anode producer requested not to be identified.

The key highlights are:

- Off-take is for a period of 10 years, beginning at the start of commercial production at the Molo Graphite Mine, with an automatic renewal for an additional 5 years.
- Exclusive right to import and sell SuperFlake® graphite concentrate in Japan.
- Provided that commercial production commences within 3 years, following the ramp-up period, the Japanese Partner will purchase 20,000 tpa of SuperFlake® graphite.
- Product prices will be negotiated on a per-order basis between the parties and will be based on the market prices (FOB basis) prevailing in the region.

On May 25, 2021, the Company announced that following a multi-year verification process, thyssenkrupp entered into a long-term partnership with NextSource and signed an off-take agreement to secure SuperFlake® graphite concentrate for their refractories / foundries, expandable graphite, (graphite foil) and battery anode production businesses. The key highlights are:

- Commercial agreement for the sale of 35,000 tpa of SuperFlake® graphite concentrate from the Molo mine.
- 10 year term with an automatic 5 year extension.
- Products under the agreement pertain to refractory, battery anode production and expandable graphite, (graphite foil) markets.
- Geographical regions include, but are not limited to, Europe, UK, North America, Mexico, China, and South Korea.
- Minimum 7,300 tpa during Phase 1 initial production.
- Ramp up to 35,000 tpa.
- Shipments in Phase 1 will be used to verify run-of-mill production to trigger the larger volume expansion.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL COMMUNITY IMPACTS

20.1 Description of Available Information and Permits

The availability of current and accurate environmental, social, and permitting information on the Project is substantial. Table 83 provides a summary of the information and permits on-hand as prepared and obtained during Molo Phase 1.

Table 83: Information Summary

Title	Author / Issuer	Year
Environmental and Social Baseline (initial).	Agetipa	2011 - 2012
Environmental Legal Review.	GCS Water and Environment.	2013
Environmental and Social Sensitivity Study (spatially integrated).	GCS Water and Environment.	2014
Final Terms of Reference (ToR) and Final Memorandum of Understanding (MOU) between ERG Madagascar SARLU (the Company) and the O.N.E. (regulator).	Agetipa and GCS / O.N.E.	2017
Environmental and Social Impact Assessment and Management Plan (ESIA & ESMP) including Residual Baseline.	GCS Water and Environment and Agetipa.	2017
Baseline Relocation Action Plan (RAP).	Agetipa	2017
Mineral Exploitation Permit for PE39807.	ERG Madagascar / Ministere des Mines et du Petrole.	2019
Approved Global Environmental Permit for Exploitation Right PE 39807	O.N.E.	2019
Approved Relocation Action Plan - Phase 1	Harizo RasoLoManana / O.N.E.	2021
Approved Specific Environmental Management Plans (SEMP) for the Phase 1 Construction and Operational Phases: SEMP Energy Generation. SEMP Accommodation Camp. SEMP Mining. SEMP Processing. SEMP Roads, Pipelines and Dams.	Harizo RasoLoManana / O.N.E.	2021-2023

Title	Author / Issuer	Year
Approved Water Abstraction Permit for Phase 1 Construction and Operations.	Harizo RasoLoManana / Autorite Nationale de l'Eau et de l'Assainissement (ANDEA).	2021
Approved Tree Clearance Permit for Phase 1.	Harizo RasoLoManana / Direction Regionale de l'Environnement et du Developpement Durable Atsimo Andrefana.	2021
Approved Corporate Social Investment Plan (Phase 1).	Globesight / NextSource Materials.	2021
Approved Re-vegetation Plan (Phase 1).	Harizo RasoLoManana / O.N.E.	2021
Approved Thermal and Solar Electrical Generation Permits for Molo Phase 1	ERG Madagascar / Ministere de l'Energie et des Hydrocarbures	2021
Approved Emphyteutic Lease for Molo Phase 1.	ERG Madagascar / Ministere de l'Amenagement du Territoire et des Services Fonciers.	2022
Approved Building Permit and Construction Authorisation.	ERG Madagascar / Ministere de l'Amenagement du Territoire et des Services Fonciers.	2022
Permitting and Stakeholder Register.	Globesight / ERG Madagascar.	2022

20.2 Applicable Laws and Standards

This component of the FS is planned in accordance with Malagasy National Legislation, Equator Principles, World Bank and IFC Standards on environmental, cultural, health and safety protection.

20.3 Environmental and Social Sensitivities Vs Development Footprint

Sensitive areas were super-imposed onto the Project design and the placement of infrastructure such as roads, infrastructure expansions, linear developments and other applicable ancillary infrastructure was undertaken in such a manner as to not have any significant impacts on the environment, or on the livelihoods of local inhabitants.

Refer to Figure 105 below for Phases 1 and Expansion for development footprints.

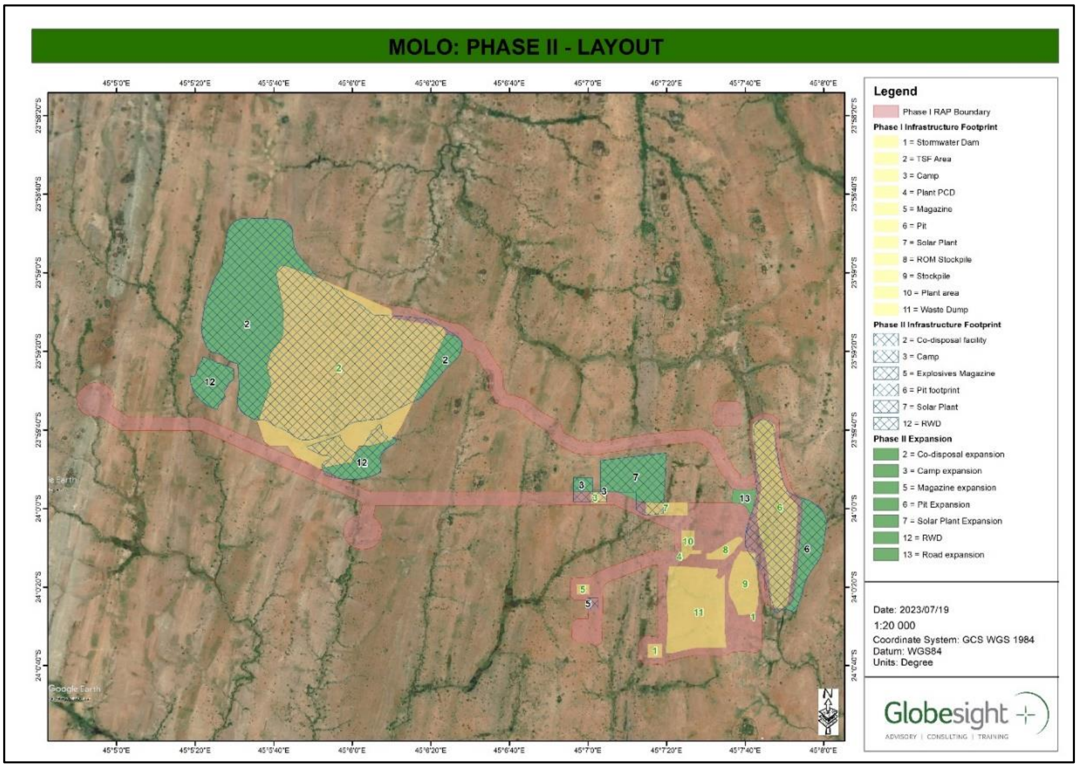


Figure 105: Phase 1 and Expansion Development Footprints

20.4 Environmental and Social Authorizations

20.4.1 Global ESIA Update

The Global Environmental and Social Impact Assessment for the Molo Phase 1 Project has been approved by the ONE in Madagascar on April 8, 2019. The Molo Expansion will, according to the ONE, require an amendment to the Phase 1 Environmental and Social Impact Assessment (ESIA). The original Global Environmental Authorization remains valid, however, an updated ESIA is required and the Chahier des charges Environnement (CCE) will then be adjusted accordingly.

This will be a 12 to 18 month process and should, therefore, be initiated in a timely manner prior to the commencement of Expansion construction and implementation works.

20.4.2 *Specific Environmental Management Plans (SEMPs)*

The proposed expansion of infrastructure, the mining pit, as well as the mine waste disposal requirements to accommodate the increased throughput capacity and the adjusted LoM plan for the expansion will require amendments to the following construction and operational specific environmental management plans (SEMPs), which were developed for Molo Phase 1, prior to commencement of the Expansion construction and implementation works:

- SEMP Energy Generation.
- SEMP Accommodation Camp.
- SEMP Mining.
- SEMP Processing.
- SEMP Mining Waste (Tailings / Co-Disposal).
- SEMP Roads, Pipelines and Dams.
- SEMP Conservation Site.
- SEMP Relocation Action Plan.

An application for amendments to the respective SEMPs should be initiated in a timely manner prior to the commencement of the expansion phase land clearance and construction works.

20.4.3 *Relocation Action Plan and Livelihood Restoration*

The Molo Expansion development plan will require an additional ~270 ha to accommodate the increased footprint of the accommodation camp, solar farm, tailings disposal facility, mining pit (with buffer) and mineral processing. The estimated affected land parcels, (crops and fields), which would require monetary compensation, within this expanded footprint amounts to ~45 ha in total. Additional allowance has been made for the relocation of certain dwellings and persons within the project footprint vicinity, as well as compensation for potential Project Affected Peoples (PAPs) affected by the ground water abstraction boreholes expansion programme.

An application for amendment of the current (Phase 1) Relocation Action Plan (RAP) should be initiated in a timely manner prior to the commencement of the expansion Phase land clearance and construction works. It is anticipated that the development and approval of the expansion RAP will require approximately 6 months to complete with the financing and compensation roll-out requiring approximately 3 months to complete. Approvals and a completed compensation process is necessary prior to accessing any lands / fields / crops affected by the Project Expansion. Timelines associated with the physical relocation of persons and dwellings will be subject to the approved programme at the time of implementation.

Tree Clearance:

Tree clearance is required in the expansion development area to accommodate the increased footprint of the accommodation camp, solar farm, tailings disposal facility, mining pit (with buffer) and mineral processing. An application for the required Tree Clearance Permit should be initiated by no later than 6 months prior to the commencement of the Molo Expansion land clearance and construction works.

20.5 Infrastructure Expansion – Permitting and Authorizations

20.5.1 Power Supply

It is envisaged that the same approach shall be followed for the Expansion relating to a total thermal generation capacity of 14 MWe, as well as solar capacity of 18MWp solar PV + 12MVA Battery Energy Storage System (BESS).

Permitting requirements will include the development and approval, (by the ONE), of a Specific Environmental Management Plan (referred to as a SEMP), which should include both the expansion of the thermal and solar components required for the Expansion. In addition, a power generation authorization is required from the Ministere de l’Energie et des Hydrocarbures. Both the SEMP and the power generation authorization applications should be initiated 6 months prior to the installation and final commissioning of the expanded power generation facilities. The solar farm footprint expansion will be included under the expansion of the Relocation Action Plan (RAP), as well as the expansion of the Land Lease area.

20.5.2 Raw Water Supply

With the raw water requirement for the Expansion being in the region of 180 m^3/h, the water supply will require the expansion of the ground water abstraction from the 3 current boreholes to approximately 25 boreholes in total, therefore, an additional 22 abstraction boreholes could be installed, depending on individual yield results. Geostratum has identified potential target locations for the borehole expansion programme.

Monetary compensation for the impact of new borehole infrastructure, including pipelines, can only be estimated at this stage and will require confirmation once target boreholes have been drilled and yield testing confirms target viability. A ground water impact assessment will confirm the zone of potential impact from the proposed well field abstraction.

Permitting requirements will include a Specific Environmental Management Plan, (referred to as a SEMP), which should include the viable borehole locations, as well as the final pipelines routing. In addition, an abstraction, (referred to locally as “uptake”), authorisation is required from the Autorite Nationale de l’Eau et de l’Assainissement (ANDEA). Both the SEMP and ANDEA applications should be initiated prior to pipeline installation and final commissioning of the ground water abstraction.

20.5.3 Waste Management

Waste handling, management and disposal will be undertaken in accordance with the following waste management hierarchy as shown in Figure 106 below.

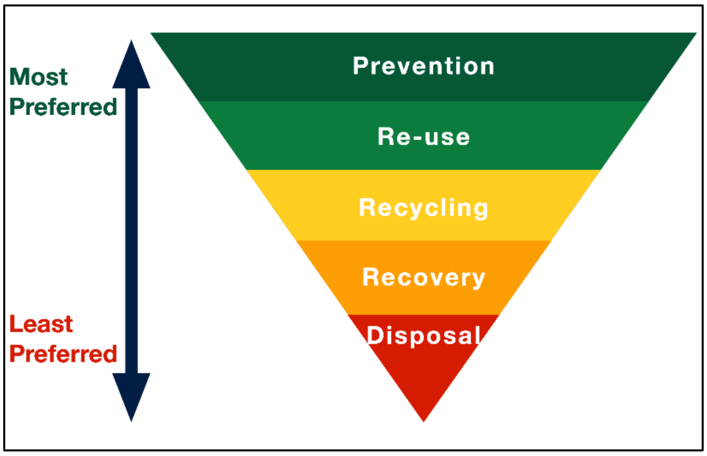


Figure 106: Waste Management Hierarchy

20.5.3.1 Domestic Waste

The expansion will utilise the landfill site that was developed and utilised during the Phase 1 operations. All materials suitable for re-use, recycling and recovery are sorted and stored within the existing salvage yard. The existing landfill site is fenced off for security and access control, has a 2m berm installed that will be concrete lined at the bottom. All non-re-usable, non-recyclable and non-recovered materials are burnt within this area.

20.5.3.2 Hazardous Waste

Medical waste will be incinerated via an incinerator which has been selected in terms of suitability with the Malagasy Ministry of Health criteria.

Sewage waste is handled via a packaged Wastewater Treatment Plant (WWTP) installed at the main camp during Phase 1. This WWTP will be upgraded to cater for the increased volumes associated with the Expansion.

Hydrocarbon waste, (used oils, greases, filters, rags etc.) will captured and disposed of according to statutory requirements.

20.5.3.3 Mining Waste and Processing Waste

The mining waste and processing waste disposal facility is to be a dry-stack facility with the upstream construction method being followed. This decision is based on the preliminary design for the TSF undertaken by ECO-Elementum (Section 18.1) that will consist of dry stacked tailings (indicated 16% to 20% moisture content), and the outer wall being a sturdy compacted waste rock wall.

Material is to be transported to the Expansion TSF site from the plant area by an overland conveyor system, (Tailings fines), and truck hauling (overburden waste rock) which will then be mechanically placed and compacted within the footprint of the TSF.

The waste rock will form the outer containment wall region with the fines being spread and compacted behind the containment wall.

Internal drainage is to be minimal and located along the lower toe regions of the TSF, the expected function of these is for drawdown of seepage that may occur from infiltration during rain events. Infiltration of rain water is to be stringently managed and minimised for the facility to remain inherently dry with no seepage.

20.5.4 Waste and Water Management

20.5.4.1 Mine Waste Assessment

A hydro-geochemical assessment was undertaken, including laboratory test work and modelling, on tailings material and representative waste rock. The results indicated that most sulphur associated with ore, tailings and waste rock consists of secondary sulphate minerals such as jarosite. The reaction rates associated with the secondary sulphate minerals are generally slow, resulting in relatively low sulphate loads in rainfall run-off and short-term operational seepage associated with the mine waste facilities. However, higher sulphate loads could be expected in the long-term, specifically post-closure.

The pH of water emanating from waste facilities will be neutral over the short term, potentially becoming slightly acidic in the long-term. Metal concentrations will be low. Please see Section 16 for more detail.

The existing geochemical assessment will be updated for Expansion, using test results of the Phase 1 tailings and waste rock material, and considering dry deposition.

Appropriate closure rehabilitation measures are foreseen to manage the long-term post closure water quality emanating from residual waste product.

20.5.4.2 Surface Water Environment

The Project site is located on the divide of three major river basins:

- Onilahy River Basin (32,361 km^2 catchment).
- Linta River Basin (6,177 km^2 catchment).
- Menarandra River Basin (8,681 km^2 catchment).

The calculated mean annual run-off is about 150 mm per annum, or roughly 18.8% of the mean annual precipitation (MAP = 799 mm).

Around the Project site, all water courses drain either to the north, or south, away from the proposed mine infrastructure with most of the proposed mine infrastructure located in the locally southward draining Linta River Basin. Water courses remain dry during the dry season and only flow during the rainy season.

Local surface water sources are largely under-developed, and no significant local water demands were identified during water use surveys.

The potential impacts on surface water from the Expansion Project will be updated.

20.5.4.3 Ground Water Environment

The Project site falls within the Ampanihy shear zone with a regional north-north-east–south-south-western trend. The local geology consists mainly of crystalline metamorphic rock such as quartz feldspathic gneiss. No significant faulting, or dyke intrusions have been identified with the site.

A number of hydrogeological boreholes were drilled and tested to investigate local aquifer conditions. The main aquifer zone is hosted within the weathered and fractures rock in the upper 50m below surface. The gneiss at the proposed mine site has low aquifer potential, with the marble layers to the west having a higher aquifer potential.

Community shallow hand dug wells and boreholes are sparsely scatters over the area. Ground water use in the larger area is low, 15m^3 to 20m^3 per day maximum.

The potential ground water impacts associated with the Phase 1 Project were mainly of low significance and site specific. Potential impacts associated with the Expansion Project will be re-assessed, taking into consideration the larger project footprint, proposed TSF co-disposal facility and higher well field abstraction.

20.5.4.4 Water Management

The potential mining related impacts for the 2015 Feasibility Study and Phase 1 Project were quantified with different ground water and surface water models and a significance rating was assigned. None of the identified potential impacts posed a high, or serious risk to the downstream environment and the majority of risks could be mitigated by the water management plan.

The impact assessment on ground water and surface water will be updated for the Expansion Project. Potential Expansion Project impacts will relate to mine dewatering, well field abstraction, dirty water run-off, run-off reduction, and potential contaminant seepage from the TSF. Apart from the Expansion Project well field, most other activities that potentially pose a risk to the water environment, have been fully, or partially investigated in the previous impact assessment studies, and will be updated accordingly to new operational methodology.

20.6 Land Tenure (Lease) Authorizations

The Molo Expansion Project development plan will require an additional ~270 ha to accommodate the expansion of the accommodation camp, solar farm, tailings disposal and mining pit. By including the land area between the existing project footprint and the Expansion TSF to the west, the total additional land lease area will reach ~430 ha.

Authorization of the expanded development footprint, in the form of what will likely be an amended Emphyteutic Lease, will be required from the Ministere de l'Amenagement du Territoire et des Services Fonciers. The application process should be initiated prior to the commencement of site establishment and construction works.

20.7 Socio-Economics

The Phase 1 Project included the implementation of the Relocation Action Plan (RAP), along with its associated 3 year Social Development Plan (SDP). In addition, the Molo Project initiated its voluntary Community Social Investment (CSI) Programme.

The Expansion Project will require additional RAP and SDP actions, for which an amendment to the Phase 1 RAP and the development of an ancillary SDP is planned. The project will continue with its voluntary CSI Programme during the Expansion.

20.8 Decommissioning and Closure Phase

The rehabilitation and closure activities associated with the domains related to the open pit, infrastructure, linear structures, mine waste, processing waste, and post-closure land use all remain as per Phase 1.

However, the conceptual Mine Rehabilitation and Closure Plan (MRCP) developed for Phase 1 has been updated to reflect the development aspects planned for the Molo expansion. The updates are centred around the following:

- A revised Bill of Quantities (BOQ).
- Updates to Unit Rates.
- Calculation of Expansion Project Mine Rehabilitation and Closure estimation.

At a conceptual level the closure plan envisages the removal and dismantling of surface infrastructure from site. This includes, but is not limited to, process plant, plant infrastructure, crusher and conveyors, site buildings, storage and other mine structures. Concrete slabs will be removed and/or covered to enable the growth of vegetation. Backfilling and levelling of ditches. At the end of the monitoring phase, if applicable, access and site roads will be scarified and revegetated.

21 CAPITAL AND OPERATING COSTS

The Molo Graphite Project is a brownfield expansion of the open-cut mine and processing plant from the initial Phase 1 ROM feed capacity of 240 ktpa to a combined 2,640 ktpa in the expanded operation.

The Phase 1 processing plant and associated infrastructure commenced with commissioning in March 2023, with first concentrate produced in June 2023. The Phase 1 plant is set to produce approximately 17 ktpa of natural flake graphite concentrate by the end of Q1 2024.

Commissioning of the expansion plant is planned for Q3Y26, which will result in a combined concentrate production capacity of approximately 150 ktpa by Q1 2027.

Tabled results are rounded for ease of legibility and, therefore, may not sum to the nearest decimal presented.

21.1 Capital Cost

21.1.1 Scope of Work

The Project's capital cost (Project CAPEX) is classified as all direct field costs, indirect field costs, and contingency incurred in enabling the expansion of the mine to a nameplate production capacity of 150 ktpa of flake graphite concentrate.

Pre-production operating costs are capitalised and reported as "Other Capitalised Costs" in the CAPEX estimate. All costs of a capital nature incurred after Commercial Production are classified as Sustaining CAPEX. Commercial production is defined as the first day following the first thirty consecutive days of the processing plant producing more than 67% of the target flake graphite concentrate of 12.5 ktpm.

No off-site infrastructure upgrades to regional roads, or bridges have been assumed. All port-related infrastructure requirements, including warehousing and laydown areas, are assumed to be provided under long-term agreements with port and logistics service providers (e.g. Velogic, Asstra).

All mine closure and rehabilitation costs have been estimated and included in the Project's Techno-Economic Model only (refer to Section 22) and have not been included in the CAPEX estimate.

Historic capital expenditure incurred prior to the Base Date are considered as sunk costs and have been excluded from the CAPEX estimate.

21.1.2 Estimate Accuracy

The Estimate is prepared in line with a Class 3 estimate as per the American Association of Cost Engineers ("AACE") Recommended Practice 47R-11.

A large proportion of the engineering deliverables are considered complete and at a suitable level of estimate maturity to support a Class 3 estimate with a target accuracy range -15% to +25% at an 80% confidence level.

Although some individual elements of the CAPEX estimate may not achieve the target level accuracy, the overall estimate falls within the parameters of the intended accuracy of a Class 3 estimate.

21.1.3 Key Assumptions

21.1.3.1 Base Date

The CAPEX estimate is stated as at 1 September 2023 (the "Base Date") and any budget estimates, or pricing provided prior to the estimate Base Date has been inflated with country-specific consumer price inflation indices.

21.1.3.2 Project Schedule

The CAPEX estimate is based on the Project satisfying all internal and external approval processes, (e.g. securing construction funding, obtaining regulatory approvals, and environmental permits) in a timely manner to maintain the Project Schedule (provided as an annexure to the report). It is also assumed that rainy season will not hamper productivity and construction progress beyond the provisions allowed for in the Project Schedule.

The project schedule has been developed using recent knowledge and experience obtained during construction of Phase 1 and using updated timing obtained from key suppliers.

The key milestones below summarise the detailed schedule and form the basis from which the estimate and cash flow were developed. (Table 84).

Table 84: Key Milestones

Milestones		
Activity	**Start Date**	**Completion Date**
Interim funding available		Month -6
Final Investment Decision		Time (0)
Detailed Engineering	Month -6	Month 1
ESIA approved	Month -6	Month 11
Permits approved	Month -12	Month 9
Offsite Modular Fabrication	Month 0	Month 12
Civil Site Works	Month 7	Month 15
Modular Construction	Month 15	Month 22
Commissioning	Month 22	Month 24
First production		Month 24

The master schedule is based on the completion of the front-end engineering and design by month 1, the ESIA and permitting by month 11. The project will utilise the same modular design approach and EPC contractor as Phase 1, who will fabricate the plant offshore from month 0 to month 12. The first plant modules will be delivered to site from month 15 and will thereafter follow the construction schedule. Onsite construction will start in month 15. Installation of the modules will occur from month 16 to month 22 and mechanical completion will occur by month 22. Commissioning and ramp-up is expected to take 2 months and be completed by month 24. To support this schedule, the Detailed Engineering works should be completed during the interim financing period, such that the Procurement and Construction process can start as soon as the final investment decision is confirmed.

21.1.3.3 Execution Plan

The CAPEX estimate is based on the appointment of an EPC contractor to design, procure, and construct the processing plant, under the guidance and management of a another EPCM contractor. The EPCM contractor will engineer and procure all non-processing related infrastructure and equipment and will manage the overarching execution of the project and the commissioning schedule.

21.1.3.4 Materials & Supplies

All earthwork and civil materials will be sourced from gravel pits or other aggregate sources located close to the project site; similar to Phase 1 construction. Excavated waste rock from the mine will be used for establishing the co-disposal tailings storage facility.

21.1.3.5 Temporary Utilities

Although the current mine operation has an established hybrid (solar / thermal) power supply system, it is assumed that all power requirements necessary for construction of the expansion project are self-supplied by the appointed contractors.

21.1.3.6 Labour Availability

Availability of skilled, semi-skilled and general hands is assumed to be sourced from the local and regional workforce.

21.1.4 Key Exclusions

The following items have been excluded from the CAPEX estimate:

- Forward-cover for inflation, escalation, and currency fluctuations.
- Management reserve for identifiable and quantifiable risk and associated mitigation plans (e.g. labour strikes, *force majeure*, etc.).
- Interest incurred during construction.
- Project financing costs.
- Closure and rehabilitation (included in the Techno-Economic Model only).
- Any sunk cost prior to the Base Date of Evaluation, (included in the Techno-Economic Model for income taxation calculation purposes only).

21.1.5 *Foreign Exchange*

The CAPEX base currency is United States dollars (US$). Most vendors (~97%) provided their pricing in United States dollars. Where quoted foreign currencies have been used in the preparation of estimates, these have been cash flowed and converted to United States dollars using the consensus forecast exchange rates as per Table 85.

Table 85: Foreign Exchange Rates

Metric	2023	2024	2025	2026	2027	LT Real
USD / MGA	4,193	4,198	4,212	4,219	4,225	4,227
USD / ZAR	17.37	17.22	17.25	17.38	17.58	17.73

21.1.6 *Key Responsibilities*

The capital costs related to the expansion of the mine, processing plant, and all supporting infrastructure have been prepared by the various external sources below and consolidated by Practara (Pty) Ltd in the Project's CAPEX estimate:

- **Mining:**

  Erudite prepared the project capital and sustaining capital for the open-cut mining operation from pit to ROM pad. The open-cut operation employs conventional excavate, drill, blast, load and haul methods, with minimal topsoil and waste stripping required to expose the flake graphite ore. Owner-mining is employed.

- **Processing:**

  SYNCS Engineering and Technology prepared the project capital estimate for the processing plant based on Engineering, Procurement, and Construction (EPC) terms. SYNCS' scope of work includes, amongst others, ore receiving and crushing, primarily mill and flash flotation, rough flotation, concentrate cleaning, drying and bagging, final tails handling and conveying, as well as all utility services, (e.g. compressed air), and support facilities, (e.g. control rooms), within the battery limits of the processing plant. It is worth noting that SYNCS was the appointed EPC contractor for the Phase 1 processing plant. Sustaining capital was factored from the direct capital cost of the plant based on industry-benchmarks and operational experience.

- **Tailings Storage Facility:**

  Eco-Elementum prepared the initial capital requirements associated with establishing the TSF footprint and starter walls, as well as the sustaining capital requirements to expand the facility in a series of 6 lifts over the LoM. The associated costs are based on a contractor rate to load, haul and compact the tailings and waste rock aggregate.

- **Support Infrastructure:**

  Erudite prepared the capital cost estimates for all bulk earthworks, civils and electrical reticulation required for the site, as well all on-site support infrastructure relating to site access and control, buildings and warehousing, accommodation camp, bulk utilities, and information and communication technology.

- **Project Management:**

  Project management costs were prepared by Erudite and SYNCS to engineer, procure, and manage the expansion project scope of work up to, and including, commissioning.

- **Owner's Cost:**

  NextSource Materials prepared the owner's cost estimate, including, but not limited to, owner's team, corporate costs, sustainable development costs, studies, permitting and insurances.

21.1.7 Estimate Structure

The estimate has been developed in accordance with the Project's Work Breakdown Structure (WBS), which is segmented into:

- Area (e.g. Processing).
- Sub-area (e.g. Comminution).
- Work package or system.
- Asset (e.g. primary mill) or element of construction.
- Commodity or trade discipline (e.g. mechanical).
- Cost type (e.g Supply).

The Level 2 WBS is provided for reference in Figure 107.

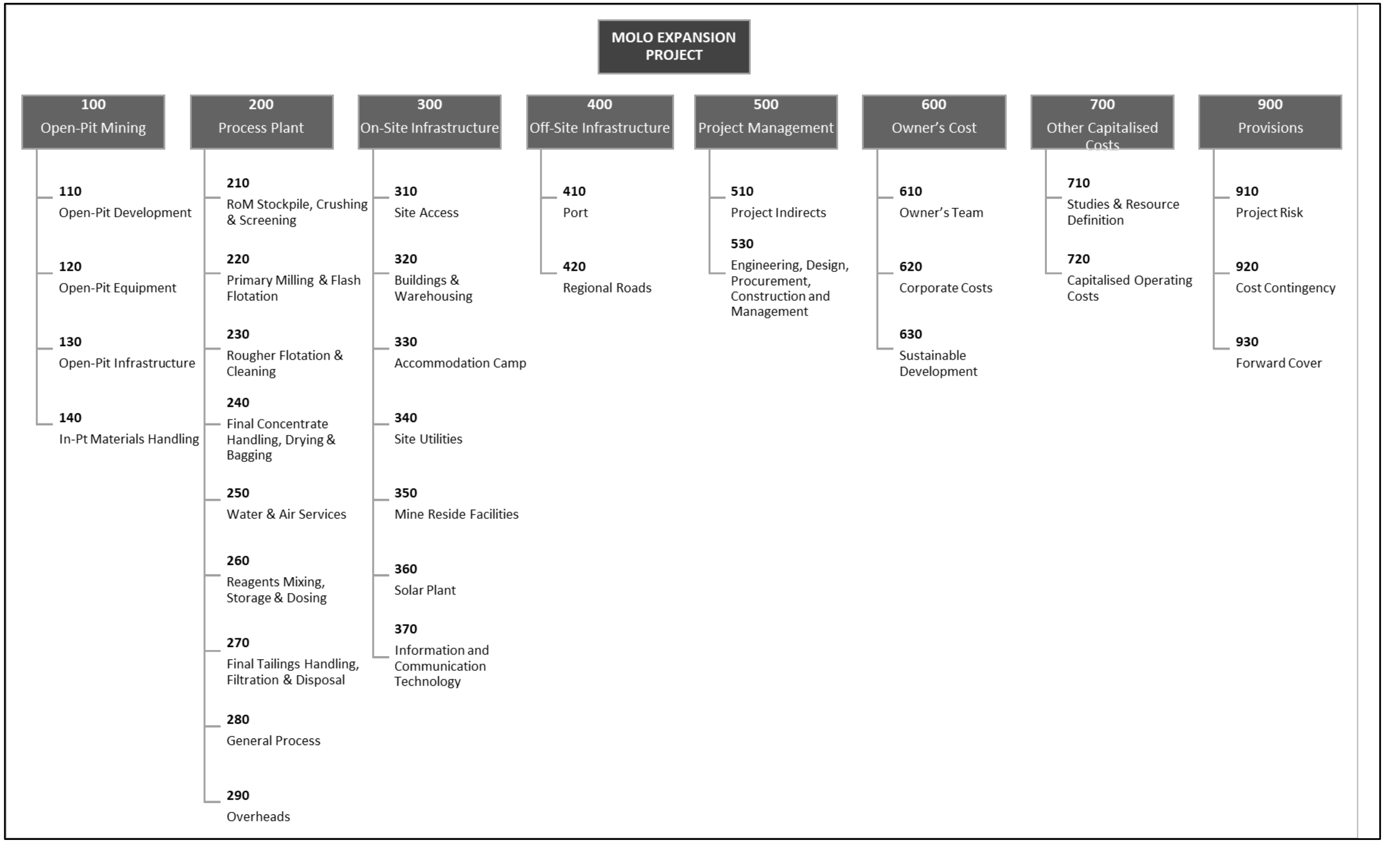


Figure 107: Work Breakdown Structure (Level 2)

21.1.8 *Area Methodology*

Quantities for the estimate are generally based on material take-offs (MTO) from engineering designs and deliverables, (e.g. site layouts, general arrangement drawings), which have been completed by the various subject matter experts. Bills of quantities (BoQs) were compiled and issued to qualified vendors and contractors for pricing.

The vendors and contractors approached for the major work packages, (e.g. processing plant, site-wide earthworks and civils, electrical, *etc.*) were all intricately involved in the Phase 1 execution project and have demonstrable experience in executing the scope of work, albeit at a smaller scale.

Unless otherwise noted, unit rates provided by the contractors are inclusive of material, transport, and direct labour to perform the scope of work.

Contractors are self-sufficient (as per the Phase 1 execution project) with no temporary utilities, fuel, accommodation, or construction support facilities provided, or free-issued, by the Project. The contractor's preliminary and general (P&Gs) expenses, therefore, includes site management, mobilisation / demobilisation, construction support equipment, temporary facilities, and associated running costs.

Freight, duties and taxes were estimated based on the Phase 1 control budget estimate and equates to approximately 10.3% of the direct capital spend.

Design development allowances have been included for specific work packages based on professional judgement applied by the greater Project team.

21.1.8.1 *Open-Pit Mining*

All mining activities for the duration of the LoM are performed on an owner-operated basis.

The Project currently mines graphite to feed the Phase 1 processing plant, with the steady-state ROM production rate of 20 ktpm (for Phase 1) scheduled to be achieved by the end of Q1 2024. Pre-production mine costs incurred from September 1, 2023 to December 31, 2024 have been capitalised in the Project's CAPEX estimate and reported as Other Capitalised Costs (Code 700). All other capital costs incurred for the Phase 1 mining project have been classified as sunk costs and are excluded from the CAPEX estimate.

The Expansion Project will increase the ROM production rate to approximately 220 ktpm by March 2027. Additional mining and service equipment to support the expansion Project have been itemised, and prices have been obtained from the current Phase 1 equipment suppliers, (*i.e.* Sany, Sinotruk, Shantui, and Taiye). A summary of the major mobile equipment procured for the expansion Project is shown below in Table 86.

Additional mine dewatering infrastructure and pit facilities have also been estimated and priced based on budget quotations received.

Pre-production mining costs incurred up until commercial production is achieved have been capitalised and reported as Other Capitalised Costs (Code 700).

Table 86: Mobile Equipment – Initial Purchases Only

Description	Current Equipment (Phase 1)	Additional Equipment	Total (Project)
Drilling Equipment:			
Drill (TAIYE X45 DTH – 25m/hr)	1	2	3
Excavating Equipment:			
Excavator (SY245H – 220 tph)	2	-	2
Excavator (SY750H – 570 tph)	-	2	2
Loading Equipment:			
Front End Loader (SYL956H5 - 3m³ Bucket)	3	-	3
Front End Loader (SYL956H6 - 4m³ Bucket)	-	3	3
Hauling Equipment:			
RDT (Sinotruk Howo 7 - 26m³)	3	11	14
Services & Support Equipment:			
Grader (Sany SMG200)	1	1	2
Dozer (Shantui SD 22W)	1	2	3
Roller (Sany SSR100C-10)	1	1	2
Water Truck (Sinotruck Howo 64290 – 20,000l)	1	1	2
Diesel Trailer (2,000l)	1	1	2
Hyundai County Bus (60 pax)	1	3	4

A summary of the mining CAPEX, excluding contingency and pre-production operating costs, is shown below in Table 87.

Table 87: Project CAPEX Summary – Open-Pit Mining (Code 100)

WBS L2 Code	WBS Level 2	WBS L3 Code	WBS Level 3	Total (US$ '000)
120	Open-Pit Equipment	121	Drilling Equipment	249
120	Open-Pit Equipment	122	Loading Equipment	1,292
120	Open-Pit Equipment	123	Hauling Equipment	888
120	Open-Pit Equipment	124	Support Equipment	809
120	Open-Pit Equipment	125	Mine Services Equipment	22
Sub Total: Open-Pit Equipment				**3,261**
130	Open-Pit Infrastructure	132	Dewatering	195
130	Open-Pit Infrastructure	134	Pit Facilities	169
Sub-Total: Open-Pit Infrastructure				**364**
Total: Open-Pit Mining (Code 100)				**3,625**

Sustaining capital was estimated on an itemised basis for the mining fleet and dewatering infrastructure, according to a planned replacement schedule based on operating hours. The total sustaining capital for the open-pit equipment and infrastructure over the LoM is estimated at US$9.2 million.

21.1.8.2 Processing

The expansion processing plant has a nameplate feed capacity of 2.4 Mtpa and will operate in parallel with the 0.24 Mtpa Phase 1 plant. It should be noted that the expansion processing plant is a standalone plant and does not involve an expansion of the Phase 1 plant.

A firm price for the engineering, procurement, construction, and commissioning of the expansion processing plant was obtained from SYNCS Engineering and Technology. Although the scope of work was priced on a lump-sum turnkey (LSTK) basis, the budget was disaggregated into estimates for supply, delivery, and installation and reported against the Project's WBS.

A high-level summary of the processing plant CAPEX, excluding first fills, critical spares, and contingency, is shown below in Table 88.

Table 88: Project CAPEX Summary – Processing (Code 200)

WBS L2 Code	WBS Level 2	TOTAL (US$ '000)
210	ROM Stockpile, Crushing & Screening	3,434
220	Primary Milling & Flash Flotation	7,154
230	Rougher Flotation & Cleaning	8,323
240	Final Concentrate Handling, Drying & Bagging	4,077
250	Water & Air Services	1,017
260	Reagents Mixing, Storage & Dosing	569
270	Final Tailings Handling, Filtration & Disposal	7,941
280	Process General	14,163
290	Overheads	11,681
Total: Processing (Code 200)		**58,359**

The sustaining capital requirements for the processing plant was factored from the plant's direct capital spend to account for major refurbishments and equipment replacements. The total sustaining capital for the processing plant is estimated at US$25.3 milion over the LoM, equating to a budget of US$1.3 M per annum.

21.1.8.3 On-Site Infrastructure

Additional site access infrastructure (code 310) was priced based on requirements for new access roads, storm water management dams, and security barracks.

Increased capacity for general offices, laboratory, workshops, ablutions, and warehouses (code 320), as well as the accommodation camp (code 330), was designed and prices obtained from a qualified vendor.

Site utilities (code 330) caters for the supply and installation of all non-process related electrical, control and instrumentation equipment and infrastructure, as well as the additional water supply, treatment, and reticulation requirements for the expanded site. Additional information and communication technology requirements relating to radio, telephony, network, and fleet tracking has been itemised and pricing obtained from a qualified vendor.

The mine residue facilities (code 350) relates primarily to the establishment of the TSF footprint, starter walls and associated water management infrastructure. It should be noted that the tailings filter cake conveyor and boom spreader is reported against Code 200 (Processing Plant).

The Project CAPEX for the hybrid solar plant (code 360) only includes the earthworks and civils for an expanded hybrid solar plant site. Power will be supplied through a power purchase agreement with CrossBoundaryEnergy ("CrossBoundary"), who procures and erects the hybrid power supply system and charges the Project a fixed tariff based on megawatt hours consumed.

Table 89: Project CAPEX Summary – On-Site Infrastructure (Code 300)

WBS L2 Code	WBS Level 2	Total (US$ '000)
310	Site Access	2,267
320	Buildings & Warehousing	3,437
330	Accommodation Camp	4,961
340	Site Utilities	8,480
350	Mine Residue Facilities	13,374
360	Solar Plant	1,155
Total : On-Site Infrastructure (Code 300)		**33,675**

Sustaining capital for on-site infrastructure (US$170.8 M) relates exclusively to the lift schedule associated with the TSF to cater for the disposal of ~58 mm^3 of co-mingled filtered plant tailings and waste rock over the LoM. It is currently assumed that a contractor would manage the load, haul, and compaction of the co-mingled tails over the LoM, which presents a substantive cost saving opportunity for the Project if it is elected to be owner-managed.

21.1.8.4 Off-Site Infrastructure

No off-site infrastructure development has been included in the CAPEX, however, the OPEX for logistics contains provisions for the supporting infrastructure required for the operation.

21.1.8.5 Project Management

Project management costs relate to project indirect costs (code 510), as well as the cost to engineer, procure, construct, manage, and commission the Project (code 520). Costs were estimated by SYNCS and Erudite respectively, based on the Project Execution Plan. The total Project Management costs of US$18.4 million translates into a factor of 19% of the direct capital costs.

Table 90: Project CAPEX Summary – Project Management (Code 500)

WBS L2 Code	WBS Level 2	WBS L3 Code	WBS Level 3	Total (US$ '000)
510	Project Indirect Cost	511	Packaging & Crating	895
510	Project Indirect Cost	512	Duties & Tariffs	4,417
510	Project Indirect Cost	513	Transport, Freight, and Warehousing	5,460
510	Project Indirect Cost	514	Commissioning Expenses	366
510	Project Indirect Cost	515	Administrative Expenses	786
510	Project Indirect Cost	519	Construction Support Equipment	1,052
Sub-Total: Project Indirect Cost				**12,976**
530	EPCM	531	Engineering and Design	2,046
530	EPCM	532	Project Management & Support	1,767
530	EPCM	533	Construction Management & Support	1,348
530	EPCM	534	Sundries	259
Sub-Total: EPCM				**5,420**
Total: Project Management (Code 500)				**18,395**

21.1.8.6 Owner's Cost

An estimate for Owner's Cost was prepared by NextSource Materials.

The estimate consists of owner's team site support (code 610) including project team during construction, operational readiness and construction phases, corporate travel and accommodation requirements, as well as a provision for general expenses.

Corporate costs (code 620) include an assessment of project insurances, performance bonds, and taxes which translates into an effective 5.5% of the direct capital spend.

Sustainable development costs (code 630) includes compensation for relocating the residences of families in the local community, performing an environmental and social impact assessment (ESIA), permit applications and approvals, and a provision for legal fees.

Table 91: Project CAPEX Summary – Owner's Cost (Code 600)

WBS L2 Code	WBS Level 2	WBS L3 Code	WBS Level 3	Total US$ '000)
610	Owner's Team	611	Project Team	5,629
610	Owner's Team	612	Operational Readiness	2,251
610	Owner's Team	613	Travel & Accommodation	1,160
610	Owner's Team	614	General Expenses	242
Sub-Total: Owner's Team Cost				**9,282**
620	Corporate Costs	622	Taxes	2,251
620	Corporate Costs	623	Financing & Insurance Charges	3,000
Sub-Total: Corporate Cost				**5,252**
630	Sustainable Development	631	Community Relations	2,267
630	Sustainable Development	633	Permitting, Approvals & ESIA	2,517
630	Sustainable Development	634	Legal	1,007
Sub-Total: Sustainable Development				**5,791**
Total: Owner's Cost (Code 600)				**20,325**

21.1.8.7 Other Capitalised Cost

Other capitalised costs include a geophysical survey of the target wellfield area, the subsequent drilling and hydrogeological analyses of the boreholes, and ad-hoc consultant support during the front-end engineering and development phase (Code 710).

Capitalised operating costs for the expansion project include first fills and critical spares, as well as pre-production operating costs incurred up until Commercial Production is achieved (Code 720).

Table 92: Project CAPEX Summary – Other Capitalised Cost (Code 700)

WBS L2 Code	WBS Level 2	WBS L3 Code	WBS Level 3	Total (US$ '000)
710	Studies & Resource Definition	711	Geophysical Surveys & Studies	116
710	Studies & Resource Definition	714	Hydrogeological Studies	710
710	Studies & Resource Definition	718	Other	1,007
Sub-Total: Studies & Resource Definition				**1,833**
720	Capitalised Operating Cost	721	Critical Spares	4,261
720	Capitalised Operating Cost	722	First Fills	3,178
720	Capitalised Operating Cost	724	Other Pre-Production Costs	21,978
Sub-Total: Capitalised Operating Cost				**29,416**
Total: Other Capitalised Costs (Code 700)				**31,249**

21.1.8.8 Provisions

The CAPEX estimate includes a provision for contingency which accounts for, amongst others, planning and estimating errors and omissions, minor price fluctuations and scope changes, and slight variations in market and environmental conditions. As such, the estimate of Contingency is expected to be expended as part of the Project's execution.

It should be noted that provisions for exchange rate fluctuations, escalation, and management reserves have been excluded from the CAPEX estimate.

The contingency estimate was prepared levering off expert judgement and predetermined contingency guidelines associated with various estimate classes as defined by AACEi. An appropriate contingency factor was derived for each work package based on the level of engineering / technical maturity of the work package, as well as the commercial basis applied in the developing the estimate.

The Base Estimate equates to US$165.6 million with contingency estimated at US$21.2 million (12.8% of the Base Estimate). The total Project CAPEX estimate, inclusive of contingency, is therefore, estimated at US$186.9 million.

21.1.9 Summary

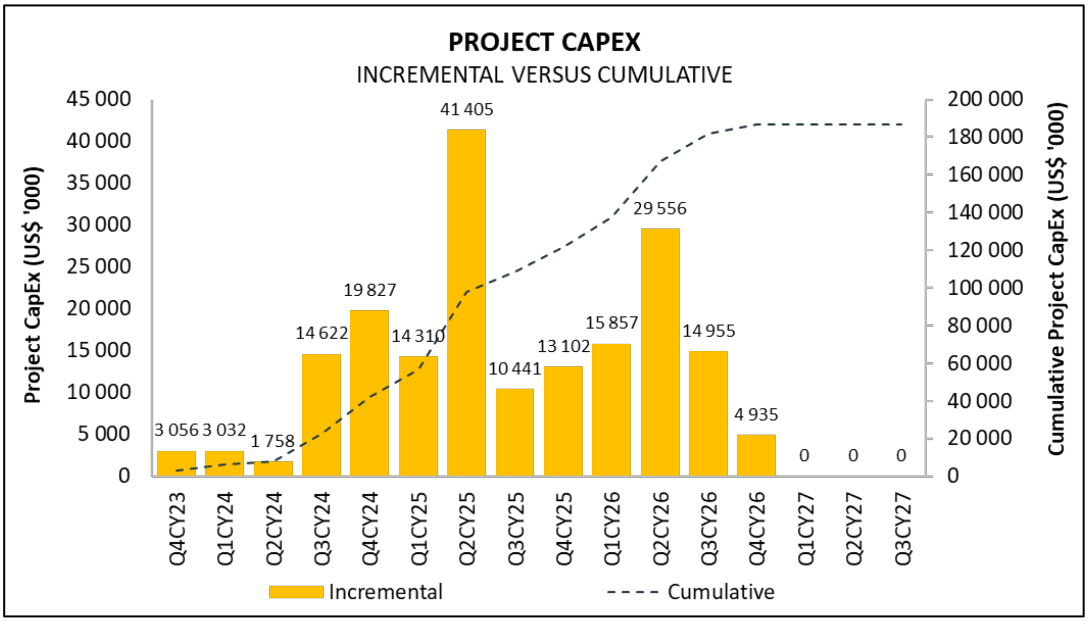
21.1.9.1 Project CAPEX

A summary of the CAPEX for the Project is shown below in Table 93. Total Project CAPEX associated with the expansion Project equates to US$186.9 million.

Table 93: Summary of Project CAPEX

Metric	UoM	Total
Direct Capital Costs	**US$ '000**	**95,659**
1000 - Mining	US$ '000	3,625
2000 - Processing	US$ '000	58,359
3000 - On-Site Infrastructure	US$ '000	33,675
Indirect Capital Costs	**US$ '000**	**69,969**
5000 - Project Management	US$ '000	18,395
6000 - Owner's Cost	US$ '000	20,325
7000 - Other Capitalised Cost	US$ '000	31,249
Provisions	**US$ '000**	**21,227**
9000 – Contingency	US$ '000	21,227
Total: Project CAPEX	**US$ '000**	**186,855**

A cash flow summary of the Project CAPEX per quarter is provided in Graph 49. The cashflow schedule is based on a high-level milestone schedule derived from the Project Master Schedule.



Graph 49: Project CAPEX Cashflow

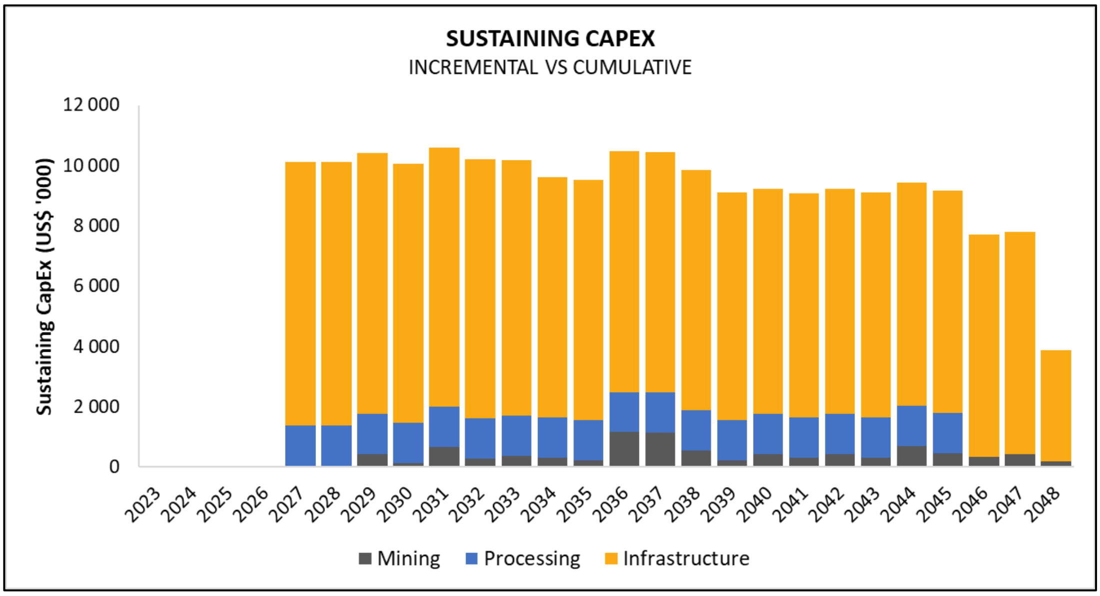
21.1.9.2 Sustaining CAPEX

Sustaining CAPEX includes all capitalised costs incurred after Commercial Production is achieved and includes the amount required periodically to maintain the designed productivity and efficiency of the mining and processing operations.

For the Molo mine, Sustaining CAPEX mainly centres on the expansion of the TSF (US$171 million), which accounts for more than 80% of the total Sustaining CAPEX estimate of US$205 million. Where the initial capital outlay for TSF caters for the development of the starter walls and associated water management infrastructure, the Sustaining CAPEX estimate includes all the scheduled lifts required to expand the TSF over the LoM.

Replacements of the mining fleet and associated infrastructure, as well as recapitalisation of the processing plant is estimated at US$9.2 M and US$25.3 million respectively.

The Sustaining CAPEX cashflow profile is shown in Graph 50 below.



Graph 50: Sustaining CAPEX Cashflow

21.1.9.3 Closure & Rehabilitation Costs

The closure and rehabilitation costs are not included in the CAPEX estimate but has been cash-flowed in the Project's Techno-Economic Model based on the Project securing a typical mine closure guarantee product.

The guarantor provides a guarantee to the regulating authorities that the Project has the financial means to close the mine, although the mine remains the primary entity responsible for mine closure and rehabilitation. To secure this facility, the Project pays 40% of the total closure liability to the guarantor on the day commercial production is achieved, with the balance of the total closure liability (60%) amortised over a 10 year period in equal instalments at an annual interest rate of 3%.

Independent consultancy GlobeSight estimated the total closure and rehabilitation cost for the Project at US$11.7 million, which translates into a total closure provision of US$12.8 million inclusive of the 3% financing charge.

The estimate is based on plot plans and layouts and includes:

- Closure of the open-cut mine.
- Dismantling and removal of all facilities.
- Promotion of rapid regeneration and renewal of native plant species.
- The closure cost estimate excludes a provision for the retrenchment of staff, (included in the operating cost estimate).
- No salvage value of equipment has been considered in the closure costs.

21.2 Operating Cost

21.2.1 Scope of Work

The operating cost estimate has been developed in line with the current phased development approach for the Project, which sees production increasing from the current Phase 1 production capacity of 17 ktpa of natural flake graphite concentrate to circa 150 ktpa during the expansion project.

The operating costs forecast developed for Phase 1 is based on the current medium-term production plan and accompanying budget provided by the Company, which has been independently reviewed by Practara (Pty) Ltd and incorporated into the Project's operating cost (OPEX) estimate.

The OPEX estimate for the expanded operation has been prepared by various external sources, leveraging extensively from information on the current operations and existing supply contracts, which has been consolidated by Practara (Pty) Ltd into the Project's OPEX estimate.

The Project's concentrate is sold on a FOB basis, as such the OPEX estimate includes all on-site cash costs incurred by the Project from pit to port. Head-office allocated costs have also been assessed and included in the OPEX estimate.

21.2.2 Key Exclusions

The following items have been excluded from the OPEX estimate:

- Forward-cover for inflation, escalation, and currency fluctuations.
- Closure and ongoing rehabilitation costs (included in the Techno-Economic Model only).
- Freight and product insurance costs (product is sold on a FOB basis).
- Market agency fees, (included in the Techno-Economic Model only).
- Government royalties, (included in the Techno-Economic Model only).
- Other royalty agreements, (included in the Techno-Economic Model only).
- Contingency.

21.2.3 Foreign Exchange

The OPEX estimate has modelled the unit prices and cost rates in the native currencies as received from the respective vendors.

A large proportion of these unit prices and cost rates have been provided in United States Dollars (US$), to the extent that 88% of the Project's OPEX is denominated in US$. The balance is made up from Malagasy Ariary (7%) and South African Rand (5%), which have been converted for cashflow purposes at the long-term real FX rates documented in Table 85.

21.2.4 Estimate Structure

Costs are captured in a generic coding structure, which enables the reporting of operating costs per:

- Main activity (e.g. Mining).
- Sub-activity (*e.g.* Drilling).
- Cost element (e.g. Labour).

An abridged estimate structure is provided in Figure 108.

21.2.5 Cost Element Methodology

21.2.5.1 Labour

An organogram was developed to support the steady state labour compliment for the Project of ~450 people (refer Figure 109). Roles were specified per operating area and allocated to a unique shift configuration applicable to that role.

Each role has been earmarked to either a local employee (LRE), or an expatriated employee (ERE) based on an assessment of available in-country skills. 89% of the permanently employed workforce are expected to be LREs.

A Paterson grading was applied to each role for the purposes of allocating the appropriate ERE / LRE labour cost rate. Labour cost rates were derived from the current Phase 1 project, with the cost-to-company for each role split into a basic salary, benefits, and on-costs. It should be noted that Madagascar is one of the poorest countries in the world, falling into the 5th percentile of GDP per capita (S&P Global, 2022) and, as such, a significant disparity between LRE and ERE labour rates are observed.

Estimates for personal protective equipment (PPE) as well as periodic medicals are catered for as fixed overheads in the site-based general and administrative costs and are excluded from the labour cost.

An annualised headcount for the Project is shown in Figure 109. It should be noted that, as the pit deepens towards the end of the LoM, the mining operation transitions from a daily 2 shift system to a 3 shift system to sustain the ROM production profile of 2.64 Mtpa. This results in an increased mine labour requirement over that period.

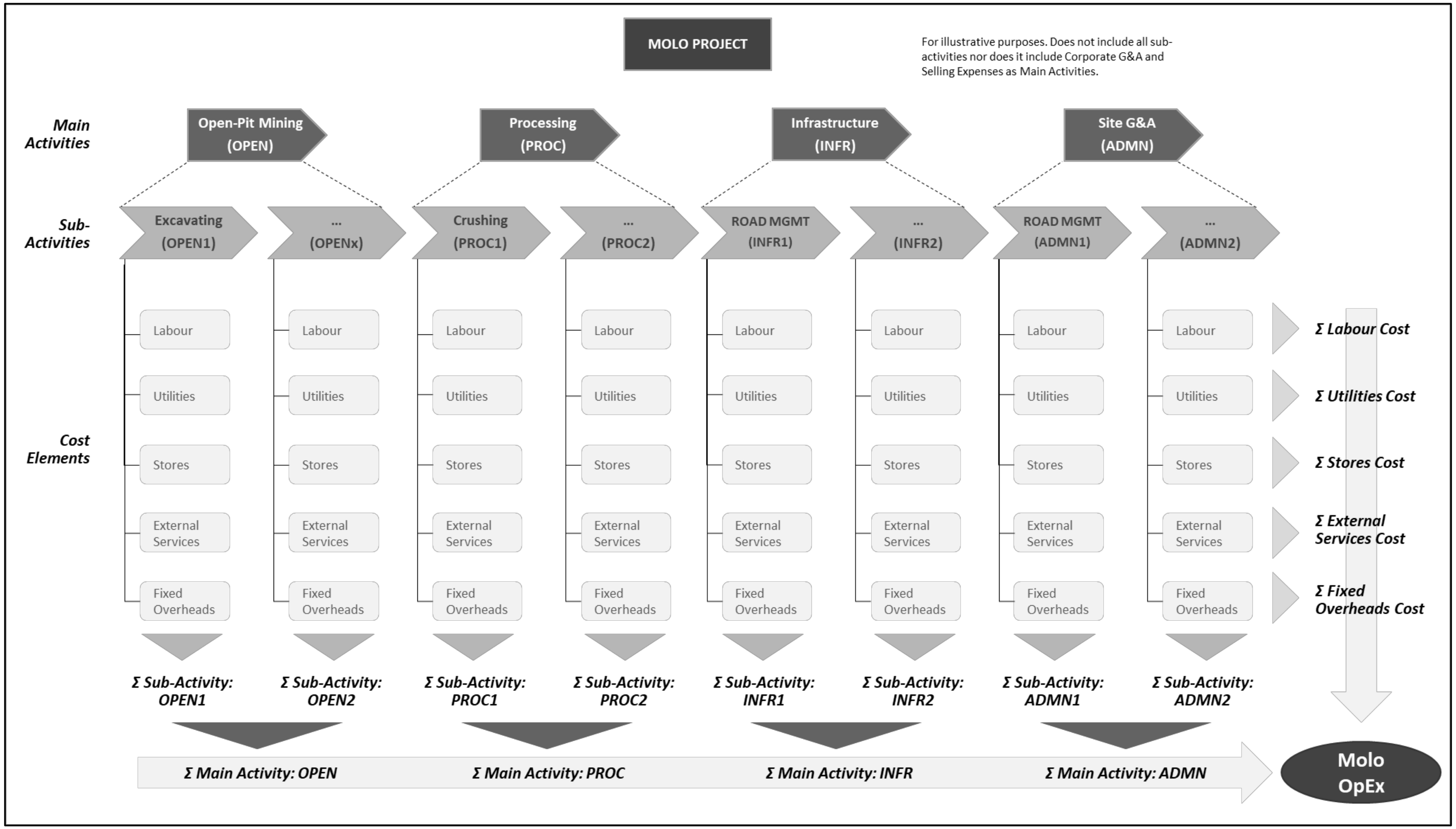


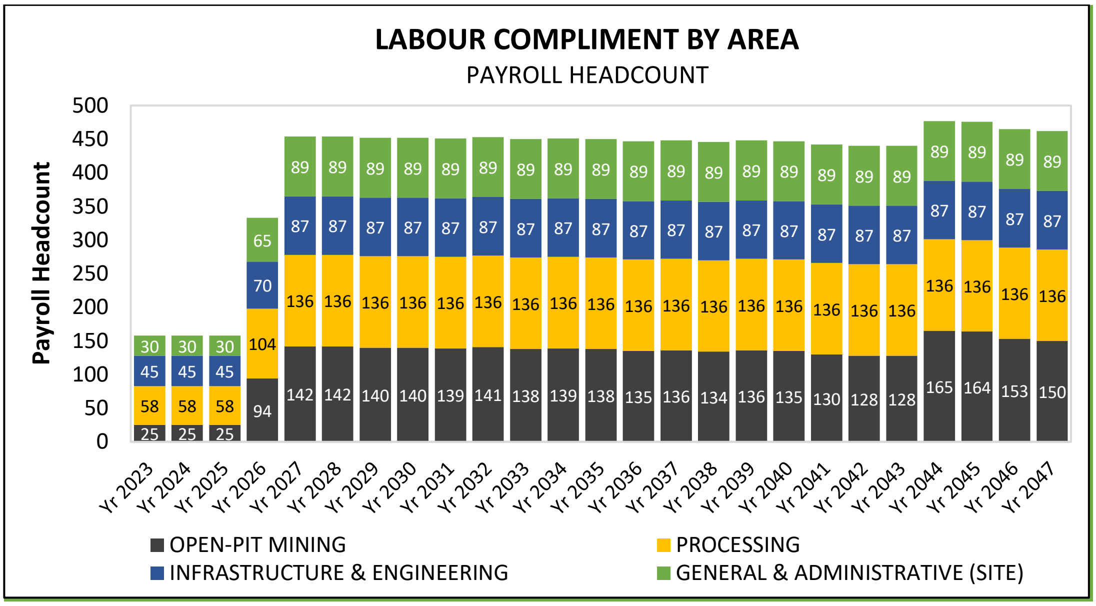
Figure 108: Activity Structure and Cost Element Structure

OPEN-PIT MINING	#
Mining Superintendent	1
Mining Shift Foreman	3
Operator: FEL	9
Operator: Excavator	9
Operator: Drill	6
Drill Assistants	6
Operator: RDT	36
Operator: Grader	3
Operator: Dozer	9
Operator: Compactor	3
Operator: Water Truck	3
Operator: Diesel Bowser/Trailer	3
Mining Services Foreman	1
Services Assistants	4
Operator: Multi-skilled relief	6
Blasting Technician	2
Blasting Assistants	4
Magazine Master	2
Maintenance Shift Foreman	2
Electricians	3
Diesel Mechanics	3
Fitters	3
Boilermaker / Welder	2
Vehicle & Tyre Supervisor	1
Vehicle & Tyre Assistants	3
Maintenance Planners	1
Maintenance Technicians	3
Mining Technical Manager	1
Technical Manager Assistant	1
Surveyor	1
Survey Assistant	2
Geologist	1
Geological Technician	2
Geological Grade Control	2
Mine Planning Engineer	1
Mine Planning Clerk	2

PROCESSING & ENGINEERING	#
Plant Manager	1
Site Administrator	1
Lab Superintendent	2
Lab Analysts	4
Lab Attendants	4
Lab Assistants	8
Plant Engineer	1
GES	1
Mechanical Foreman	1
Rigger	1
Rigger Assistants	1
Crane Operators	4
Fitter/Millwright	6
Millwright Assistants	6
Diesel Mechanic	2
Boilermaker	4
Boilermaker Assistants	4
E,C&I Foreman	1
Electrician	2
E&I Assistant	2
Instrument Mech	2
Instrument Tech	1
Asset Care Manager	1
Maintenance Planner	1
Maintenance Data Capturer	2
Plant Metallurgist	1
Jnr Metallurgist	1
Control Room Operators	8
Process Superintendent	1
Production Superintendent	1
Shift Superintendent	4
Process Supervisors	16
Process Operators	16
Process Attendants	20
Tailings Engineer (Civil & Structural)	1
Shift Superintendent (CDF)	4

GENERAL & ADMINSTRATIVE	#
General Manager	1
GM Assistant	1
Shared Services Manager	1
Shared Services Manager Assistant	1
Camp Manager	1
Camp Supervisor	1
Camp Staff	40
Finance Manager	2
Accountants	6
Finance Admin Clerk	3
Accounts Clerk	3
Payroll Clerk	3
HR Manager	1
HR Superintendent	1
HR Officer	3
Recruiter	1
Travel Department	2
SHEQ Manager	1
Safety Superintendent	2
SHEQ Co-ordinator	1
Assist. SHEQ Co-ordinator	1
Safety Officer	6
Paramedic	3
Doctor	1
Clinical Officer / Nurse	2
Training Officer	3
Environmental Superintendent	1
Environmental Officer	2
Community Relations Manager	1
Community Liaison Representatives	2
Community Admin Clerk	1
IT Superintendent	1
IT Technician	3
Procurement Superintendent	1
Procurement Supervisor	2
Procurement Officer	4
Store Supervisor / Controller	4
Store Attendants	8
Contracts Manager	2
Contracts Administrators	2
Security Superintendent	1
Security Supervisor	3
Security Guards	32
Logistics Manager	1
Logistics Manager Assistant	1
Road Transport Drivers	4
Bus Operators	4

Figure 109: Organogram for Expanded Operation



Graph 51: Labour Head Count

A summary of the labour cost per operating area, excluding capitalised operating cost, is shown in Table 94. Labour cost contributes ~9% of the total site cash cost.

Table 94: OPEX Summary – Labour

Main Activity	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Open-Pit Mining	25,880	0.46	8.36
Processing	33,696	0.60	10.89
Infrastructure	11,684	0.21	3.78
Site G&A	37,415	0.66	12.09
Corporate G&A	-	-	-
Concentrate Transport	-	-	-
Total: Labour Cost	**108,674**	**1.93**	**35.12**

21.2.5.2 Utilities

Power costs are based on load lists per area that are built-up from individual motor control centres (MCCs), with draw rates applied per MCC to derive the effective power consumption.

The hybrid solar plant delivers up to 30% of the total power requirements for the Project, with the balance of power supplied by diesel gensets. The average power consumption for the expanded operation is ~92 GWh per annum.

The effective power tariffs are based on definitive power supply agreement for a hybrid solar and thermal generation solution from the independent power producer (IPP) Cross Boundary Energy. The IPP tariff equates to US$79/MWh, excluding the cost of diesel to operate the diesel gensets, which, if included, increases the effective tariff to US$264/MWh. It should be noted that the diesel cost to operate the gensets are included in the Stores costs of the OPEX estimate.

Water supply costs are based on the establishment of a wellfield with 22 boreholes and associated pumping infrastructure to support the additional raw water requirements of the Project. As such, no water is procured "over-the-fence". Any costs incurred in managing the water and sewerage treatment plants are included in the Stores cost of the OPEX estimate.

A summary of the Utilities cost per operating area, excluding capitalised operating cost, is shown in Table 95. Utilities cost contributes ~13% of the total site cash cost.

Table 95: OPEX Summary - Utilities

Main Activity	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Open-Pit Mining	743	0.01	0.24
Processing	150,530	2.68	48.65
Infrastructure	10,024	0.18	3.24
Site G&A	-	-	-
Corporate G&A	-	-	-
Concentrate Transport	-	-	-
Total: Utilities Cost	**161,297**	**2.87**	**52.13**

21.2.5.3 Stores

All operating and maintenance consumable costs for the Project were prepared by Erudite and NextSource Materials based on detailed calculations and unit costs obtained from existing supplier contracts entered into by the Project:

- **Mining:**

  Mining stores costs for the fleet and support equipment were calculated based on planned operating hours for each piece of equipment. Fuel burn and other key consumable rates (e.g. ground engagement tools, tyres / tracks, oil, maintenance, *etc.*) were either provided by the original equipment manufacturers (OEMs), or benchmarked against similar equipment from other OEMs. Blasting consumables were estimated from the blast design, which specifies the required blasting consumables, (e.g. ANFO, down-hole delays, surface connectors, boosters, *etc.*), to support the required number of planned blasts per month.

- **Processing:**

  Processing stores costs for key consumables, (e.g. liners, steel balls, reagents, diesel, etc.) are derived from modelled dosage and consumption rates supported by the process design criteria, flowsheet, and mass balance. An annual planned maintenance budget for the plant was also derived and has been included in the OPEX estimate in accordance with a monthly maintenance schedule. Laboratory costs are extrapolated from the current Phase 1 budget, based on the grade control, circuit sampling, and concentrate sampling protocols for the Project.

- **Infrastructure:**

  Infrastructure stores costs are primarily driven by the diesel consumption rate of the gensets (220 L/MWh) to support the hybrid power supply system. Consumable costs for water and sewerage treatment, as well as camp operating and maintenance consumables have been included in the Stores cost for the Project based on calculated estimates.

- **G&A:**

  Miscellaneous stores costs relating to environmental monitoring, medical supplies, PPE, and office consumables, have been budgeted monthly and reported against Site-based G&A.

The supply of diesel fuel is a major input cost for the Project. The Project is negotiating a long-term supply agreement of Light Fuel Oil (LFO), with a leading petroleum supply and distribution company in Madagascar. The agreement sees the supplier responsible for installing and commissioning a fuel depot on-site and guaranteed delivery of the required LFO 5 days after order placement. The delivered LFO rate is linked to the Platts Gasoil FOB Arab Gulf price, which has been highly volatile since the onset of the conflict between Russia and Ukraine. Despite this volatility, the Project has adopted a fixed long-term supply rate of US$1.20/L (delivered), which correlates to a Platts Gasoil price of US$100/bbl (FOB Arab Gulf). As such, material upside to the Project could be realised if oil prices were seen to revert to levels seen pre-2021.

A summary of the Stores cost per operating area, excluding capitalised operating cost, is shown in Table 96. Stores cost contributes ~67% of the total site cash cost.

Table 96: OPEX Summary - Stores

Main Activity	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Open-Pit Mining	158,105	2.81	51.10
Processing	268,782	4.78	86.87
Infrastructure	379,249	6.74	122.58
Site G&A	10,410	0.19	3.36
Corporate G&A	-	-	-
Concentrate Transport	-	-	-
TOTAL: Stores Cost	**816,545**	**14.51**	**263.91**

21.2.5.4 External Services

The following key externally contracted services have been budgeted for in the OPEX estimate:

- Concentrate logistics and warehousing.
- Catering.
- Auditing.
- Tax compliance.
- Banking fees.
- Transportation.
- Recruitment fees.
- Specialist consultant support.

Budget quotes were obtained for the major contracted services, (i.e. concentrate logistics and warehousing and catering), which account for more than 90% of the total external services cost for the Project.

A summary of the External Services cost per operating area, excluding capitalised operating cost, is shown in Table 97. It should be noted that concentrate transport and warehousing is reported as Selling Expenses for the purposes of calculating NSR-based royalty payments in the Project’s Techno-Economic Model.

Table 97: OPEX Summary – External Services

Main Activity	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Open-Pit Mining	3 036	0.05	0.98
Processing	-	-	-
Infrastructure	17,007	0.30	5.50
Site G&A	39,547	0.70	12.78
Corporate G&A	-	-	-
Concentrate Transport	446,373	7.93	144.27
TOTAL: External Services Cost	**505,962**	**8.99**	**163.53**

21.2.5.5 Fixed Overheads

Fixed Overheads for the Project are based on estimated cost for items such as:

- Regional offices in Fortadrevo, Fort Dauphin, Tulear and Antananarivo.
- Software licenses and fees for mine planning, process control, ERP, and accounting:
- Land leases.
- All-risk insurance.
- Community projects.
- Internet and telephony access.
- Head-office allocations.

A summary of the Fixed Overheads cost per operating area, excluding capitalised operating cost, is shown in Table 98.

Table 98: OPEX Summary – Fixed Overheads

Main Activity	LoM Total (US$ '000)	Unit Cost (US$ / t ROM)	Unit Cost (US$ / t concentrate)
Open-Pit Mining	2,234	0.04	0.72
Processing	761	0.01	0.25
Infrastructure	12,515	0.22	4.04
Site G&A	53,076	0.94	17.15
Corporate G&A	16,440	0.29	5.31
Concentrate Transport	-	-	-
Total: Fixed Overheads Cost	**85,027**	**1.51**	**27.48**

21.2.6 Summary

A summary of the LoM OPEX per Main Activity is shown below in Table 99. The total site cash cost equates to US$393/t of concentrate, which increases to US$542/t on inclusion of the concentrate transport and corporate G&A costs.

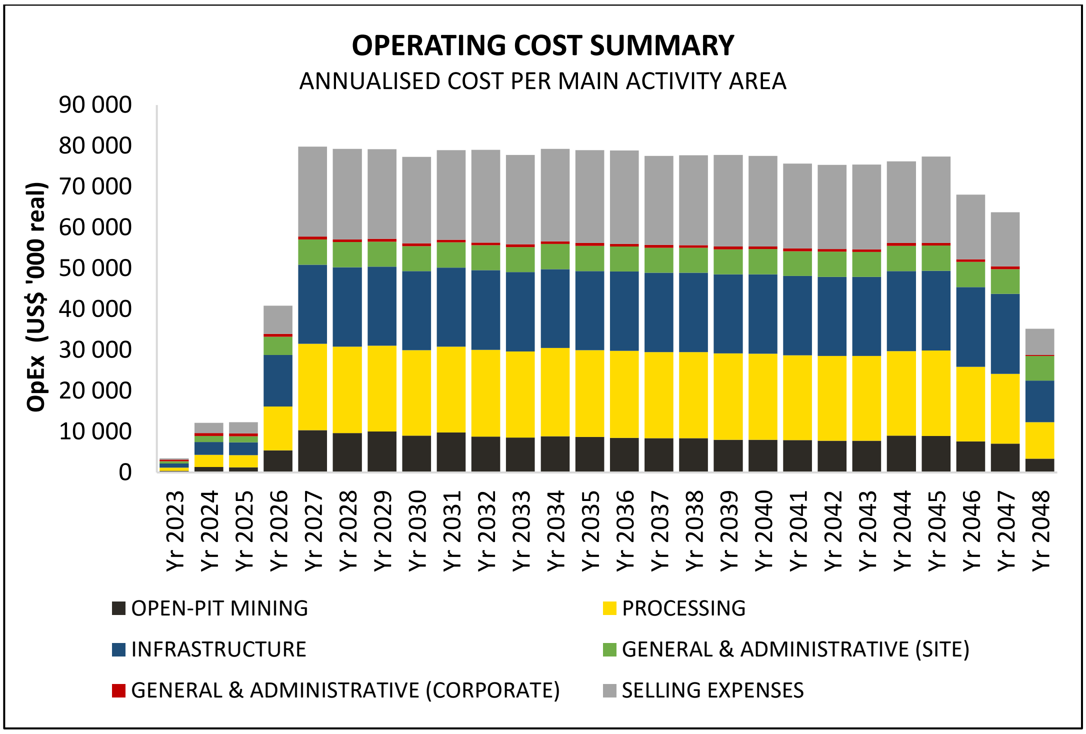
All mineral royalties and market agency fees payable by the Project are excluded from the results presented below and only assessed in the Project's Techno-Economic Model (refer Section 22).

Table 99: OPEX Summary per Main Activity Area

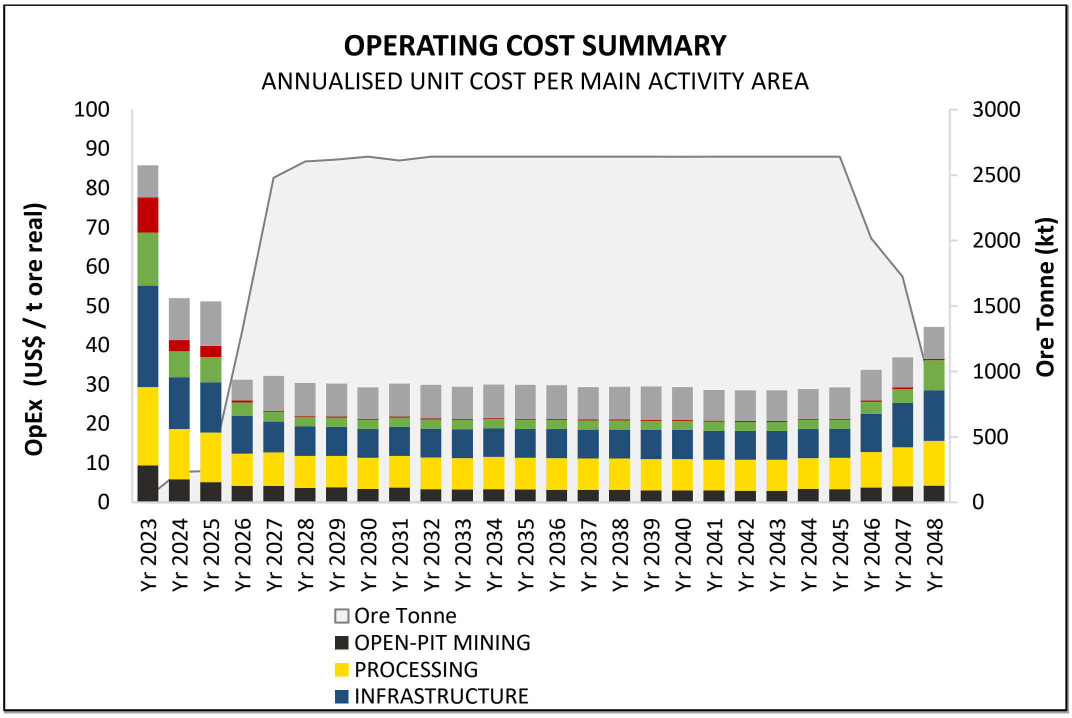
Sub-Activity	LoM Total (US$ '000)	Unit Cost (US$ / t ROM)	Unit Cost (US$ / t concentrate)
Open-Pit Mining	189,997	3.38	61.41
Processing	453,769	8.06	146.66
Infrastructure	430,478	7.65	139.13
Site G&A	140,448	2.50	45.39
Sub-Total: Site Cash Cost	**1,214,692**	**21.59**	**392.59**
Corporate G&A	16,440	0.29	5.31
Concentrate Transport	446,373	7.93	144.27
Total: OPEX	**1,677,505**	**29.81**	**542.18**

Total OPEX excludes Royalties and Marketing Fees, which is accounted for in the Project's TEM.

The annualized operating cost per main activity area is presented in Graph 52, and the equivalent unit cost per tonne ore processed is illustrated in Graph 53.



Graph 52: OPEX Cashflow per Main Activity Area



Graph 53: OPEX Unit Cost Summary per Main Activity Area

21.2.6.1 Open-Pit Mining

Open-pit mining is performed on an owner-operated basis with the unit cost averaging US$2.52/t rock mined over the LoM. Due to the low-strip ratio and relatively short haul distances, open-pit mining accounts for a relatively small proportion (~16%) of the total site cash cost. Site cash cost includes all operational expenses on-site, but excludes Corporate G&A and selling expenses.

A summary of the Mining OPEX per cost element and sub-activity is provided in Table 99Table 100 and Table 101 respectively.

Table 100: OPEX by Cost Element - Open-Pit Mining

Cost Element	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t rock mined)
Labour	25,880	0.46	0.34
Utilities	743	0.01	0.01
Stores	158,085	2.81	2.09
External Services	3 036	0.05	0.04
Fixed Overheads	2,234	0.04	0.03
TOTAL: Open-Pit Mining Cost (excluding Capitalised OPEX)	**189,977**	**3.38**	**2.52**

Table 101: OPEX by Sub-Activity- Open-Pit Mining

Sub-Activity	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t rock mined)
Excavating	11,494	0.20	0.15
Drilling	7,504	0.13	0.10
Blasting	69,390	1.23	0.92
Loading	3,977	0.07	0.05
Hauling	37,874	0.67	0.50
Rehandling	5,667	0.10	0.08
Support Services	25,245	0.45	0.33
Engineering	5,451	0.10	0.07
Technical	14,358	0.26	0.19
Management	12,012	0.21	0.16
TOTAL: Open-Pit Mining Cost (including Capitalised OPEX)	**192,972**	**3.43**	**2.56**
[-] Capitalised OPEX	(2,995)	(0.05)	(0.04)
TOTAL: Open-Pit Mining Cost (excluding Capitalised OPEX)	**189,977**	**3.38**	**2.52**

21.2.6.2 Processing

The Processing unit cost is estimated US$8.06/t processed over the LoM and accounts for 37% of the total site cash cost.

A summary of the Processing OPEX per cost element and sub-activity is provided in Table 102 and Table 103 respectively. It should be noted that the Utilities cost of US$2.68/t processed excludes the cost of diesel to power the gensets, which is reported against the Infrastructure Stores cost.

Table 102: OPEX by Cost Element - Processing

Cost Element	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Labour	33,696	0.60	10.89
Utilities	150,530	2.68	48.65
Stores	268,782	4.78	86.87
External Services	-	-	-
Fixed Overheads	761	0.01	0.25
TOTAL: Processing Cost (excluding Capitalised OPEX)	**453,769**	**8.06**	**146.66**

Table 103: OPEX by Sub-Activity – Processing

Sub-Activity	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Crushing & Screening	21,533	0.38	6.96
Milling	69,216	1.23	22.37
Flotation	120,478	2.14	38.94
Concentrate Cleaning	27,371	0.49	8.85
Concentrate Drying	90,387	1.61	29.21
Concentrate Handling	33,583	0.60	10.85
Tailings Handling	37,563	0.67	12.14
Services	39,114	0.70	12.64
Engineering	10,585	0.19	3.42
Management	9,231	0.16	2.98
TOTAL: Processing Cost (including Capitalised OPEX)	**459,062**	**8.16**	**148.37**
[-] Capitalised OPEX	(5,293)	(0.09)	(1.71)
TOTAL: Processing Cost (excluding Capitalised OPEX)	**453,769**	**8.06**	**146.66**

21.2.6.3 Infrastructure

The Infrastructure unit cost is estimated US$7.65/t processed over the LoM and accounts for 35% of the total site cash cost. Infrastructure OPEX is primary driven by the fuel required to power the on-site diesel gensets, which accounts for ~88% of the total cost over the LoM.

A summary of the Infrastructure OPEX per cost element and sub-activity is provided Table 104 and Table 105 respectively.

Table 104: OPEX by Cost Element – Infrastructure

Cost Element	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Labour	11,684	0.21	3.78
Utilities	10,024	0.18	3.24
Stores	379,249	6.74	122.58
External Services	17,007	0.30	5.50
Fixed Overheads	12,515	0.22	4.04
TOTAL: Infrastructure Cost (excluding Capitalised OPEX)	**430,478**	**7.65**	**139.13**

Table 105: OPEX by Sub-Activity – Infrastructure

Sub-Activity	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Power Supply (Diesel Only)	384,362	6.83	124.23
Water Treatment	1,506	0.03	0.49
Buildings	1,490	0.03	0.48
Sewerage Treatment	4,419	0.08	1.43
Camp Operations	25,739	0.46	8.32
Site Communication	14,304	0.25	4.62
Security	6,253	0.11	2.02
TOTAL: Infrastructure Cost (including Capitalised OPEX)	**438,073**	**7.79**	**141.59**
[-] Capitalised OPEX	(7,595)	(0.13)	(2.45)
TOTAL: Infrastructure Cost (excluding Capitalised OPEX)	**430,478**	**7.65**	**139.13**

21.2.6.4 Site G&A

The Site G&A OPEX totals US$140.4 million over the LoM, which equates to an average budget of US$6.1 million per annum for the expanded operation. Site G&A accounts for 12% of the total site cash cost.

A summary of the Site G&A OPEX per cost element and sub-activity is provided in Table 106 and Table 107 respectively.

Table 106: OPEX by Cost Element – Site G&A

Cost Element	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Labour	37,415	0.66	12.09
Utilities	-	-	-
Stores	10,410	0.19	3.36
External Services	39,547	0.70	12.78
Fixed Overheads	53,076	0.94	17.15
TOTAL: Site G&A Cost (excl Capitalised OPEX)	**140,448**	**2.50**	**45.39**

Table 107: OPEX by Sub-Activity – Site G&A

Sub-Activity	LoM Total (US$ '000)	LoM Ave Unit Cost (US$ / t ROM)	LoM Ave Unit Cost (US$ / t concentrate)
Accounting, Finance, Audit & Tax	55,242	0.98	17.85
Human Resources	9,392	0.17	3.04
Procurement	6,781	0.12	2.19
Logistics & Warehousing	2,895	0.05	0.94
Transport	7,659	0.14	2.48
SHEQ	20,791	0.37	6.72
Community Relations	6,112	0.11	1.98
General Management	34,056	0.61	11.01
TOTAL: Site G&A Cost (including Capitalised OPEX)	**142,928**	**2.54**	**46.20**
[-] Capitalised OPEX	(2,480)	(0.04)	(0.80)
TOTAL: Site G&A Cost (excluding Capitalised OPEX)	**140,448**	**2.50**	**45.39**

21.2.6.5 Corporate G&A

Corporate G&A cost is allocated to the Project as a fixed overhead to account for internal audits, corporate reporting, and periodic corporate travel. The costs total US$16.4 million over the LoM, which equates to an average budget of US$670k per annum for the expanded operation and makes up ~1% of the Project's all-in-sustaining cost (FOB Madagascar).

21.2.6.6 Concentrate Transport

Concentrate Transport include the logistics and warehousing costs from mine to port applicable to the sale of natural flake graphite concentrate. The LoM average unit cost equates to US$144/t concentrate, which equates to ~24% of the Project's all-in-sustaining cost (FOB Madagascar).

22 ECONOMIC ANALYSIS

22.1 Introduction

This section revolves around the economic analysis and investment evaluation of the combined, Phase 1 and expansion, TEM for the Molo deposit prepared for NextSource Materials Inc., which encapsulates the following key aspects:

- A statement of and justification for the principal inputs and assumptions applied in the TEM.
- A review of the key project drivers (ore production, mass yield and recovery, CAPEX and OPEX) developed by the various subject matter experts in support of this FS.
- A tabulated summary and graphical representation of the forecast LoM free cash flow per annum.
- A summary of the regulatory costs forecast on the basis of the legislation and regulations currently enacted in Madagascar at the date of this report, which largely pertain to governmental mineral royalties, mine rehabilitation and closure costs, and corporate income tax.
- A summary of royalty interests in the project, namely the Net Smelter Return royalty and the royalty held by Vision Blue Resources.
- A summary and analysis of the key business return metrics, which include NPV, IRR, payback period and the peak funding requirement.
- An analysis of the key results metrics' sensitivity to movements in key inputs and assumptions such as commodity prices, foreign exchange rates, grade, CAPEX and OPEX and discount rate.

The economic analysis presented in this chapter contains forward-looking information with regards to the commodity prices, foreign exchange rates, proposed mine production plan, projected mass yield and recovery ratios, CAPEX and OPEX costs. The results of the economic analysis are subject to various known and unknown risks, uncertainties and other factors that cause actual results to differ materially from those presented here.

22.2 Basis of Evaluation

The investment evaluation principles applied in the economic analysis are aligned to best practices for the evaluation of mineral projects at a FS level of accuracy.

A detailed TEM was developed in Microsoft Excel to analyse the economic feasibility of the Project.

The real, post-tax, unlevered, (i.e. assumed to be financed 100% by equity), undiscounted free cash flows generated by the TEM are used to determine the IRR, payback period and peak funding requirement of the Project. These cash flows are discounted at a real discount rate to determine the NPV of the project.

Reporting of key results metrics has also been provided on a pre-tax basis.

The economic analysis and accompanying free cash flow profile generated by the TEM have been presented in United States dollars, which are aligned to the presentation and functional currency of NEXT.

Cost estimates have been prepared in the currency in which they are incurred. Commodity prices are based on forecasts from an expert market research firm and foreign exchange rates are based on consensus forecasts.

The sensitivity analyses have been performed on a post-tax basis to assess the impact of variations in the commodity price, foreign exchange rates, grade, CAPEX and OPEX and discount rate.

Table 108 lists the basis of evaluation assumptions associated with the Project.

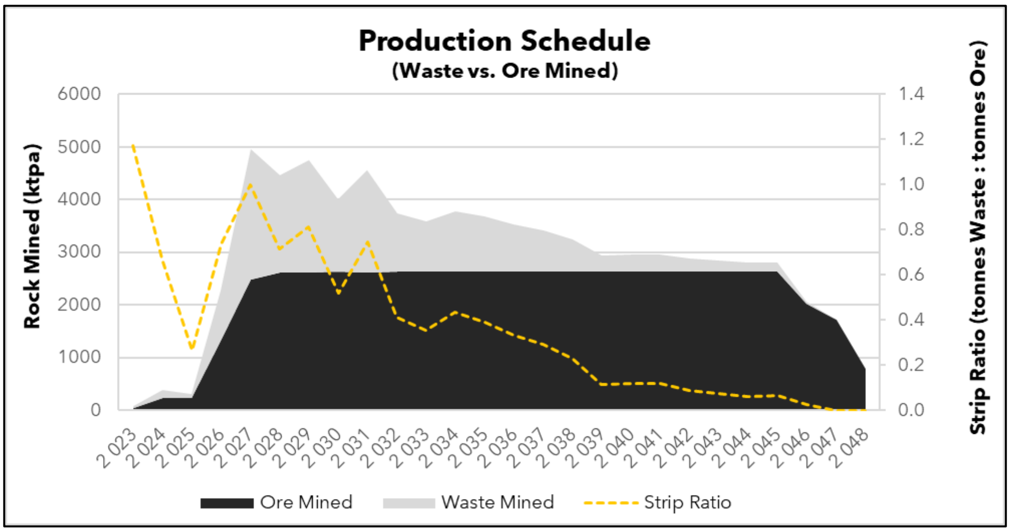
Table 108: Basis of Evaluation Assumptions

Factor	Assumption
Method of Analysis	Discounted Cash Flow Analysis
Cash Flow Terms	Real Terms
Base Currency	United States Dollar (US$)
Base Date of Evaluation	September 1, 2023
Discount Rate	8.0% (US$ Real)

22.3 Physical Drivers

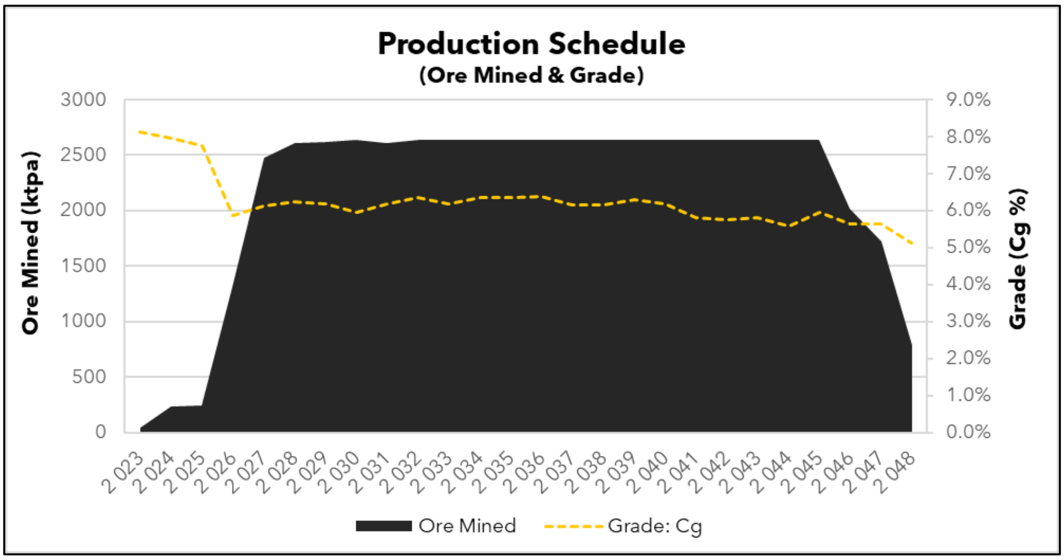
22.3.1 Production Schedule

A monthly production schedule has been included in the financial model. The production schedule encapsulates the pre-stripping waste, high-grade ore and low-grade ore mined from the open pit over the LoM. The annualised LoM waste and ore production profile is illustrated in Graph 54 below.



Graph 54: Production Schedule (Waste vs. Ore Mined)

The annualised LoM ore mined and grade profile is illustrated in Graph 55 below.



Graph 55: Production Schedule (Ore Mined & Grade)

Phase 1 of the TEM production schedule commences in September 2023 and continues through May 2026. Steady-state ore production during this phase targets 240 ktpa at an average head grade of 7.9% Cg. The waste to ore tonnes strip ratio during Phase 1 is 0.5.

Phase 2 of the TEM production schedule commences from June 2026 and continues over the remaining life of mine. Steady-state ore production ramps-up to 2.6 Mtpa at an average head grade of 6.1% Cg. The waste to ore tonnes strip ratio during Phase 2 is 0.3.

A summary of mine physicals for the total LoM is shown in Table 109 below.

Table 109: Mine Physicals

Metric	Unit of Measurement	Total
Waste Mined (LoM Total)	Mt	19.2
Ore Mined (LoM Total)	Mt	56.3
Strip Ratio	ratio (t_{waste}: t_{ore})	0.3
Average ROM Grade	% Cg	6.1%
Contained Graphite (LoM Total)	Mt	3.4
Life of Mine	years (active)	24.8
Life of Mine	years (at steady-state production)	21.7
Nameplate Ore Production	Mtpa	2.6
Average Annual Ore Production	Mtpa	2.2

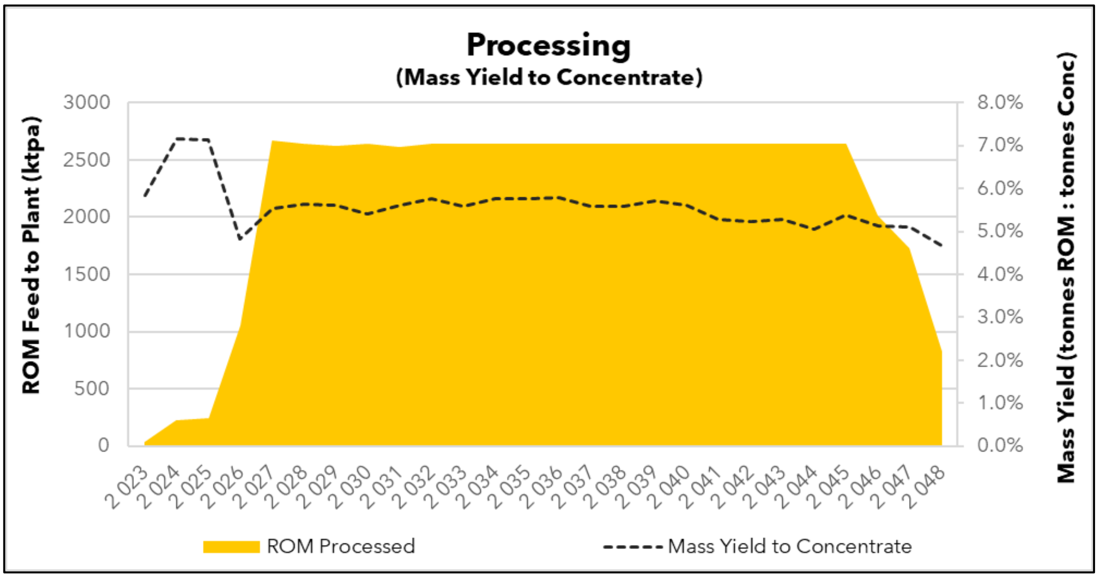
Ore mined from the open pit is fed to the onsite concentrator plant(s), subject to minimum levels of emergency ore stockpiles maintained on surface. An emergency stockpile of 15,000t of ROM is maintained during Phase 1 and 25,000t of ROM is maintained during Phase 2 for the TEM.

22.3.2 Recovery and Yield to Concentrate

The financial model assumes that the Phase 1 processing plant is in the process of ramping-up and is set to achieve steady-state throughput of 240 ktpa and recovery of 88.3% from January 2024.

It also assumes that first fills of the Expansion Project processing plant will commence in June 2026 and will ramp-up over 7 months until steady-state throughput of 2.4 Mtpa and recovery of 88.3% are achieved from December 2026. The combined normal capacity of both processing plants is approximately 2.64 Mtpa.

The LoM profile of the head feed and mass yield to concentrate is shown in Graph 56 below.



Graph 56: Concentrator Plant Feed and Mass Yield

Phase 1 is expected to produce at a saleable concentrate rate ranging between 16,800 and 17,000 dry tonnes per annum. Steady-state concentrate production on completion of the Project (combining the concentrate output of both plants), will yield 150 ktpa. Commercial production is attained once each plant achieves concentrate production rates of 67% of the respective plants steady-state concentrate production rates over a time-period of at least 1 month.

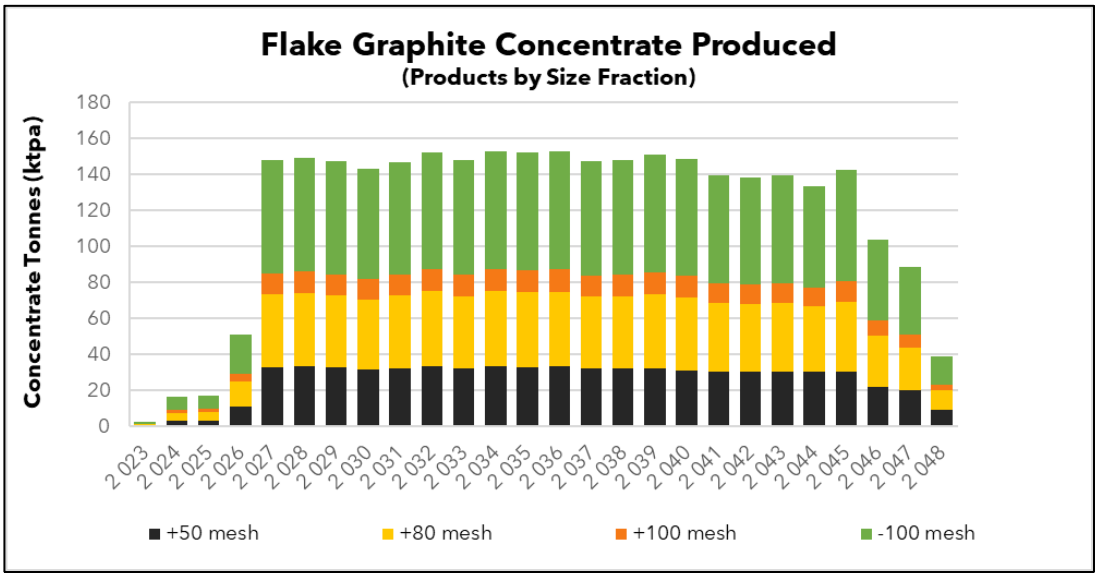
The LoM recovery and mass yield to concentrate is shown in Table 110 below.

Table 110: Processing Recovery & Mass Yield

Metric	Unit of Measurement	Total
Recovery	%	88.1%
Mass Yield to Flake Graphite Concentrate	% (t : t)	5.5%

The concentrate produced is filtered and dried prior to screening into four sized products. Each sized product is bagged into standalone bulk bags, stockpiled and transported to port for eventual export.

The LoM distribution of concentrate amongst the 4 sized products is shown in Graph 57 below.



Graph 57: Flake Graphite Concentrate Production

22.4 Inputs and Assumptions

22.4.1 Commodity Prices

Commodity prices for all flake graphite concentrate products were based on the real prices for 94%-95% Cg reference products, as provided by Benchmark Mineral Intelligence and stated on an FOB China incoterms basis.

No value-in-use price adjustment for differential freight rates between Madagascar and China, or price premium for non-Chinese sourced graphite, have been assumed when determining sales prices for the project on a FOB Port of Tulear basis.

These prices have, however, been adjusted to account for the quality premium associated with the higher average grade-in-concentrate the project aims to achieve (ie. 97.3% Cg), relative to the grade-in-concentrate of the reference products.

The quality premium is calculated as a pro-rata price increase of 4.3% for every 1% the grade-in-concentrate exceeds 94.5% Cg. The grade premium is advised by Fastmarkets and is based on recently observed premia. All prices in this section are inclusive of the quality premium (unless stated otherwise).

The following three commodity price scenarios were adopted for the purposes of the economic analysis:

- Expert Analyst Forecast (expert forecast).
- Three-year trailing average prices up to September 1, 2023 (3-year trailing average).
- Spot Prices as of September 1, 2023 (spot prices).

The weighted-average graphite price is comprised of the average of the price of each final product, in the proportion to the physical distribution of each final product produced.

Table 111 summarises the LoM average realised commodity price for each final product produced (Real, September 2023)

Table 111: Commodity Price Scenarios

Concentrate Product	Concentrate Grade	Distribution	Expert Forecast	3 Year Trailing Average	Spot
Unit of Measurement	**% Cg**	**%**	**US$/t of concentrate sold (Real Sept 2023)**		
+50 Mesh Superflake® Graphite	96.9%	21.8%	1,352	1,721	1,371
+80 Mesh Superflake® Graphite	97.0%	27.3%	1,219	1,239	1,183
+100 Mesh Superflake® Graphite	97.2%	8.0%	1,160	1,042	934
-100 Mesh Superflake® Graphite	97.6%	42.9%	1,097	764	733
Weighted Average	97.3%	100.0%	1,191	1,124	1,011

Source: Benchmark Mineral Intelligence & Fast Markets

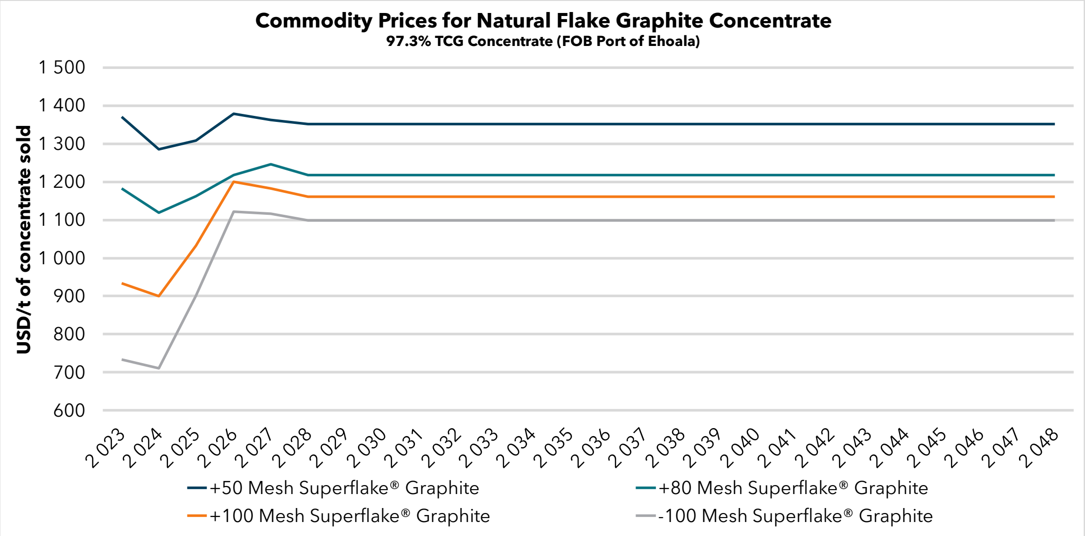
The Expert Analyst Forecast assumes that the forward-looking real prices across the various physical flake size distributions will converge over the short-term. As illustrated in Graph 58, prices for +50 Mesh Superflake Graphite and +80 Mesh Flake Graphite trend lower, while prices for +100 Mesh Flake Graphite and -100 Mesh Flake Graphite trend higher between the BDV and 2028.

As a result, the weighted average graphite price trends upwards from US$995/t in 2023 up to US$1,193/t in 2028. Prices quoted in 2028 are assumed to persist thereafter and are unchanged in real terms into the long-term. The LoM average realised commodity prices are reflected in Table 112.

Table 113 reports the first five years and long-term, real, forecast commodity prices that encompass the Expert Forecast, which is also illustrated over the full LoM in Graph 58.

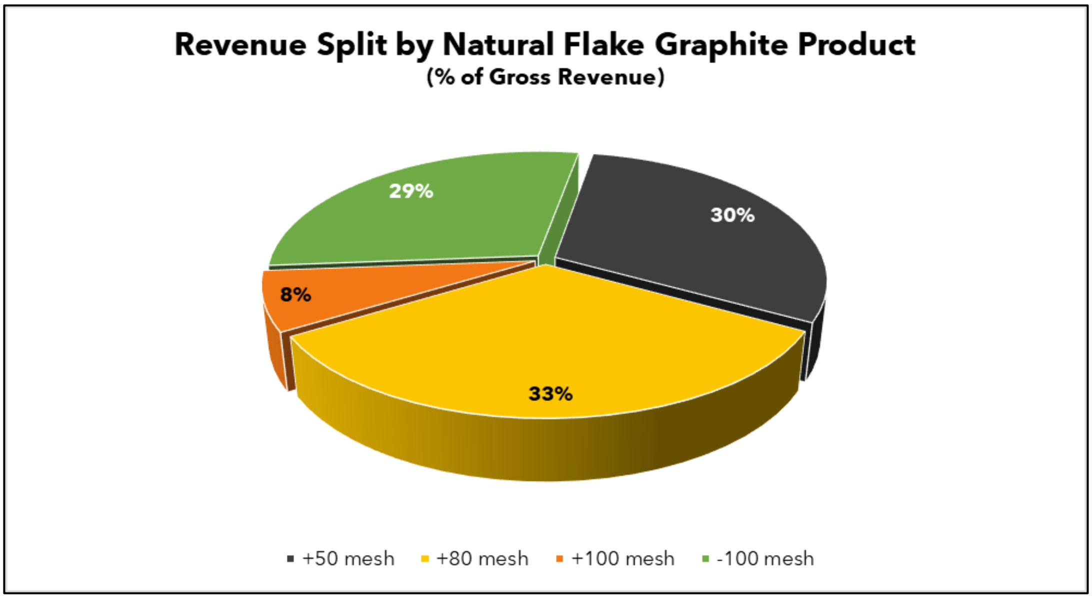
Table 112: Expert Forecast Commodity Price Scenario per year

Product	2023	2024	2025	2026	2027	2028 (Long-Term)
Unit of Measurement	**US$/t of concentrate sold (Real, Sept 2023)**					
+50 Mesh Superflake® Graphite	1,371	1,285	1,309	1,379	1,362	1,351
+80 Mesh Superflake® Graphite	1,183	1,119	1,163	1,218	1,246	1,218
+100 Mesh Superflake® Graphite	934	900	1,032	1,200	1,183	1,161
-100 Mesh Superflake® Graphite	733	711	901	1,122	1,116	1,099
Weighted Average Price	**995**	**948**	**1,062**	**1,209**	**1,212**	**1,193**



Graph 58: Expert Commodity Price Forecast (Source: Benchmark Minerals Intelligence & Fastmarkets)

Based on the quantities of each of the concentrate products sold (Graph 59), and the forecast commodity prices for those products (as indicated in Graph 59) the proportional contribution to revenue of each product over the LoM is shown in Graph 60.



Graph 59: Revenue Split by Natural Flake Graphite Product Size

22.4.2 Foreign Exchange Rates

The following three foreign exchange rate scenarios were adopted for the purposes of the economic analysis:

- Expert Forecast.
- Three-year trailing average prices up to September 1, 2023 (3 year trailing average).
- Spot Prices as of September 1, 2023 (spot prices).

Foreign exchange rates are based on econometric models developed by IHS Global Insight, a reputable source of macro-economic and socio-economic data, accessed via S&P Global's Capital IQ platform as of September 2023.

Nominal exchange rates are converted into real exchange rates based on the corresponding PPP between the foreign currency and the United States Dollar (US$), which is the presentation and functional currency of NextSource.

The foreign exchange rates forecast for 2028 are held flat in real terms over the long-term.

Cost estimates developed in the currency in which the cost is incurred have been translated into United States Dollar-based cash flows. The weighting of each source currencies' contribution to the periodic capital and operating costs are used to assess the project's cash flow sensitivity to foreign exchange rate fluctuations and the overall impact on key results metrics.

Prices for graphite products are denominated in United States Dollars.

As such, strengthening of the US$ relative to other currencies has the effect of lowering cost and improving the project's business case.

Table 113 summarises the LoM annual average foreign exchange rate for each currency the project is exposed to (Real, September 2023).

Table 113: Foreign Exchange Rate Scenarios per year (Source: S&P Global Capital IQ / IHS Global Insight)

FX Rate	Unit of Measurement	2023	2024	2025	2026	2027	2028
Malagasy Ariary to United States Dollar (MGA:US$)							
Expert Forecast	MGA / US$	4,193	4,198	4,212	4,219	4,225	4,227
3-Year Trailing Average	MGA / US$	4,312	4,312	4,312	4,312	4,312	4,312
Spot	MGA / US$	4,225	4,225	4,225	4,225	4,225	4,225
South African Rand to United States Dollar (ZAR:US$)							
Expert Forecast	ZAR / US$	17.37	17.22	17.25	17.38	17.58	17.73
3-Year Trailing Average	ZAR / US$	16.01	16.01	16.01	16.01	16.01	16.01
Spot	ZAR / US$	16.00	16.00	16.00	16.00	16.00	16.00

22.4.3 Inflation and Escalation

All cash flows in the final undiscounted cash flow analysis are presented in real, September 2023 terms. As such, all final undiscounted cash flows are exclusive of any inflation-linked increases.

Certain calculations performed in the financial model require the evaluation of cash flows denominated in nominal money terms, for instance, the carry-forward of any unutilised assessed tax losses and capital allowances, or calculation of royalty obligation payments under the Vision Blue Royalty. All nominal cash flows are converted to real cash flows before inclusion in the final undiscounted free cash flow.

No above, (or below) inflation cost escalations have been applied.

22.4.4 Selling Costs

Selling cost refer to the offsite costs incurred to advance the final products from an ex-works (EXW) basis to the point at which control of the products transfer to the customer and revenue is realised, ie. on a FOB Port of Tulear basis. This definition excludes any royalties, as these are considered separately.

Selling costs are comprised of inland transportation costs (including port fees and handling charges) and agency fees:

- **Inland transportation**.

  The cost of transporting the bagged flake graphite product to Port of Tulear has been supported by a quote and proposed terms of agreement for logistics services by a local contractor. The services include the supply of trucks and trailers operating in convoy to carry product free-on-truck from Fotadrevo to FOB Port of Tulear.

- **Marketing and Agency Fees**.

  Agency fees refer to a 3% sales commission applicable to the gross revenue earned on the proportion of final product sold to a Japanese Trading Company ("Japanese partner"), with whom the project has a binding offtake agreement. Under the agreed terms, the Japanese partner will purchase up to 20,000 tpa of dried and screened natural graphite product. The agreement will see the Japanese partner leverage its sales relationships and act as an exclusive agent for sales, marketing and trading of downstream products sourced from the project.

22.4.5 Mineral Royalties

The project is subject to 3 mineral royalties:

- Government of Madagascar Royalty.
- Net Smelter Return Financing Royalty.
- Vision Blue Financing Royalty.

22.4.5.1 Government of Madagascar Royalty

This section is focused on the calculation of the Government of Madagascar Royalty, as it pertains to the application of the Law on the Mining Code (2023) and its interrelation with the LGIM Code. The application and interpretation of local legislation has relied on the expert opinion provided by John W Ffooks & Co ("JWF"), who have acted in the capacity of in-country legal experts advising on tax-related matters.

Please refer to section 22.4.10 for further information regarding the application and interpretation of the tax legislation that has been applied as part of the economic analysis.

The Government of Madagascar Royalty, otherwise entitled "Special Duties and Taxes on Mining Products" (known as "DTSPM"), has been charged to the respective tax entities in the following manner:

- **Holder of the Mining Permit Entity**:

  DTSPM rate of 5% has been charged against the revenue attributable to the local sale of ore to the Transformation Entity utilizing the placeholder local market value of US$4.23/t ore mined. The DTSPSM rate reduces to 2.5% given the stipulation in the LGIM Code which allows a 50% reduction to the rate applicable to point of the first sale of the mining products.

- **Transformation Entity**:

  Under the new Mining Code, the Transformation Entity benefits from a 30% reduction to the DTSPM charge. As such, the DTSPM rate of 3.5% has been charged against the FOB value of the graphite concentrate being exported at Port of Tulear. This rate remains consistent at 3.5% once LGIM status comes into effect as no further benefit is attributed to the Transformation Entity.

The Techno-Economic Model utilized the rates applicable under the new Mining Code (without LGIM Code application) from October 2023 and the rates applicable under the New Mining Code (with LGIM Code application) from June 2026 until the end of the LoM.

A summary of how the rates have been attributed to either tax entity is summarised in Table 114**Error! Reference source not found.**.

Table 114: Mineral Royalty Rate

Assumption	Holder of the Mining Permit Entity	Transformation Entity
DTSPM Charge (without LGIM application)	5% of revenue	3.5% of FOB value of sales
DTSPM Charge (with LGIM application)	2.5% of revenue	3.5% of FOB value of sales

The sum of the Government of Madagascar Royalty payable by either entity is included in the cash flows of the project.

22.4.5.2 Net Smelter Return Financing Royalty

A royalty imposing a charge of 1.5% of the NSR value of all industrial minerals produced by the project is currently held by one of the directors of NextSource Materials Inc., Mr. Brett Whalen. This royalty was purchased in his individual capacity from Capricorn Metals Limited [ASX:CMM] (formerly called Malagasy Minerals Limited [ASX:MGY] until February 2016) and has been held by him since April 29, 2016, (prior to his appointment as director on July 20, 2020).

22.4.5.3 Vision Blue Financing Royalty

The payments due to Vision Blue Resources Limited are calculated in accordance with the royalty financing agreement:

- During the “Minimum Payment Period” (“MPP”), the agreement requires bi-annual payments (June 30 and December 31) of the greater of US$1.65 million and 3% of the gross sales revenue (“GSR”) during the trailing 6 months from graphite concentrate sales until such time that cumulative royalty payments total US$16.5 million. The end of the MPP is estimated to be reached in September 2027.

- Thereafter, the agreement requires only the bi-annual payments of 3% of the GSR during the trailing 6 months from graphite concentrate sales until the end of the LoM. NEXT has the option at any time after the MPP to reduce the rate to 2.25% of gross sales revenue by paying US$20 M to Vision Blue. The TEM assumes the option to reduce the rate is not exercised. All cash flows calculations relating to the Vision Blue Royalty (including the minimum payments) are performed in nominal US$ terms. Royalty settlement payments are converted back to real, September 2023 US$ terms for inclusion into the project cash flow evaluation.

22.4.6 Capital Expenditure

A capital budget estimate was prepared in accordance with the approved WBS.

All capitalised costs incurred prior to the Commercial Production is reported as Project Capital (US$186.9 million) and all capitalised costs incurred post Commercial Production is reported as Sustaining Capital (US$205.3 million), or as Closure Capital (US$12.8 million).

A summary of Total Capital Expenditure (Project, Sustaining and Closure) is reported Table 115.

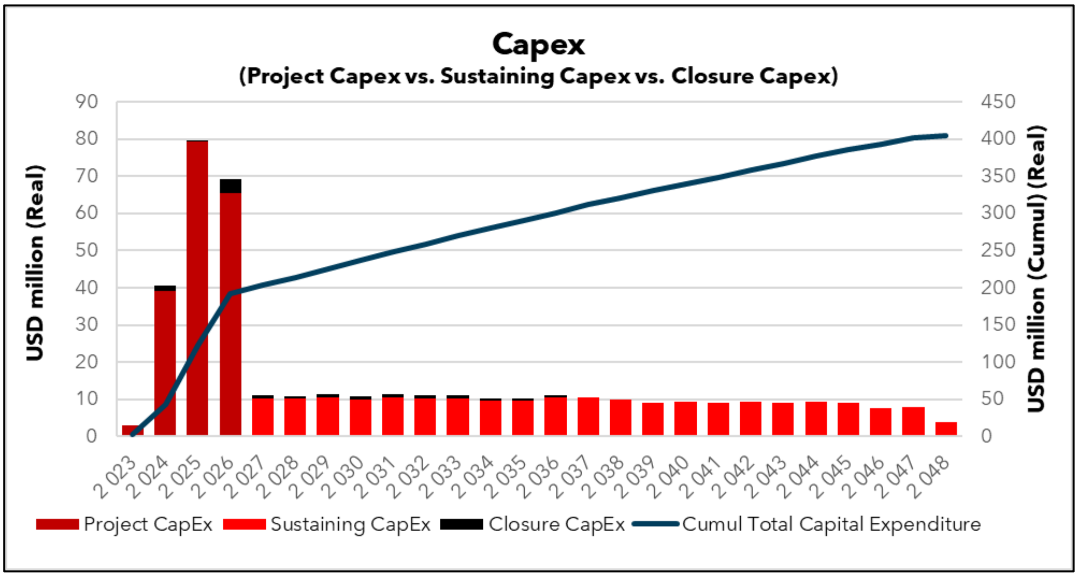
Table 115: Capital Expenditure Summary per Work Breakdown Structure Level 1

WBS L1 Code	Metric	Unit of Measurement	Total
100	Open Pit Mining	US$ M (Real, Sept 2023)	3.6
200	Process Plant	US$ M (Real, Sept 2023)	58.4
300	On-Site Infrastructure	US$ M (Real, Sept 2023)	33.7
500	Project Management	US$ M (Real, Sept 2023)	18.4
600	Owner's Cost	US$ M (Real, Sept 2023)	20.3
700	Other Capitalised Costs	US$ M (Real, Sept 2023)	31.2
900	Provisions	US$ M (Real, Sept 2023)	21.2
Sub Total: Project CAPEX (incl. Capitalised OPEX)		**US$ M (Real, Sept 2023)**	**186.9**
Sustaining CAPEX		US$ M (Real, Sept 2023)	205.3
Closure CAPEX		US$ M (Real, Sept 2023)	12.8
Total CAPEX		**US$ M (Real, Sept 2023)**	**405.0**

Project CAPEX includes an assessment of the initial materials and supplies that are incurred prior to Commercial Production and are consumed in the production process. These upfront consumables amounting to U$25.2 M are included in the capitalised operating costs line, (as detailed in section 22.4.9). As such, Project CAPEX excluding the working capital component totals US$161.7 million.

Contingencies of US$21.2 million contribute 11.3% to the Project CAPEX estimate.

Capital has been cash flowed in line with the project's execution schedule. The annualised Capital Expenditure cash flows as per the execution schedule is shown Graph 60.



Graph 60: Capital Expenditure over Life of Mine

Any capitalised costs incurred prior to September 1, 2023 are considered sunk and not included in the economic evaluation model.

22.4.7 *Operating Expenditure*

An Operating Expenditure model was prepared to estimate all "on-mine" costs as well as the inland transportation costs, port fees and handling charges that are included within selling costs. The estimate was prepared in accordance with the approved WBS. The operating expenditure model was built up from various techniques, (ie. first principals, zero-based. etc.), to develop the forecast cost of production.

A summary of the LoM average All-In Sustaining Unit Cost, reported by operating area, is shown in Table 116 below:

Table 116: Operating Expenditure Unit Cost Summary per Area

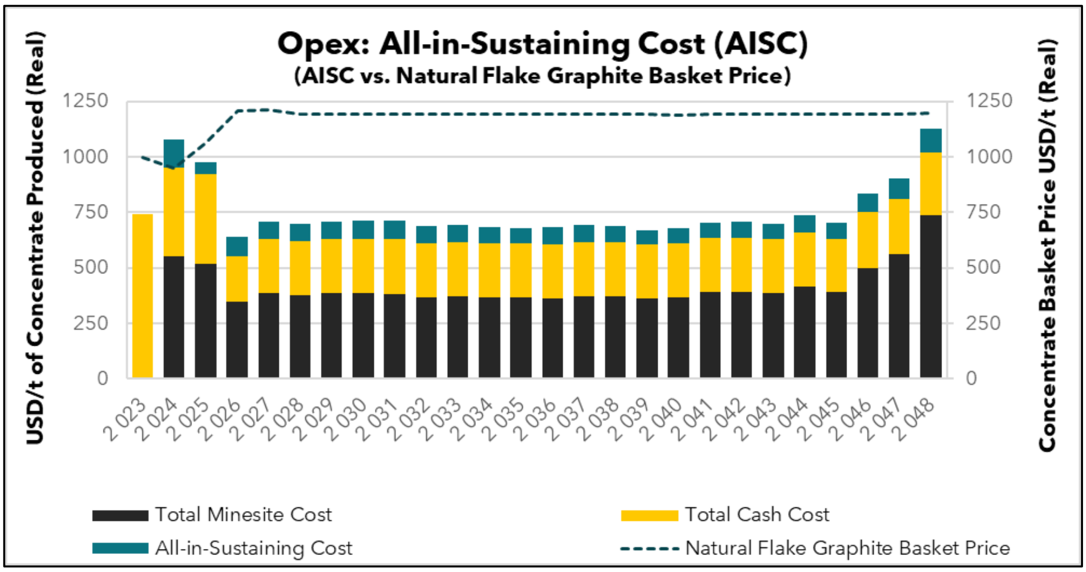
Metric	Unit of Measurement	Total
Open-Pit Mining	US$ /t of Concentrate Produced (Real)	61.4
Processing	US$ /t of Concentrate Produced (Real)	146.7
Infrastructure & Engineering	US$ /t of Concentrate Produced (Real)	139.1
On-site General & Administrative	US$ /t of Concentrate Produced (Real)	45.4
Sub Total: Minesite Cost [EXW]	**US$ /t of Concentrate Produced (Real)**	**392.6**
Selling Cost	US$ /t of Concentrate Produced (Real)	148.8
Royalties	US$ /t of Concentrate Produced (Real)	97.1
Sub Total: Cash Cost [FOB]	**US$ /t of Concentrate Produced (Real)**	**638.5**
Corporate General & Administrative	US$ /t of Concentrate Produced (Real)	5.3
Reclamation & Closure Provision	US$ /t of Concentrate Produced (Real)	4.1
Sustaining CAPEX	US$ /t of Concentrate Produced (Real)	66.4
Total: All-in-Sustaining Cost	**US$ /t of Concentrate Produced (Real)**	**714.3**

A reconciliation of the LoM average Minesite cost, reported by cost element, is shown in Table 117 below:

Table 117: Minesite Cost Unit Cost Reconciliation per Cost Element

Metric	Unit of Measurement	Total
Labour	US$ /t of Concentrate Produced (Real)	35.1
Utilities	US$ /t of Concentrate Produced (Real)	52.1
Materials & Stores	US$ /t of Concentrate Produced (Real)	263.9
External Services	US$ /t of Concentrate Produced (Real)	19.3
Fixed Overheads	US$ /t of Concentrate Produced (Real)	27.5
Less: Corporate General & Administrative	**US$ /t of Concentrate Produced (Real)**	**-5.3**
Total: Minesite Cost [EXW]	**US$ /t of Concentrate Produced (Real)**	**392.6**

Graph 61 depicts the profile of All-In-Sustaining Costs incurred over the LoM on a unit cost basis, overlaid with the weighted average graphite basket price. The average LoM EBITDA margin is 45.9% of gross revenue.



Graph 61: Operating Cost Reporting Key Benchmarks

22.4.8 *Mine Rehabilitation and Closure Costs*

An evaluation of the Mine Rehabilitation and Closure Plan (MRCP) has been performed by GlobeSight (Pty) Limited and included in the Techno-Economic Model. The MRCP put forward conforms to local law and international standards of best practice, such as The Equator Principles' risk management framework, the International Finance Corporation's (IFC) Performance Standards (2012) on environmental and social responsibility and Principle 6 of the International Council of Mining and Metals' prerogative for sustainable development.

The mitigation of environmental impacts and rehabilitation of the mining area is a legislative requirement within the Madagascan Mining Code. As stipulated in Mining Code, the mechanism for managing the provision of environmental rehabilitation consists of setting aside dedicated funds in escrow that are in reasonable proportion to the eventual Rehabilitation and Closure liability. This includes provisioning for the possible outcome of early or temporary closure.

In response to the requirement, the financial model considers a lump-sum cash contribution of 40% of the expected final Mine Rehabilitation and Closure cost into a third-party managed cell at the point where Commercial Production is reached. Thereafter, periodic payments over the forthcoming 10 years increase the relative size of the Environmental Asset in proportion to expected Rehabilitation and Closure liability, until the requisite proportion of funds have been set aside to settle the liability (from the fund), at the end of the LoM.

22.4.9 Working Capital

Working capital is regarded as the cash commitment to inventory, trade receivables and trade payables applied in the TEM:

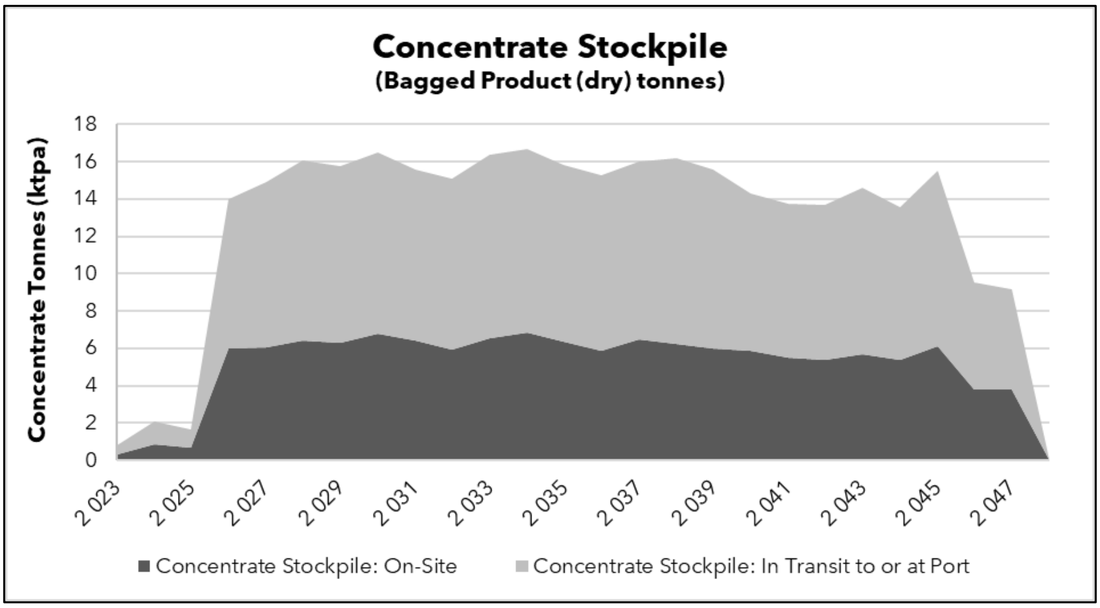
- **Capitalised Operating Cost**:

  Pre-production operating costs for the expansion incurred prior to attaining Commercial Production include First Fills and Other Pre-Production Costs (excluding Critical Spares). These costs represent the working capital committed to purchase the initial materials and supplies to be consumed in the production process. These upfront consumables totalling U$25.2 M are capitalised as inventory within the Project Capital estimate (Code 700 per the WBS).

- **Finished Goods Inventory**:

  The delay between concentrate production, stockpiling at the mine site, trucking, stockpiling at the warehouse at the Port of Tulear and eventual loading to ship on a FOB basis is modelled as a delay to revenue recognition and has an implicit working capital implication.

The proportion of concentrate production stockpiled on site and at port respectively for each month of the year is shown Graph 62 below.



Graph 62: Concentrate Stockpile

- **Accounting Accrual**:

  The delay between the point at which revenue is earned upon the loading of concentrate to ship at Port of Tulear and the following receipt of cash is modelled as a trade receivable. Payment terms have been associated with each stream of final product that will be purchased by each off-taker associated with the project.

Similarly, the delay between the point at which expenses are incurred and cash payment is made is modelled as a trade payable.

The netting-off of changes to trade receivables against changes to trade payables is reported as the 'net change in working capital' and modelled as a separate line in the project's cash flows.

The trade receivable days for each offtake is shown in Table 118 below.

Table 118: Trade Receivable Days

Graphite Product	Unit of Measurement	Thyssenkrupp	Japanese Trader	Battery Anode Market	Weighted Average
+50 Mesh Superflake® Graphite	Trade receivable days outstanding	19	45	16	16
+80 Mesh Flake Graphite	Trade receivable days outstanding	19	45	16	18
+100 Mesh Flake Graphite	Trade receivable days outstanding	19	45	16	32
-100 Mesh Flake Graphite	Trade receivable days outstanding	19	45	16	22
Weighted Average: Trade Receivables	**Trade receivable days outstanding**				**20**

The trade payable days for each cost element is shown in Table 119 below.

Table 119: Trade Payable Days

Cost Element	Unit of Measurement	Result
Labour	Trade payable days outstanding	0
Utilities	Trade payable days outstanding	0
Materials & Stores	Trade payable days outstanding	30
External Services	Trade payable days outstanding	30
Fixed Overheads	Trade payable days outstanding	15
Selling Expenses	Trade payable days outstanding	60
Weighted Average: Trade Payables	**Trade payable days outstanding**	**33**

No adjustment for Value-Added Tax or General Sales Tax has been considered.

22.4.10 Corporate Income Tax

The focus of this section is to explain the interpretations of Madagascan tax legislation that have been applied as part of the economic analysis and, more specifically, the components that affect the calculation of corporate income tax. Section 22.4.5.1 details the components of the legislation that relate to the calculation of government royalty charges.

22.4.10.1 Updated Law on the Mining Code

On July 27, 2023, the Government of Madagascar made the announcement of and effectively enacted an updated legislation, the Law on the Mining Code (2023). The nascent law replaced the former Law on the Mining Code (2005). The Law on the Mining Code (2023) came into full force in Malagasy legislation on the Official Gazette Date on October 2, 2023 and consequentially bears on the Molo Project from this date.

The new Mining Code introduces uncertainties to the financial evaluation as certain components remain unclear, contradictory to its own and other bodies of pre-existing legislation, and untested in industry. Clarification of specific terms by way of an Implementing Decree is expected to be received from the government for industry consultation, although its release date has not been communicated.

The application and interpretation of local legislation has relied on the expert opinion provided by John W Ffooks & Co ("JWF"), who have acted in the capacity of in-country legal experts advising on tax-related matters.

As advised by JWF, the new Mining Code requires distinction and separation of the entity holding the mining permit and the entity responsible for "transformation". The definition of "Transformation of mineral substances" has been interpreted to include the processing of graphite ore into a graphite concentrate, given that the enhancement of grade is commensurate with the enhancement of value across the process. Separate taxable income calculations have been prepared in the TEM to conform to this requirement.

The new Mining Code stipulates that the entity holding the mining permit is to sell the mineral substances to the Transformation Entity at a local market value regulated by the Order of the Minister of Mines. However, at the time of this report, no such circular from that sphere of government exists and given that no universal reference price for graphite ore exists either, the matter of a local market value sales remains open to interpretation. The economic analysis assumes a placeholder local market value of US$4.23/t ore mined, which allows for a 25% EBITDA margin mining costs in the taxable income of the mining permit entity.

The calculation of taxable income of the entity holding the mining permit allocated all operating costs incurred in the production of ore and all capital costs associated with the purchase of mining equipment. The calculation of taxable income in the Transformation Entity includes the purchase cost of the ore, as well as all non-mining operating and capital costs (including the corporate and site overhead costs).

The assumptions applied in accordance with the new Mining Code, (without application of the LGIM Code, which is the focus of section 22.4.10.2) are summarised in Table 120 below.

Table 120: Corporate Income Tax Assumptions Applied under the New Mining Code (no LGIM)

Assumption	Holder of the Mining Permit Entity	Transformation Entity
Corporate Income Tax Rate	20% of taxable income	20% of taxable income
Synthetic Tax *(applicable in years where taxable income is less than MGA 200 million)*	3% of revenue	3% of revenue
Minimum Tax	0.5% of revenue plus MGA 100,000	0.5% of revenue plus MGA 100,000

Mineral Royalties due to the government are deductible from income tax, however Royalties payable on financing arrangements are not deductible from income tax. This is applicable to both tax entities.

Any balance of unutilized assessed losses expires if after a period of 5 years the assessed losses remain unutilized. This is applicable to both tax entities.

The balance of unutilized capital tax allowances which offset future taxable income pertain substantially to the sunk capital expenditures on the Phase 1 process plant and onsite infrastructure. The balance of MGA 80.6 billion is brought down in the Holder of the Mining Permit Entity as of September 1, 2023.

The sum of the corporate income tax payable by either entity is included in the cash flows of the project.

22.4.10.2 Large Scale Investment Status

The legal opinion provided by JWF included an assessment of the special regime for Large Scale Investments ("LGIM Code") in the Malagasy Mining Sector (Law n°2001-031 on October 8, 2002, modified by Law n°2005-022 of October 17, 2005). The assessment served to opine on the eligibility of the project for the LGIM certification, its proposed interaction with the new Mining Code (given that certain interpretations are particularly mismatched with the new Mining Code) and provide confirmation of the inputs and assumptions to be applied in the economic analysis.

JWF has confirmed in their opinion that as the Expansion Project's capital investment into the country exceeds MGA 50 billion (approximately US$11 million) and given due consideration to meeting other minimum requirements, the Project is eligible for the beneficial tax regime offered under the LGIM certification.

Importantly, certain assumptions applied in the economic analysis that pertain to instances where the LGIM Code contradicts the new Mining Code have pre-empted changes to the current LGIM Code, on the basis that the new Mining Code takes preference, as it is aligned with the direction of the latest legislative requirements.

A key point of departure concerns the definition of "Transformation" under the current LGIM Code, which has been assumed to be amended to include the activities associated with the separation and concentration of mineral substances. These activities are specifically excluded from the current definition.

JWF has advised that the current LGIM Code has updated interpretations which are due for release from the government, whether by virtue of an entirely refreshed LGIM Code or by specific stipulations within the awaited Implementation Decree of the new Mining Code.

Based on the assessment of the LGIM Code performed, it was determined that the beneficial tax regime offered by LGIM certification was a viable, value accretive initiative available to the Project. NEXT has indicated that it would be reasonable to expect government to complete the approval process for LGIM status from June 1, 2026.

The assumptions applied in accordance with the new Mining Code (with application of the LGIM Code) are summarised in Table 121.

Table 121: Corporate Income Tax assumptions applied under the new Mining Code (with LGIM)

Assumption	Holder of the Mining Permit Entity	Transformation Entity
Corporate Income Tax Rate	20% of taxable income	10% of taxable income
Synthetic Tax *(applicable in years where taxable income is less than MGA 200 million)*	3% of revenue	3% of revenue
Minimum Tax	0.5% of revenue plus MGA 100,000	0.5% of revenue plus MGA 100,000

The Minimum Tax above is subject to a waiver of the minimum tax payable for the first 5 years after approval of LGIM status.

A scenario has been performed under the premise that LGIM certification has not been obtained. Refer to section 22.6.3.1 for further information.

22.4.11 *Discount Rate*

The discount rate applied to value the undiscounted, unlevered free cash flows as at September 2023 is 8% real. This discount rate compares well to peer graphite projects that have announced Feasibility Study results within the last 24 months.

The pre-tax discount rate has been back-calculated from the equivalent Post-Tax $NPV_{8\%}$ and is 9.05% real. This is only used for reporting purposes.

Please refer to section 22.6.3.3**Error! Reference source not found.** where various discount r ate scenarios have been applied to assess the impact on NPV.

22.5 Summary of Results

The financial results are presented for the Molo Graphite Project, and represent the forward-looking, overall execution plan for both the Phase 1 and expansion parts of the project as of the Base Date of Valuation, September 1, 2023.

Headline results are presented in terms of a scenario that encompasses forward-looking macro-economic assumptions (Forecast). Results for two other scenarios are also presented that rank *pari-passu* with the headline scenario, that consider commodity prices and foreign exchange rate assumptions at 3 year trailing prices and at spot prices respectively.

Key economic results are shown in Table 122 below.

Table 122: Key Economic Results

Metric	Unit of Measurement	Forecast	3 Year Trailing Average	Spot
$NPV_{8\%}$ (Post-Tax)	US$ million	370.0	299.8	182.7
IRR (Post-Tax)	%	29.0%	26.0%	19.7%
$NPV_{8\%}$ (Pre-Tax)	US$ million	424.1	345.7	216.2
IRR (Pre-Tax)	%	31.1%	27.8%	21.2%
Undiscounted Payback Period	Years (from first Phase 2 concentrate production)	3.1	3.5	4.8
Undiscounted Payback Period	Years (from first Phase 2 Capex)	5.8	6.1	7.5
Peak Funding Requirement	US$ million	178.5	177.9	183.5

The business case is value accretive across all 3 macro-economic scenarios.

The project generates a post-tax Net Present $Value_{8\%}$ (NPV) of US$370.0 million (Forecast Scenario), US$299.8 million (3 Year Trailing Average Scenario) and US$182.7 million (Spot Scenario). The post-tax Internal Rate of Return (IRR) is 29.0% (Forecast Scenario), 26.0% (3 Year Trailing Average) and 19.7% (Spot Scenario) respectively.

The payback where total, undiscounted project outflows are paid off with cash generated by operations is reached during May 2029. The payback period has been measured against two reporting norms, namely:

- The period commencing from the date of first concentration produced in respect of the expansion (June 2026).
- The period commencing from the date of first expansion CAPEX spend (October 2023).

Peak Funding Requirement measures the point of highest cash injection by equity-holders into the project. This is reached during October 2026.

22.5.1 Project Cash Flows

The annualised, undiscounted free cash flows for the Molo Graphite Project, assuming forecast macro-economic assumptions, are illustrated in Graph 63 on a detailed basis and in Graph 64 on a net free cash flow and cumulative free cash flow basis.

The undiscounted and discounted free cash flows, assuming forecast macro-economic assumptions, are reported in Table 123.

The undiscounted and discounted free cash flows, assuming 3 Year Trailing Average macro-economic assumptions, are reported in Table 124.

The undiscounted and discounted free cash flows, assuming spot macro-economic assumptions, are reported in Table 125.

Table 123: Undiscounted Cash Flow Summary at Forecast Macro-Economic Assumptions (US$ '000)

Metric	Sum of Row	First 5 years and 4 months						5-Year Aggregate	5-Year Aggregate	5-Year Aggregate	5-Year Aggregate
		2023 (4 months)	2024	2025	2026	2027	2028	(2029-2033)	(2034 - 2038)	(2039 - 2043)	(2044 - 2048)
Revenue from the sale of +50 mesh	938,531	413	3,838	4,559	11,598	46,207	45,730	224,945	227,011	214,347	159,884
Revenue from the sale of +80 mesh	1,060,947	500	4,691	5,688	13,215	51,621	50,804	252,364	257,816	245,583	178,665
Revenue from the sale of +100 mesh	295,584	124	1,184	1,584	3,880	14,094	14,004	70,029	72,131	69,107	49,446
Revenue from the sale of -100 mesh	1,499,875	516	4,963	7,334	19,410	71,573	71,304	356,168	366,354	350,660	251,593
Gross Revenue	3,794,937	1,554	14,675	19,164	48,104	183,495	181,843	903,506	923,311	879,697	639,588
Other Selling Costs	-14,422	-8	-140	-180	-198	-658	-667	-3,357	-3,459	-3,316	-2,438
Net Revenue [FOB]	3,780,515	1,546	14,535	18,984	47,906	182,836	181,175	900,149	919,853	876,381	637,150
Inland Transportation	-459,561	0	-2,497	-2,735	-5,059	-22,015	-22,173	-109,654	-112,073	-106,734	-76,621
Net Revenue [EXW]	3,320,953	1,546	12,037	16,249	42,847	160,821	159,003	790,495	807,780	769,647	560,529
OPEX: Open-Pit Mining	-189,997	0	-1,371	-1,232	-2,799	-10,328	-9,660	-46,206	-42,884	-39,438	-36,080
OPEX: Processing	-456,823	0	-2,986	-3,039	-6,262	-21,179	-21,174	-105,283	-106,364	-104,689	-85,847
OPEX: Infrastructure & Engineering	-430,478	0	-3,080	-3,071	-6,025	-19,391	-19,462	-96,992	-96,934	-96,972	-88,552
OPEX: On-site General & Administrative	-140,509	0	-1,538	-1,548	-2,536	-6,194	-6,130	-30,771	-30,711	-30,544	-30,536
OPEX: Corporate General & Administrative	-16,440	0	-670	-670	-756	-670	-670	-3,350	-3,350	-3,350	-2,954
Mineral Royalties	-309,064	-1,723	-3,957	-4,112	-5,506	-14,533	-14,396	-71,522	-73,084	-69,629	-50,602
Forex Adjustment	0	0	0	0	0	0	0	0	0	0	0
EBITDA	1,777,642	-177	-1,565	2,577	18,963	88,526	87,511	436,370	454,453	425,025	265,957

Metric	Sum of Row	First 5 years and 4 months						5-Year Aggregate	5-Year Aggregate	5-Year Aggregate	5-Year Aggregate
		2023 (4 months)	2024	2025	2026	2027	2028	(2029-2033)	(2034 - 2038)	(2039 - 2043)	(2044 - 2048)
Project CAPEX	-186,932	-3,061	-39,240	-79,254	-65,378	0	0	0	0	0	0
Sustaining CAPEX	-205,317	0	0	0	0	-10,130	-10,118	-51,458	-49,916	-45,690	-38,005
Closure CAPEX	-12,780	0	-1,433	-212	-3,709	-811	-811	-4,056	-1,747	0	0
Forex Adjustment	0	0	0	0	0	0	0	0	0	0	0
Working Capital: Accounts Receivable	0	-449	-493	-94	-4,612	-1,732	-607	-225	93	835	7,284
Working Capital: Accounts Payable	0	7	972	-31	5,175	671	328	35	399	-926	-6,630
Total Undiscounted Cash Flow, (Pre-Tax)	1,372,612	-3,679	-41,759	-77,014	-49,560	76,524	76,303	380,666	403,282	379,244	228,606
Corporate Taxes	-159,533	-9	-105	-132	-236	-895	-6,935	-39,577	-44,231	-42,031	-25,381
Total Undiscounted Cash Flow (Post-Tax)	1,213,080	-3,688	-41,864	-77,146	-49,796	75,628	69,369	341,089	359,050	337,212	203,225
Cumulative Undiscounted Cash Flow (Post-Tax)		-3,688	-45,552	-122,697	-172,493	-96,865	-27,496	313,592	672,643	1,009,855	1,213,080
Total Discounted Cash Flow (Post-Tax) (8.0%)	400,625	-3,639	-38,595	-67,199	-40,564	56,136	47,868	187,512	135,182	86,772	37,151
Cumulative Discounted Cash Flow (Post-Tax) (8.0%)		-3,639	-42,234	-109,433	-149,997	-93,861	-45,993	141,519	276,701	363,473	400,625

Table 124: Undiscounted Cash Flow Summary at Three-Year Trailing Macro-Economic Assumptions (US$ '000)

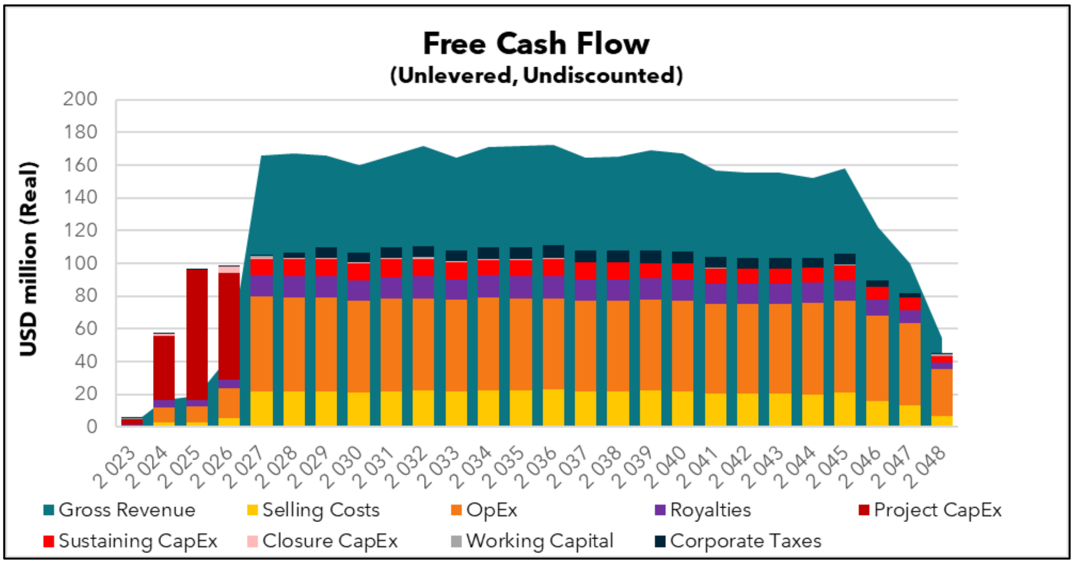
Metric	Sum of Row	First 5 years and 4 months						5-Year Aggregate	5-Year Aggregate	5-Year Aggregate	5-Year Aggregate
		2023 (4 months)	2024	2025	2026	2027	2028	(2029-2033)	(2034-2038)	(2039-2043)	(2044-2048)
Revenue from the sale of +50 mesh	1,195,205	519	5,141	5,997	14,479	58,383	58,253	286,545	289,176	273,045	203,667
Revenue from the sale of +80 mesh	1,078,139	524	5,193	6,058	13,434	51,313	51,649	256,561	262,103	249,667	181,636
Revenue from the sale of +100 mesh	265,522	138	1,371	1,599	3,370	12,415	12,572	62,869	64,756	62,042	44,390
Revenue from the sale of -100 mesh	1,044,608	538	5,332	6,219	13,218	48,988	49,558	247,546	254,626	243,718	174,864
Gross Revenue	3,583,473	1,719	17,037	19,872	44,502	171,099	172,033	853,522	870,661	828,471	604,558
Other Selling Costs	-11,311	-9	-165	-205	-181	-501	-516	-2,600	-2,678	-2,568	-1,888
Net Revenue [FOB]	3,572,162	1,710	16,871	19,668	44,321	170,597	171,517	850,922	867,983	825,903	602,669
Inland Transportation	-459,561	0	-2,497	-2,735	-5,059	-22,015	-22,173	-109,654	-112,073	-106,734	-76,621
Net Revenue [EXW]	3,112,601	1,710	14,374	16,933	39,262	148,582	149,344	741,268	755,910	719,170	526,048
OPEX: Open-Pit Mining	-189,997	0	-1,371	-1,232	-2,799	-10,328	-9,660	-46,206	-42,884	-39,438	-36,080
OPEX: Processing	-456,823	0	-2,986	-3,039	-6,262	-21,179	-21,174	-105,283	-106,364	-104,689	-85,847
OPEX: Infrastructure & Engineering	-430,478	0	-3,080	-3,071	-6,025	-19,391	-19,462	-96,992	-96,934	-96,972	-88,552
OPEX: On-site General & Administrative	-140,509	0	-1,538	-1,548	-2,536	-6,194	-6,130	-30,771	-30,711	-30,544	-30,536
OPEX: Corporate General & Administrative	-16,440	0	-670	-670	-756	-670	-670	-3,350	-3,350	-3,350	-2,954
Mineral Royalties	-292,349	-1,727	-4,053	-4,176	-5,418	-13,543	-13,616	-67,543	-68,895	-65,552	-47,827
Forex Adjustment	-7,506	0	-16	-14	-102	-413	-423	-1,957	-1,770	-1,582	-1,228
EBITDA	1,578,498	-17	660	3,184	15,364	76,864	78,210	389,165	405,002	377,043	233,025
Project CAPEX	-186,932	-3,061	-39,240	-79,254	-65,378	0	0	0	0	0	0
Sustaining CAPEX	-205,317	0	0	0	0	-10,130	-10,118	-51,458	-49,916	-45,690	-38,005
Closure CAPEX	-12,780	0	-1,433	-212	-3,709	-811	-811	-4,056	-1,747	0	0

Metric	Sum of Row	First 5 years and 4 months						5-Year Aggregate	5-Year Aggregate	5-Year Aggregate	5-Year Aggregate
		2023 (4 months)	2024	2025	2026	2027	2028	(2029-2033)	(2034-2038)	(2039-2043)	(2044-2048)
Forex Adjustment	-7,879	4	-177	-184	-109	-413	-423	-1,965	-1,782	-1,596	-1,235
Working Capital: Accounts Receivable	0	-500	-603	-11	-3,968	-1,513	-658	-171	78	738	6,609
Working Capital: Accounts Payable	0	8	976	-32	5,158	659	329	33	401	-925	-6,607
Total Undiscounted Cash Flow (Pre-Tax)	1,165,589	-3,565	-39,819	-76,508	-52,642	64,656	66,528	331,548	352,036	329,569	193,787
Corporate Taxes	-138,538	-10	-127	-143	-222	-853	-3,890	-34,766	-38,631	-37,583	-22,314
Total Undiscounted Cash Flow (Post-Tax)	1,027,051	-3,575	-39,946	-76,651	-52,863	63,803	62,638	296,782	313,405	291,986	171,473
Cumulative Undiscounted Cash Flow (Post-Tax)		-3,575	-43,521	-120,172	-173,035	-109,233	-46,595	250,187	563,593	855,579	1,027,051
Total Discounted Cash Flow (Post-Tax) (8.0%)	328,254	-3,528	-36,802	-66,768	-42,990	47,368	43,168	163,154	118,038	75,160	31,455
Cumulative Discounted Cash Flow Post-Tax) (8.0%)		-3,528	-40,330	-107,098	-150,089	-102,721	-59,553	103,601	221,640	296,799	328,254

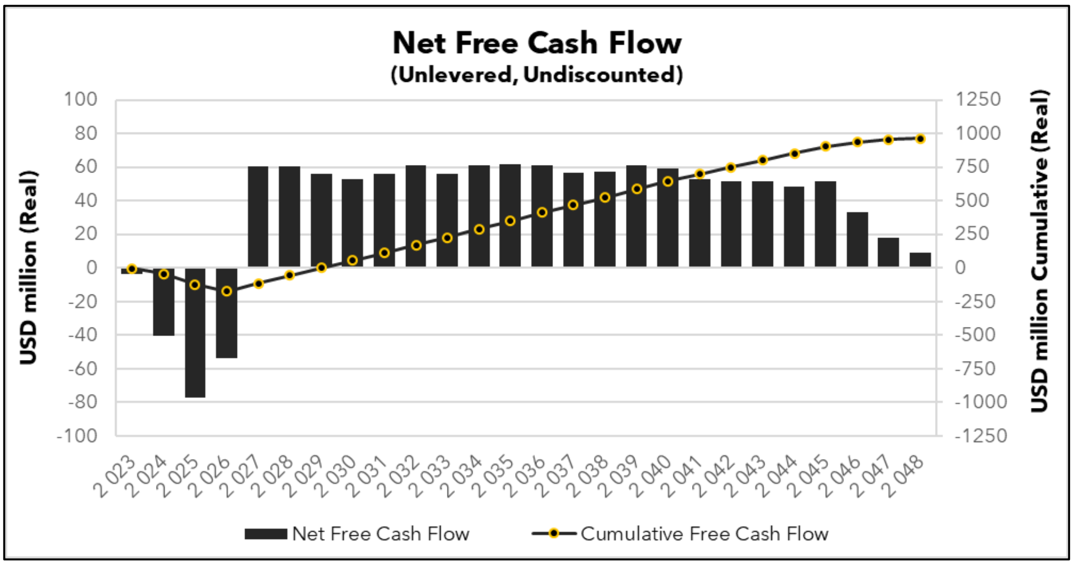
Table 125: Undiscounted Cash Flow Summary at Spot Macro-Economic Assumptions (US$ '000)

Metric	Sum of Row	First 5 years and 4 months						5-Year Aggregate	5-Year Aggregate	5-Year Aggregate	5-Year Aggregate
		2023 (4months)	2024	2025	2026	2027	2028	(2029-2033)	(2034-2038)	(2039--2043)	(2044-2048)
Revenue from the sale of +50 mesh	951,777	413	4,094	4,775	11,530	46,492	46,389	228,184	230,279	217,434	162,186
Revenue from the sale of +80 mesh	1,029,550	500	4,959	5,785	12,829	49,000	49,322	244,998	250,291	238,415	173,451
Revenue from the sale of +100 mesh	237,965	124	1,228	1,433	3,020	11,126	11,267	56,345	58,035	55,603	39,783
Revenue from the sale of -100 mesh	1,002,692	516	5,118	5,970	12,688	47,022	47,570	237,613	244,409	233,939	167,847
Gross Revenue	3,221,985	1,554	15,400	17,963	40,067	153,641	154,547	767,141	783,015	745,390	543,267
Other Selling Costs	-10,568	-8	-148	-183	-165	-469	-483	-2,434	-2,507	-2,404	-1,768
Net Revenue [FOB]	3,211,417	1,546	15,252	17,780	39,903	153,172	154,064	764,707	780,508	742,986	541,500
Inland transportation	-459,561	0	-2,497	-2,735	-5,059	-22,015	-22,173	-109,654	-112,073	-106,734	-76,621
Net Revenue[EXW]	2,751,856	1,546	12,754	15,045	34,844	131,157	131,892	655,053	668,435	636,252	464,879
OPEX: Open-Pit Mining	-189,997	0	-1,371	-1,232	-2,799	-10,328	-9,660	-46,206	-42,884	-39,438	-36,080
OPEX: Processing	-456,823	0	-2,986	-3,039	-6,262	-21,179	-21,174	-105,283	-106,364	-104,689	-85,847
OPEX: Infrastructure & Engineering	-430,478	0	-3,080	-3,071	-6,025	-19,391	-19,462	-96,992	-96,934	-96,972	-88,552
OPEX: On-site General & Administrative	-140,509	0	-1,538	-1,548	-2,536	-6,194	-6,130	-30,771	-30,711	-30,544	-30,536
OPEX: Corporate General & Administrative	-16,440	0	-670	-670	-756	-670	-670	-3,350	-3,350	-3,350	-2,954
Mineral Royalties	-263,864	-1,718	-3,969	-4,078	-5,195	-12,155	-12,226	-60,680	-61,932	-58,953	-42,957
Forex Adjustment	-8,966	0	-33	-31	-128	-476	-485	-2,267	-2,075	-1,886	-1,583
EBITDA	1,244,778	-172	-894	1,376	11,143	60,763	62,084	309,503	324,185	300,421	176,369
Project CAPEX	-186,932	-3,061	-39,240	-79,254	-65,378	0	0	0	0	0	0
Sustaining CAPEX	-205,317	0	0	0	0	-10,130	-10,118	-51,458	-49,916	-45,690	-38,005
Closure CAPEX	-12,780	0	-1,433	-212	-3,709	-811	-811	-4,056	-1,747	0	0
Forex Adjustment	-9,373	0	-204	-205	-136	-476	-485	-2,278	-2,090	-1,904	-1,595

Metric	Sum of Row	First 5 years and 4 months						5-Year Aggregate	5-Year Aggregate	5-Year Aggregate	5-Year Aggregate
		2023 (4months)	2024	2025	2026	2027	2028	(2029-2033)	(2034-2038)	(2039--2043)	(2044-2048)
Working Capital: Accounts Receivable	0	-449	-542	-9	-3,598	-1,383	-597	-165	72	675	5,997
Working Capital: Accounts Payable	0	7	974	-32	5,159	656	329	33	402	-925	-6,602
Total Undiscounted Cash Flow (Pre-Tax)	830,375	-3,675	-41,340	-78,336	-56,519	48,619	50,401	251,579	270,906	252,577	136,163
Corporate Taxes	-104,427	-9	-119	-134	-200	-766	-770	-26,201	-30,233	-29,585	-16,410
Total Undiscounted Cash Flow (Post-Tax)	725,949	-3,683	-41,459	-78,470	-56,718	47,853	49,631	225,378	240,674	222,991	119,753
Cumulative Undiscounted Cash Flow (Post-Tax)		-3,683	-45,142	-123,612	-180,330	-132,478	-82,847	142,531	383,205	606,196	725,949
Tota Discounted Cash Flow (Post-Tax) (8.0%)	207,659	-3,634	-38,214	-68,343	-46,037	35,531	34,151	123,841	90,702	57,486	22,178
Cumulative Discounted Cash Flow (Post-Tax) (8.0%)		-3,634	-41,848	-110,192	-156,229	-120,698	-86,547	37,294	127,996	185,481	207,659



Graph 63: Undiscounted Free Cash Flows (Forecast Macro-Economic Assumptions)



Graph 64: Net and Cumulative Free Cash Flow (Forecast Macro-Economic Assumptions)

22.6 Sensitivity and Scenario Analysis

22.6.1 Deterministic Sensitivity Analysis

A deterministic sensitivity analysis was developed based on the Project Evaluation and Review Technique ("PERT") method, which establishes a triangular distribution using a minimum value, maximum value and point estimate for each key driver identified in Table 126, which is based on inputs received from discipline experts.

The subsequent NPV and IRR that arises from applying the P10 and P90 estimate for each driver is plotted and assessed as a movement relative to the base results. For this purpose, the sensitivity has utilised the projected Post-tax, Real, NPV8% (US$370.0 million) and IRR (29.0%) results for the Forecast Macro-Economic scenario as the basis for the sensitivity. Figure 110 and Figure 111 depict results in a tornado chart format that highlight the key drivers the project's NPV and IRR are most sensitive to.

Table 126: PERT Chart / Risk Analysis - Inputs (Units)

ID	Metric	Unit of Measurement	Min	P10	Point Estimate	P90	Max
1	Mining – Head Feed Grade.	% (Cg)	5.5%	5.9%	6.2%	6.5%	6.8%
2	Processing Recovery.	%	84.0%	86.4%	88.1%	89.1%	90.0%
3	Processing Concentrate Grade.	% (Cg)	96.5%	96.9%	97.3%	97.4%	97.5%
4	Price: +50 Mesh Superflake® 94%-95% C Graphite.	US$/t of concentrate sold (Real Sept 2023).	1,019	1,133.7	1,225.3	1,317.0	1,432
5	Price:+80 Mesh Superflake® 94%-95% C Graphite.	US$/t of concentrate sold (Real Sept 2023).	866	998	1,101	1,192	1,302
6	Price: +100 Mesh Superflake® 94%-95% C Graphite.	US$/t of concentrate sold (Real Sept 2023).	838	949	1,039	1,136	1,260
7	Price: -100 Mesh Superflake® 94%-95% C Graphite.	US$/t of concentrate sold (Real Sept 2023).	799	889	968	1,068	1,202
8	VIU Adjustment: Grade Premium.	%	3.1%	3.8%	4.3%	4.4%	4.5%
9	Foreign Exchange: MGA.	FX:US$ (Real Sept 2023).	3,801.53	4,036.40	4,223.92	4,411.44	4,646.31
10	Foreign Exchange: ZAR	FX:US$ (Real Sept 2023).	17.00	17.36	17.67	18.03	18.50

ID	Metric	Unit of Measurement	Min	P10	Point Estimate	P90	Max
11	OPEX: Labour	US$/t of concentrate sold (Real Sept 2023).	29.0	31.7	34.1	37.0	40.9
12	OPEX: Utilities	US$/t of concentrate sold (Real Sept 2023).	40.5	46.1	50.6	55.1	60.7
13	OPEX: Stores	US$/t of concentrate sold (Real Sept 2023).	192.9	230.8	257.2	270.7	282.9
14	OPEX: Services	US$/t of concentrate sold (Real Sept 2023).	15.9	17.4	18.7	20.3	22.5
15	OPEX: Overheads	US$/t of concentrate sold (Real Sept 2023).	22.7	24.8	26.7	29.0	32.0
16	Selling Costs	US$/t of concentrate sold (Real Sept 2023).	126.4	138.0	148.7	164.4	185.9
17	Project CAPEX	US$ M (Rea Sept 2023).	149.5	170.9	186.9	199.9	215.0
18	Sustaining CAPEX	US$ M (Real Sept 2023).	143.7	179.1	205.3	224.7	246.4
19	Closure CAPEX	US$ M (Real Sept 2023).	11.5	12.1	12.8	13.8	15.3

Please note that a scenario analysis has been performed on various discount rate increments in section 22.6.3.3, hence have not been considered as part of the sensitivity analysis.

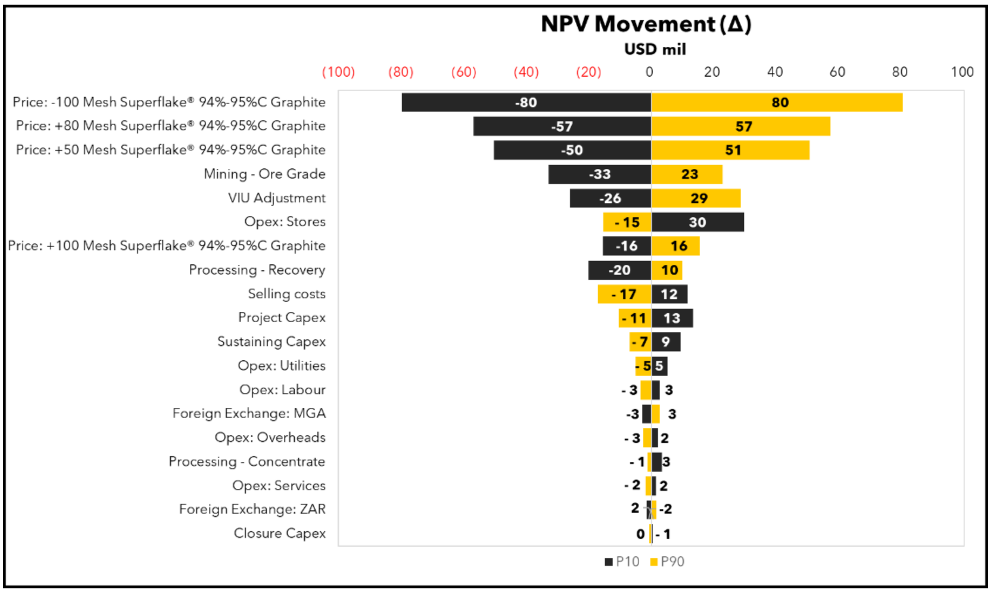


Figure 110: Tornado Chart - NPV Movement

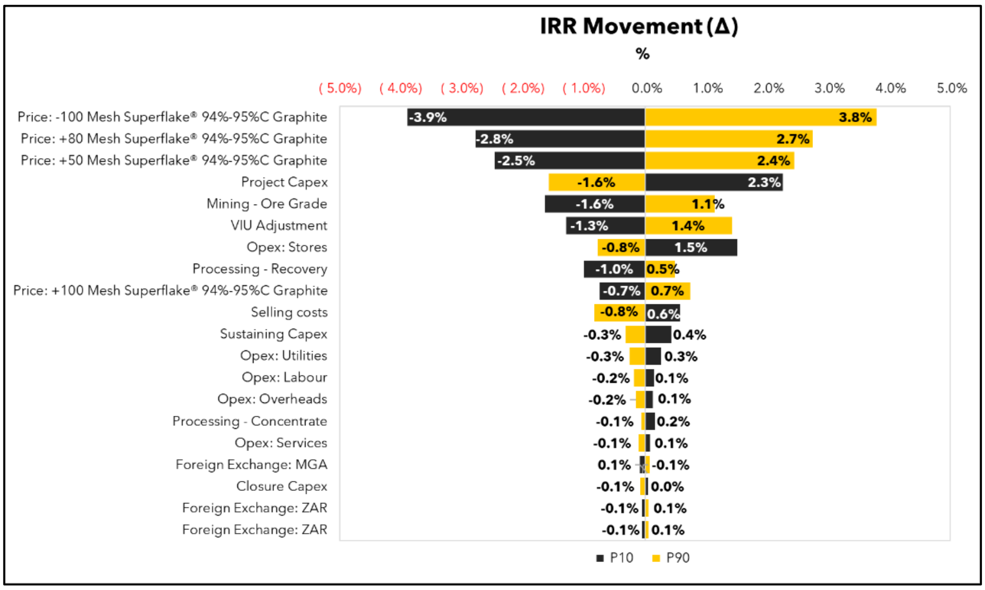


Figure 111: Tornado Chart - IRR Movement

The Project NPV is the most sensitive to the following key drivers:

- Commodity Prices (specifically to -100 Mesh Superflake® graphite concentrate).
- Mining: Ore Grade.
- Operating Cost: Stores.
- Processing: Recovery.

Commodity price ranges are based on the high case and low case forecast price scenarios provided by Benchmark Mineral Intelligence for the each of the 94%-95% Cg reference products (FOB China), exclusive of any VIU adjustment. The P10 weighted average reference price of US$977/t of concentrate sold is 8.3% lower than the point estimate reference price of US$1,066/t of concentrate sold, which amounts to an aggregate Post-Tax $NPV_{8\%}$ downside of US$203 million (IRR reduction of 10.4%). The P90 reference price is US$1,162/t of concentrate sold is 9.0% higher than the point estimate reference price, which amounts to an aggregate Post-Tax $NPV_{8\%}$ upside of US$203 million (IRR increase of 9.3%).

The -100 Mesh Superflake® graphite concentrate is the largest share of the final product distribution (refer to Graph 59), which underpins the larger contribution it makes to the commodity-price movement in NPV, relative to the other products.

The sensitivity assessment of the head feed grade at the P10 and P90 inputs presumes the same quantities of ore will be issued to the concentration plants, regardless of grade. This implies a linear relationship between head feed grade and mass yield to flake graphite concentrate. At P10 levels, a 5.1% reduction in head grade to 5.8% Cg would result in a Post-Tax $NPV_{8\%}$ value erosion of US$39 million (IRR decrease of 2%). At P90 levels, a 3.8% increase in head grade to 6.3% Cg would result in a Post-Tax $NPV_{8\%}$ value accretion of US$29 million (IRR decrease of 1.4%).

Considering the various global geopolitical events that are taking place, an elevated diesel price assumption has been included in the point estimate for the stores operating cost. The P10 assumption offers skewed upside to the project's value assuming the price of diesel would normalise should the geopolitical tensions diminish. The P10 assumption reduces the stores cost by 10.3% which adds US$ 30 M Post-Tax $NPV_{8\%}$ to the project (IRR increases by 1.5%). The P90 assumption increases the stores cost by 5.3% which diminishes the Post-Tax $NPV_{8\%}$ by US$ 15 million (IRR reduces by 0.8%).

The point estimate contained graphite recovery is 88.1% to produce an on-specification concentrate. The sensitivity assessment of the recovery at the P10 and P90 inputs presumes that a change to recovery rates will affect the mass yield to flake graphite concentrates and that the concentrate grades remain unchanged. A movement of 1.9% from the point estimate to a P10 recovery assumption of 86.4% results in the Post-Tax $NPV_{8\%}$ reducing by US$23 million. An improvement of 1.1% from the point estimate recovery to 89.1% results in a Post-Tax $NPV_{8\%}$ increase of US$13 million.

Foreign exchange exposure is not a sensitive driver to the project given that the capital and operating cost estimates are substantially made up of US dollar-based estimates.

The Project IRR is the most sensitive to the following key drivers:

- Commodity Prices (specifically to -100 Mesh Superflake® graphite concentrate).
- Project CAPEX.
- Mining: Ore Grade.
- Operating Cost: Stores.

Project CAPEX affects the IRR to a greater extent that it does NPV, since IRR is majorly affected by the first 10 years of free cash flow. If the Project can reduce the upfront capital outlay requirement by 8.6% to US$170.8 million, it is possible to improve the Post-Tax $NPV_{8\%}$ by US$13 M and IRR by 2.2% respectively.

22.6.2 Robustness Assessment

The assessment of robustness has been performed on the forecast macro-economic scenario set of results, i.e. the base case. The purpose of the robustness assessment is to deterministically evaluate and analyse how a combination of macro-economic and project economic scenarios can influence key economic results. This is achieved by assigning each key project driver as either an exogenous or endogenous variable as shown in Table 127. An exogenous variable is not typically within the reasonable control of the project team (e.g. metal prices). An endogenous variable is largely within the reasonable control of the project team (e.g. on-site costs).

Table 127: Exogenous and Endogenous Variables

Exogenous Variables	Endogenous Variables
Physicals: Mining Head Grade	Physicals: Recovery
Commodity Price: 94-95% Cg Reference Price	Physicals: Concentrate Grade
Commodity Price: Premium	Operating Costs: Labour
Foreign Exchange	Operating Costs: Utilities
	Operating Costs: Stores
	Operating Costs: Services
	Operating Costs: Overheads
	Selling Costs
	Project CAPEX
	Sustaining CAPEX
	Closure CAPEX

Utilising the P25, Point Estimate and P75 parameters of the of the variables established in Table 128**Error! Reference source not found.**, a matrix table is created to evaluate a combination of scenarios to assess the robustness of the business case to movements in exogenous variables and the extent to which the project team is able to effectively control the endogenous variables to ensure sustained profitability and capital intensity.

Table 128 defines the nine combinations of scenarios evaluated and Table 129 subsequently evaluates the results of the scenarios against key economic result metrics.

The analysis shows that the Molo Expansion Project is value accretive in all nine scenarios examined, which is indicative of a robust business case.

The 'value engineering case' highlights the importance of good execution, governance and operational performance. In the value engineering scenario, the Post-Tax NPV8% increases to US$415.6 million from the base case of US$370.0 million, while the IRR improves from 29.0% to 32.2%. This is largely attributable to a 5.9% reduction in operating costs, a 4.9% reduction in Project Capex and 7.5% decrease in Sustaining Capex.

Table 128: Definitions of Scenario-Based Tests of Robustness

		Endogenous Parameters		
Exogenous Parameters	**Item**	**High**	**Medium**	**Low**
	High	Favourable Market Conditions. Excellent Project Performance. **(Theoretical Best Case).**	Favourable Market Conditions. Planned Project Performance.	Favourable Market Conditions. Poor Project Performance.
	Medium	Forecasted Market Conditions. Excellent Project Performance. **(Value Engineering Case).**	Forecasted Market Conditions. Planned Project Performance. **(Base Case).**	Forecasted Market Conditions. Poor Project Performance.
	Low	Weak Market Conditions. Excellent Project Performance.	Weak Market Conditions. Planned Project Performance.	**Weak Market Conditions.** **Poor Project Performance.** **(Theoretical Worst Case).**

Table 129: Results of Scenario-Based Tests of Robustness

		Endogenous Parameters					
Exogenous Parameters	**Item**	**High**		**Medium**		**Low**	
	High	Post-tax $NPV_{8\%}$:	US$645.9M	Post-tax $NPV_{8\%}$:	US$600.2M	Post-tax $NPV_{8\%}$:	US$556.9M
		Post-tax IRR:	43.1%	Post-tax IRR:	39.6%	Post-tax IRR:	36.8%
		Payback:	2.0 years	Payback:	2.2 years	Payback:	2.4 years
		Peak Funding:	US$159.8M	Peak Funding:	US$170.5M	Peak Funding:	US$178.3M
	Medium	Post-tax $NPV_{8\%}$:	US$415.6M	Post-tax $NPV_{8\%}$:	US$370.0M	Post-tax $NPV_{8\%}$:	US$328.8M
		Post-tax IRR:	32.2%	Post-tax IRR:	29.0%	Post-tax IRR:	26.4%
		Payback:	2.8 years	Payback:	3.1 years	Payback:	3.4 years
		Peak Funding:	US$167.6 M	Peak Funding:	US$178.5M	Peak Funding:	US$186.5 M
	Low	Post-tax $NPV_{8\%}$:	US$187.3M	Post-tax $NPV_{8\%}$:	US$142.4M	Post-tax $NPV_{8\%}$:	US$102.9M
		Post-tax IRR:	20.2%	Post-tax IRR:	17.1%	Post-tax IRR:	14.6%
		Payback:	4.5 years	Payback:	5.2 years	Payback:	6.1 years
		Peak Funding:	US$176.0M	Peak Funding:	US$186.9M	Peak Funding:	US$195.2M

22.6.3 Scenario Analysis

The following alternative scenarios have been assessed to enhance the transparency of the economic analysis, specifically for factors that are outside of the control of the entity and have not been considered as part of the deterministic sensitivity analysis environment in Section 22.6.1.

22.6.3.1 Large Scale Investment Status Not Obtained

Should the LGIM certification not be obtained:

- The associated beneficial tax regime discussed in sections 22.4.10.2 and 22.4.5.1 would not accrue to the Project.
- The effective corporate income tax rate would increase from 10.3% to 20.2% of taxable income, and the effective Malagasy government royalty rate would increase from 3.7% to 3.8% of revenue.

The result of the alternative scenario is reported in Table 130.

Table 130: Alternative Scenario where LGIM Status is Not Obtained

Metric	Unit of Measurement	Base Case (with LGIM)	Adverse Case (without LGIM)
$NPV_{8\%}$ (Post-Tax)	US$ million	370.0	316.0
IRR (Post-Tax)	%	29.0%	26.8%
Undiscounted Payback Period	Years (from first Phase 2 concentrate production)	3.1	3.2
Peak Funding Requirement	US$ million	178.5	177.2

22.6.3.2 Applicability of Transformation Entity is Repealed

Should the interpretation of a Transformation Entity be adversely affected by the proposed Implementation Decree, such that Project's processing activities per the Law on the Mining Code (2023) do not constitute a "Transformation of mineral substances" as advised by John W Ffooks & Co ("JWF"), who have acted in the capacity of in-country legal experts advising on tax-related matters:

- The entity which is the Holder of a Mining Permit will revert to being responsible for both mining and processing activities under definition of an "integrated mining activity".
- The associated beneficial tax treatment as discussed in section 22.4.10.1 and 22.4.5 will not accrue to the project.
- The effective corporate income tax rate would increase from 10.3% to 20.1% of taxable income, which would be partially offset by the effective Malagasy government royalty rate would decrease from 3.7% to 2.7% of revenue.

The result of the alternative scenario is reported in Table 131.

Table 131: Alternative Scenario where Applicability of Transformation Entity is Repealed

Metric	Unit of Measurement	Base Case (with Transformation Entity)	Adverse Case (without Transformation Entity)
$NPV_{8\%}$ (Post-Tax)	US$ million	370.0	329.2
IRR (Post-Tax)	%	29.0%	27.4%
Undiscounted Payback Period	Years (from first Phase 2 concentrate production)	3.1	3.1
Peak Funding Requirement	US$ million	178.5	179.0

22.6.3.3 Discount Rate Scenarios

Refer to Table 132 where discount rates at 6%, 8% (that aligns to the results presented in Table 132), 10% and 12%, have been applied to determine the resulting NPV.

Table 132: Discount Rate Scenarios

Real Discount Rate Scenario	Units of Measurement	Net Present Value		
		Forecast	3-Year Trailing Average	Spot
6%	US$ million	484.5	397.8	254.3
8% (Base Case)	US$ million	370.0	299.8	182.7
10%	US$ million	283.4	225.8	128.7
12%	US$ million	217.1	169.2	87.5

23 ADJACENT PROPERTIES

Most recently, BlackEarth Minerals announced a 20 - 34 Mt exploration target at 10% - 20% TGC at their Ianapera Graphite Project, which is located just 10 km north of Molo. BlackEarth Minerals now Evion Group, (www.eviongroup.com) holds the Maniry Graphite project located 60 km south of Molo and completed a Bankable Feasibility Study.

No relevant information from adjacent properties was used in the compilation of this technical report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Hazard Study Reports

Two Hazard reports, namely, Hazard on Graphite Dust Exposure and Hazard on Silica Dust Exposure, were completed by Globesight. The reports are based on the Hazard study completed for Phase 1 of the FS, which also applies to the expansion:

- Crushing.
- Milling and flash flotation.
- Rougher flotation.
- Cleaner circuits.
- Thickening.
- Filtration and drying.
- Bagging.
- Tailings handling.
- Reagent storage and supply.
- Supply of services.
- Possible causes.
- Consequences.
- Preventative avoidance measures.
- Protective mitigating measures.
- Actions needed if the measures were inadequate.
- Determination of causes of the hazard.
- Determination of the consequences of the hazard.
- Evaluation of any existing safeguards / controls, (preventative or mitigating).
- New safeguards if existing are judged inadequate.
- Action and responsibilities to execute the action.

24.2 Lessons Learnt from Phase 1

The Expansion Project actively tracks the learnings from Phase 1 and these learnings, available at the time during drafting this FS, have been incorporated into the expansion planning and designs.

25 INTERPRETATION AND CONCLUSIONS

The results of the FS demonstrate that the expansion is financially sound and remains economical within the limits of the sensitivity analysis performed on CAPEX and OPEX costs as well as on selling price.

25.1 Geology

The Company's 2011 exploration program delineated a number of new graphitic trends in southern Madagascar. The resource delineation drilling undertaken during 2012 to 2014 focused on only one of these, the Molo Deposit, and this has allowed for an Independent, CIM compliant, updated resource statement for the Molo deposit.

The total Measured and Indicated Resource is estimated at 100.37 Mt, grading at 6.27% C. Additionally, an Inferred Resource of 40.91 Mt, grading at 5.78% C is stated. When compared to the November 2012 resource statement (Hancox and Subramani, 2013), this shows a 13.7% increase in tonnage, a 3.4% decrease in grade, and a 9.8% increase in graphite content. The reason for the increase in tonnage is due to the 2014 drilling on the previously untested north-eastern limb of the deposit, which added additional new resources. Additionally, 23.62 Mt, grading at 6.32% Carbon, have been upgraded by infill drilling from the Indicated to Measured Resource category.

25.2 Mining

Mineral reserves of 53,746 kt, (Proven 21,334 kt at 6.25% C head grade and Probable 32,412 kt at 6.09% C head grade) have been declared in the Feasibility Report with an average grade of 6.15% C head grade. Based on the information contained in the Technical Report, it is possible to economically mine this deposit.

The mine planning scenario described in this section has included Inferred Resources in the conceptual mine planning. While the Inferred Resource is included in the pit optimisation model, the percentage of the ore considered to be associated with an Inferred Resource is 4.5% of total ROM ore. This renders the Inferred Resource category as a minor contributor to the total mineable ore.

The open pit mining operations were planned by utilising an unconstrained LoM production scenario. The scenario indicates that a 150 ktpa carbon concentrate plant can be sustainably supported by the orebody over the planned LoM period.

25.3 Tailings

The final tailings will be conveyed to the final tailings stacking conveyor which discharges and distributes the material at the final tailings deposition area. Meanwhile waste rock will be trucked from the pit to the conveyor feeding the final tailings stacking conveyor where it will be loaded onto the belt. In the next phase of development detailed design will be completed, complete with environmental and social impact assessment and closure planning.

25.4 Permitting

While the major permits have been granted for the Project, the Global Environmental Permit and the Mining Permit, various supplementary sectoral permits will need to be obtained.

25.5 Metallurgical Test Work

Comprehensive metallurgical test programs culminated in a process flowsheet that is capable of treating the Molo ore using conventional and established mineral processing techniques. This was done during the Molo Phase 1 study. The optimized flowsheet and conditions shown in Figure 24 was chosen for the variability flotation program. Each variability composite was subjected to the flowsheet using this flowsheet with minor adjustments to the primary grind time, flotation times and reagent dosages.

Process risks associated with the variability with regards to metallurgical performance have been mostly mitigated through the addition of an upgrading circuit. The upgrading circuit treated the combined concentrate after the secondary cleaning circuit. Reduced flake degradation and an improved process flexibility may be obtained by employing separate upgrading circuits for the coarse and fine flakes. This could be reviewed during detail design for the expansion.

26 RECOMMENDATIONS

Whilst the results of this FS demonstrate that the expansion to 150 ktpa is financially sound there are implementation options which may further optimise the economic results. The following recommendations have been identified throughout the FS:

26.1 Metallurgical Test Work

No additional external metallurgical test work has been identified for the expansion project. However, the recommended test work below can be carried out by current operations in the next phase.

- Complete a review of the Phase 1 operations data against the test work assumptions made in this FS.
- Calibrate laboratory testing procedures to align with the current experience on the Phase 1 concentrator.
- Quantify the impact of the mill operating conditions in the cleaner circuit on metallurgical performance.
- Perform grind mill optimization using the Phase 1 concentrator (pulp density, media type, mill speed).
- Complete a review of current graphite processing technology and equipment selection as the body of knowledge has changed as the market has grown.

26.2 Recovery Methods

Compare performance of the Phase 1 concentrator with original mass and water balance and process design criteria to optimize the circuit and achieve the specified KPIs (Key Performance Indicators).

Perform analysis of the commissioning process of the Phase 1 plant to determine areas for improvement and to develop a strategy to reduce mechanical and metallurgical challenges during Phase 2.

Review the overall process prior to front end engineering design to ensure the latest graphite processing knowledge is taken into account for Phase 2.

26.3 Infrastructure

Further study work to be conducted into the hybrid hauling / conveying solution for waste material to evaluate cost benefits.

The following are items identified that will require more testing / analysis and investigation to allow a more informed and applicable design of the TSF:

- The TSF development is reliant on a material balance that allows for sufficient course / waste rock material to be used as structural material. The current material sources are the material sourced within the basin and the overburden / waste rock generated from mining activities. As part of further planning, it must be formalised and agreed what the cut-off grade will be of ore / waste material that is to be used (sterilized) in the TSF.

- An in-depth investigation and analysis of the discard, chemical additives and tailings material is required in terms of the physical and geochemical reactions. This will contribute to the optimisation in terms of TSF stability and the integration into the dust suppression process on the TSF.

Water quality and quantity data is required to provide a baseline for comparison once the Molo mine is commissioned. To provide the necessary baseline data, regular ground and surface water quality monitoring must be carried out leading up to the date when the Molo mine will be commissioned. Additional production and monitoring boreholes must be installed. This also should include the installation of flow meters on relevant pipelines to verify the dynamic water balance with measured flow rates during operations.

Quantitative and predictive water balance, ground water and geochemical analyses should be undertaken at regular intervals to update the water management plan.

The raw water make-up needs to be re-evaluated and the potential re-use and management of mine water must be verified and optimized through a dynamical water balance model update.

A detailed assessment of surrounding aquifers and wellfield abstraction for the project Expansion, including borehole drilling, testing and ground water abstraction. Wellfield abstraction must be sustainable in terms of available resources, groundwater recharge and the social and environmental aspects of the area.

The updated water management plan must mitigate any risks impacting water quality. Potential discharge from the PCDs (pollution control dams) need to be re-evaluated, carefully controlled and monitored. PCD water under discharge conditions, and downstream surface water at the compliance point will comply with Madagascar and international criteria.

Following the detail design phase of the TSF, a dam breach assessment will be required in line with GISTM standards to determine potential downstream impacts on incremental losses such as surface water resources, (i.e., environment, health, social and culture, infrastructure, and economics).

26.4 Environmental, Social and Permitting

The expansion will require application for amendments as and when applicable to the stand-alone documentation from the original Phase 1 ESIA, SEMPs, RAP, SDP, Tree Clearance Permit, and Water Uses.

Application for all necessary permits, authorisations, and land tenure (refer section 20.4 and 20.5) is necessary prior to commencement of the Expansion construction and implementation. These require lengthy regulatory processes and should be initiated timeously.

Security of land tenure is a process and is estimated to take 6 to 9 months, thus this process should be commissioned as early as possible. The total additional property area to accommodate the expansion over and above the Phase 1 requirement is a further 231 ha.

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DRA, 2014. Soalara Port vs Fort Dauphin Port Trade-off Study. South Africa.

Energizer, 2012. Baseline Studies: Chapter 2: Soil, Surface Water and Groundwater Studies. Madagascar.

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GCS Water and Environment (GCS), 2013. Madagascar Environmental Legal Protocol. South Africa.

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International Finance Corporation (IFC), 2012. Performance Standards and Guidance Notes. Available from http://www.ifc.org/wps/.

SRK, 2015. Molo Graphite Project Madagascar: Tailings Storage Facility Feasibility Study. South Africa.

UWP, 2012. Final Study Report Revision 00. November 2012. Green Giant Graphite Mine Hydrological Desktop Study. Prepared for DRA Mineral Projects.

Vorster, L, 2014. Provisions Governing Mining Exploration: Molo Project Law Summary Extract of Madagascar Legal Protocol. South Africa.

World Bank Group, 2007. WBG Environmental, Health, and Safety Guidelines. Available from http://www.ifc.org/wps/.

27.7 TSF Reference Material

For reference the following pertinent documents was accessed for consideration of the TSF design:

- Molo Graphite Project Madagascar: Tailings Storage Facility Site Selection Study, *SRK Consulting (Report Number 474874/1)*, June 2014.
- Molo Graphite Project Madagascar: Final Tailings Storage Facility Feasibility Study, *SRK Consulting (Report Number 474874/2)*, October 2014.
- Molo Graphite Project Madagascar: Tailings Storage Facility Feasibility Study - Final Report, *SRK Consulting (Report Number 474874/4)*, February 2015.
- MOLO Feasibility Study National Instrument 43 - 101 Technical Report, Next *Source Materials Inc (Report Number CMGDM1770 – TRPT - 001 Rev 0)*, May 2019.
- Molo Graphite Project, Fotadrevo, Province of Toliara, Madagascar Technical Report NI 43 – 101, *Energizer Resources Incorporated*, September 2014.
- Geotechnical design for molo graphite project, Open House Management Solutions (Pty) Ltd (Report Number LC/140911/MO5047/B24), September 2014.

28 CERTIFICATES OF AUTHORS

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Johann de Bruin
Address	277 The Hillside Road, Lynnwood, Pretoria, Gauteng
Occupation	Director
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report Effective date: September 1, 2023
Qualifications	I graduated with a B Eng (Civil Engineering) degree from the University of Pretoria in 1992. I am a registered Professional Engineer with ECSA, Registration Number: Pr.Eng 970123 I completed the Director Development Programme through the University of Cranfield (England) in 2013. My professional memberships include: University of Pretoria: Member of the Advisory Board of the Engineering Faculty to ensure that the curriculum remains aligned with global standards and practices. I am also a part-time lecturer for final year students in engineering and project management. Engineering Council of South Africa (ECSA). South African Institute of Civil Engineers (SAICE). For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I visited the site on various dates in 2015, 2016, 2017, 2022 & 2023
Responsibilities	I am responsible for the following sections: Overall report.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have been involved in this project for multiple years and various phases of the project lifecycle.
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading
Dated	16 January 2024
Signature	Johann De Bruin Director Erudite Projects (Pty) Ltd

Summary of Recent Experience:

Year	Client	Commodity	Type	Description
2019	Two Rivers Platinum	PGM	Project	Development of a crushing and materials handling project
2019	Baobab Vanadium	Vanadium	Concept	Analysis of project development strategies
2019	IAMGOLD	Gold	BFS	Product recovery project
2018	South32	Manganese	BFS	Analysis of multiple product handling solutions
2018	Anglo Platinum	PGM	FS	Materials handling and plant optimisation
2018	Anglo Platinum	PGM	FS	Multiple plant improvement studies
2000 - 2017	Multiple	Various	BFS	Development of multiple Feasibility studies in international jurisdictions

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Hector Mapheto
Address	277 Hillside Streer, Lynwood, Tshwane, 0081, South Africa
Occupation	Director
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report Effective date: September 1, 2023
Qualifications	I graduated with a B Sc. Eng. (Chemical Engineering) degree from the University of Cape Town in 2004. I am a registered Professional Engineer with ECSA, Registration Number: Pr. Eng. 20100105 My professional memberships include: Council Member of Mine Metallurgical Managers Association (MMMA) Engineering Council of South Africa (ECSA). South African Institute of Mining and Metallurgy (SAIMM). For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I visited the site in 2023.
Responsibilities	I am responsible for the following sections: Section 17.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have no prior involvement with this Project.
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.
Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
Dated	16 January 2024
Signature	Hector Mapheto Director

	Erudite Projects (Pty) Ltd

Summary of Recent Experience:

Year	Client	Commodity	Type	Description
2019	Two Rivers Platinum	PGM	Project	Development of a crushing and materials handling project
2019	Baobab Vanadium	Vanadium	Concept	Analysis of project development strategies
2020	Two Rivers Platinum	PGM	Project	Milling and Float Plant Expansion
2020	AssOre	PGM	Scoping	Chrome Tailing Recovery Plant
2020	Anglo Platinum	PGM	FS	Multiple plant improvement studies
2021 - Present	Multiple	Various	BFS	Development of multiple Feasibility studies in international jurisdictions

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

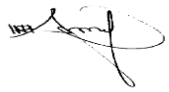
Name	Schalk Pienaar (Pr.Eng).
Address	277 Hillside Street, Lynnwood, Tshwane, 0081, South Africa.
Occupation	Civil and Structural Engineer.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report. Effective date: September 1, 2023
Qualifications	Schalk is a registered Professional Engineer (Pr.Eng.) with a Bachelor of Science degree in Civil Engineering. I am a Professional Engineer registered with Engineering Council of South Africa (ECSA) – 20020073 These qualifications are complemented by over 20 years of working experience across various sectors in the civil and structural engineering industry. Schalk has an array of experience that range from the construction management to the various stages of design development of complex industrial Projects. Schalk has worked on mining, petrochemical, and infrastructure development Projects across the African continent. For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	N/A. No existing infrastructure to inspect.
Responsibilities	I am responsible for the following sections: General Infrastructure as part of section 18.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have no prior involvement with this Project.
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading
Dated	16 January 2024
Signature	Schalk Pienaar (Pr Eng) Civil and Structural Engineer Erudite Projects (Pty) Ltd

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Hercu Smit.
Address	Hillside Offices, 277 The Hillside, Lynnwood, 0081.
Occupation	Director.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report. Effective date: September 1, 2023
Qualifications	I graduated with a B-Eng (Electrical Engineering) degree from the University of Pretoria in 1996. I am a registered Professional Engineer with ECSA, Registration Number: Pr.Eng 2000038 as well as a Senior Member of the SAIEE : Registration Number : 11938. I completed the Management course at the Graduate School of Management, University of Pretoria in 2000. My professional memberships include: Engineering Council of South Africa (ECSA). South African Institute of Electrical Engineers (SAIEE). For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	N/A. No existing infrastructure to inspect.
Responsibilities	I am responsible for the following sections: Section 18: Project Infrastructure (Electrical).
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have been involved in this project for multiple years and various phases of the project lifecycle.
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
Dated	16 January 2024.
Signature 	Hercu Smit. Director. Erudite Projects (Pty) Ltd.

Summary of Recent Experience:

Year	Client	Commodity	Type	Description
2023	Two Rivers Platinum	Bulk Power	Project	132kV Overhead Line & 132/11kV, 3 x 40MVA Substation
2023	MC-Mining	Coal	Project	Development of a cola handling project
2022	Two Rivers Platinum	PGM	Project	Development of a milling & float and materials handling project
2019	Two Rivers Platinum	PGM	Project	Development of a crushing and materials handling project
2019	Baobab Vanadium	Vanadium	Concept	Analysis of project development strategies
2019	IAMGOLD	Gold	BFS	Product recovery project
2018	South 32	Manganese	BFS	Analysis of multiple product handling solutions
2018	Anglo Platinum	PGM	FS	Materials handling and plant optimisation
2018	Anglo Platinum	PGM	FS	Multiple plant improvement studies
2003 - 2017	Multiple	Various	BFS	Development of multiple Feasibility studies in international jurisdictions

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Alkie Marais.
Address	113 Severn Place, Rietvalleirand, Pretoria, South Africa.
Occupation	Principal Hydrogeologist.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report . Effective date: September 1, 2023
Qualifications	I graduated with a M.Sc. (Geohydrology) degree from the University of Free State in 1998. I am a registered Professional Scientist (Earth Science) with the South African Council for Natural Scientific Professionals (SACNASP), Registration Number: 400012/06. My professional memberships include the Groundwater Division of South Africa. For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I visited the site in 2014.
Responsibilities	I am responsible for the Water and Environmental aspects in Section 18 and 20.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have no prior involvement with this Project.
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.
Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading
Dated	16 January 2024.
Signature	Alkie Marais. Principal Hydrogeologist. Geostratum Groundwater and Geochemistry Consult (Pty) Ltd.

Summary of Recent Experience:

Year	Client	Commodity	Type	Description
2023	Kamoa Copper	Copper	Operational	Development of a mine dewatering strategy for Kakula Mine in the DRC
2023	Robex Resources Inc	Gold	DFS	Definitive Feasibility Study for a gold mine in Liberia. Groundwater component, ESIA and dewatering study.
2023	Lemur Resources	Coal	FS	Imaloto Coal Feasibility Study in Madagascar. Undertook the mine water component.
2023	UMS Group	Copper	Implementation	Boa Esperanca Project in Brazil. Assisted UMS Group with review and management of the water components of the mine project
2023	ERG	Copper	FEL3/Feasibility	Review study for the revised Comide Project in the DRC. Was Project director/manager on Water for previous Feasibility Studies in 2017 to 2018 and 2021.
2004 to 2022	Multiple	Various	PFS/BFS/DFS	Development of multiple Feasibility studies in Africa according to international (and local) criteria. Managed the Water and Environmental components. These include the Molo Graphite Project in 2014 to 2015 and 2017.

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Philip John Hancox.
Address	30 Seventh Avenue, Parktown North, Randburg, Gauteng, South Africa.
Occupation	Director.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report. Effective date: September 1, 2023
Qualifications	I graduated with a B.Sc. Honours (1990) and Ph.D. (1998) from the University of Witwatersrand - Johannesburg. I am a member in good standing of the South African Council for Natural Scientific Professions (SACNASP No. No. 400224/04) as well as a Fellow of the Geological Society of South Africa. I have practiced my profession since 1998. My professional memberships include: Fellow of the Geological Society of South Africa (GSSA) Fellow of the Society of Economic Geologists (SEG) Lifetime Member of the Geostatistical Association of South Africa (GASA). Core Member of the Prospectors and Developers Association of Canada (PDAC). Member of the Palaeontological Society of South Africa (PSSA). For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I visited the site on various dates in May of 2012 and 2013.
Responsibilities	I am responsible for the following sections: Sections 6,7,8,9,10 and parts of Section 1, 25 and 26.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have no prior involvement with this Project.
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
Dated	16 January 2024.
Signature 	Philip John Hancox. Director. Caracle Creek International Consulting (Pty) Ltd.

Summary of Recent Experience:

Year	Client	Commodity	Type	Description
2023	Zeb Nickel Corporation	Ni/PGM	Technical Report	NI 43-101 Technical Report on the Zeb Nickel Project
2023	ICVL	Coal	CPR	Competent Persons Report and coal resource estimation on the 7521C Mining Concession, Tete Province, Mozambique
2022	Next Source	Graphite	PEA	National Instrument 43-101 Technical Report on the Molo Graphite Project
2021	Blue Rhino Capital Corporation	Ni	Technical Report	Technical Report on the Zebediela nickel sulphide Project
2021	Transnet	Coal	Technical Report	Segment Strategy – Coal
2021	Domino Capital	Caol	DD	Project lead on due diligence on Vale's coal assets in Mozambique
2005 - 2017	Multiple	Various	Technical Reports	Development of multiple Technical Reports in international jurisdictions

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Sivanesan (Desmond) Subramani.
Address	90 Beryl Avenue, Bramley North, Sandton, 2090.
Occupation	Principal Geologist – Mineral Resources.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report. Effective date: September 1, 2023
Qualifications	I am a graduate of the University of KwaZulu Natal, with a BSc Honours in Geology and Economic Geology. I am a registered professional member of the South African Council for Natural Scientific Professions (Reg. No. 400184/06). I am a member of the Geological Society of South Africa, and a member of the Geostatistical Association of Southern Africa. For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I visited the site from the 15th to 19th February 2014
Responsibilities	I am responsible for the following sections: Section 14 – Mineral Resource Estimates.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I co-authored the report entitled Molo Graphite Project, Fotadrevo, Province of Toliara, Madagascar Technical Report NI 43-101 dated January 15, 2013.
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.
Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
Dated	16 January 2024
Signature	Subramani

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Oliver Peters.
Address	102 Milroy Drive, Peterborough, Ontario, K9H 7T2, Canada.
Occupation	President & Principal Metallurgist.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report. Effective date: September 1, 2023
Qualifications	I graduated with a M.Sc (Mineral Processing) degree from the RWTH Aachen, Germany in 1998 and an MBA from Athabasca University 2007. I am a registered Professional Engineer with the Professional Engineers of Ontario (PEO) # 100078050. For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I have not visited the site.
Responsibilities	I am responsible for the following sections: Section 13 and parts of Sections 1, 25, and 26.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have been involved in the process development of the project from initial scoping tests to FEED study.
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.
Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
Dated	16 January 2024.
Signature	Oliver Peters. President & Principal Metallurgist. Metpro Management Inc.

Examples of Recent Graphite Experience (only clients have been included my name in market announcements)

Year	Client	Type	Description
2022/2023	Sovereign Metals	PFS	Process development
2019-/2022	Northern Graphite		Process optimization support
2013-2021	Nouveau Monde	Scoping to FS	Process design, engineering support, demo plant design and commissioning
2021-2023	LoMiko	Scoping & PEA	Process development
2021-2023	Evolution Battery Minerals	FEED & DFS	Process optimization and engineering support
2018-2023	Misc	FS	Due diligence study

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Ruan Albert Daffue.
Address	Building 1 Maxwell Office Park,37 Magwa Crescent, Waterfall City, Midrand, Gauteng, South Africa, 2090.
Occupation	Director.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report. Effective date: September 1, 2023
Qualifications	I graduated with a B.Eng (Industrial) degree from Stellenbosch University in 2010 and an M.Sc.Eng (Management) degree from Stellenbosch University in 2012. I am a member of the South African Institute of Mining and Metallurgy (SAIMM), with membership number M709830. I am a 'Competent Person' as defined in the SAMREC Code which allows me to act as a Qualified Person under ROPO. I have more than 10 years' experience in the Metals and Mining industry, of which the majority of these years were spent in an advisory capacity. For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I have not visited the site, reliance was therefore made on previous site visits undertaken by Qualified Persons to the Right area, as well as date stamped photographs of the present site conditions.
Responsibilities	I am responsible for the following sections: Section 21 – Capital and Operating Costs, and Section 22 – Economic Analysis.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have no prior involvement with this Project.

Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with the same.
Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
Dated	16 January 2024.
Signature:	Ruan Albert Daffue. Director. Practara (Pty) Ltd.

Summary of Recent Experience:

Year	Client	Commodity	Type	Description
2023 - Present	Global Atomic Corporation	Uranium	FS	Economic Analysis of a U3O8 Project in Niger.
2023 – Present	Verdant Minerals	Phosphate	Scoping	Economic Analysis for a Phosphate mine in Australia.
2023	South32	Manganese	PFS	Economic Analysis of the Mamatwan mine in RSA.
2023	Assore	Chromite	Scoping	Economic Analysis & Costing for a producing mine in RSA.
2022	CD Capital	PGEs and Base Metals	PFS	Economic Analysis for the Suhanko Arctic project in Finland.
2021	CD Capital	Copper	Scoping	Economic Analysis & Costing for the Los Calatos project in Peru.
2019	Barrick (then Acacia)	Gold	PFS	Economic Analysis and Costing for the Nyabirama mine in Tanzania.
2018	Platinum Group Metals	PGE	FS	Economic Analysis and Costing for the Waterberg project in RSA.

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Nico Hamman.
Address	472 Botterklapper Street, Block@Nature, Block B, Die Wilgers, Pretoria East, 0184.
Occupation	Principal Technologist – Engineering, Dams and Tailings.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report. Effective date: September 1, 2023
Qualifications	National DipLoMa: Civil Eng Technikon Pretoria, South Africa 2001. Baccalareus Technologiae: Civil Eng. - Geotechnical Technikon Pretoria, South Africa 2002. I am a registered Professional Technologist with ECSA, Registration Number: Pr.Tech Eng 201170142. Approved as Professional Person (APP) for Category II dam safety inspections and design by Department Water Affairs. My professional memberships includes: Engineering Council of South Africa (ECSA). South African Institute of Civil Engineers (SAICE). South African National Council of Large Dams (SANCOLD). Approved as Professional Person (APP) for Category II dam safety inspections and design by Department Water Affairs. For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I visited the site on dates in 2023
Responsibilities	I am responsible for the following sections: Section 18, the tailings content.

Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have no prior involvement with this Project
Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.
Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
Dated	14 November 2023.
Signature	Nico Hamman. Principal Technologist – Engineering, Dams and Tailings Eco Elementum Engineering (Pty) Ltd.

Summary of Recent Experience:

KEY EMPLOYMENT AND EXPERIENCE			
DATE	**EMPLOYER**	**DESCRIPTION**	**PROJECT ROLE**
Historical and current employment			
Current	Principal Technologist – Engineering, Dams and Tailings (Eco Elementum Engineering (Pty) Ltd)	Function as an APP and tailings specialist. While focused on the fields of tailings, pollution control and earth embankment dams where the works involves the remedial design, monitoring, rehabilitation, auditing and maintenance of existing infrastructure and design of new measures and structures as and when required.	Engineering Technologist and APP.
2006-2016	SRK Consulting	Infrastructure related to the tailings production from the mining industry, including the investigations, designs, construction, maintenance and auditing of tailings storage facilities, return water and holding dams.	Senior Engineering Technologist and APP.
2002-2006	Newtown Landscape Architects	Bulk earthworks, civil designs and draught works. Assisted with the engineering components required for the various structures of public spaces, including wetlands, parks, industrial and residential areas.	Senior Landscape Technologist.
2000-2002	Centre for Landfill Technology	Soils and municipal waste laboratory, responsible for implementation and testing of samples to specifications and codes.	Junior Laboratory Technician.
1999-2000	Jaco Swart Consulting Engineers	Construction of 110 km irrigation steel and AC pipeline with diameters from 1.8 m to 0.3 m, responsible for QA for pipelines and installation, certificates and quantity verifications.	Junior Resident Technologist.
Tailings Facility Engineering			
Concept development, risk, and site selection matrix.			
Site investigations, life of mine planning and tailings deposition optimisation.			
Detail design and development of appurtenant infrastructure e.g. return water dams.			

Hazard, risk, and stability assessment in terms of local and international accepted standards and specifications.
Monitoring and professional assistance in the design and implementation of operational requirements for the TSF.
Compilation and updating of the Mandatory code of practice, Operations manual and Emergency preparedness planning.
Dam Engineering
APP for Category II Small and Medium Dams: Safety Inspections and design for the Dam Safety Office.
Dam audits and remedial designs.
Stream and wetland rehabilitation including flood plain recovery and soil remediation and stabilisation.
Monitoring and professional assistance on various pollution and water dams.
Assist with the design and implementation of earth embankment dams.
Engineering
From preliminary site investigations to detail and final design work on various water and tailings facilities and associated infrastructure.
Design small civil structures including gabion works, concrete weirs and reservoirs.
Effluent and tailings facilities rehabilitation, including storm water and seepage mitigation.

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 2023-07-25 Part 8.

Name	Eugène de Villiers.
Address	Farm Mooirust 613, Heilbron, 9650, Free State South Africa.
Occupation	Director.
Title and Effective Date of Technical Report	Molo Expansion FS, National Instrument 43-101 Technical Report. Effective date: September 1, 2023
Qualifications	I graduated with a B Eng (Mining Engineering) degree from the University of Pretoria in 1993. I obtained my Mine Managers Certificate of Competency for Coal mines in 1996. I have done various courses through Universities and other educational institutions. My professional memberships include: SAIMM (South African Institute of Mining and Metallurgy) - (700348). ECSA (Engineering Council of South Africa - Pr.eng (20080066). SACMA (South African Coal Managers Association) - Ordinary Member (1742). For the purposes of NI 43-101, this person is a qualified person.
Site Inspection	I visited the site in 2023.
Responsibilities	I am responsible for the sign-off of the following sections: Section 15 & 16.
Independence	I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.
Prior Involvement:	I have no prior involvement with this Project.

Compliance with NI 43-101	I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.
Disclosure	As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
Dated	16 January 2024.
Signature 	Eugène de Villiers. Director and Consulting Mining Engineer ECMA Consulting (Pty) Ltd.

Summary of Recent Experience:

Year	Client	Commodity	Type	Description
2022/3	Blue Mining Services	Anthracite	Project	Entire Mine design. Surface infrastructure, Mine Access and Underground Layout and LoM Scheduling
2022	Stefanutti Stocks, Mining Services	Coal	Arbitration	Expert witness
2019	Sasol Coal/IXIA JV	Coal	BFS	Detail design of entire mine
2018	UMK	Manganese	Advisory	Independent Consulting Mining Engineer as part of Management Team reporting to CEO – Manganese Project and Mine
2017/18	Mbuyelo Coal	Coal	BFS	Various mine designs, LoM, Medium term and short term planning
2017	Cennergi	Various	FS	Mining Due Diligence on potential acquisition of mine assets in Botswana
2016	South 32/BHP	Coal	Auditing	Auditing of various mining models and various mines
2011-2017	Multiple	Various	BFS/Advisory	Provide a wide range of mine consulting services to the mining industry