

MOLO FEASIBILITY STUDY

National Instrument 43-101 Technical Report

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



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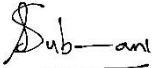
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Abbreviations, Symbols, Acronyms and Units of Measure

ADT	Articulated Dump Truck
Ammonium Nitrate / Fuel Oil	ANFO
Agence de Promotion du Secteur Minier	APSM
Ariary Madagascan Currency	MGA
Betsimisaraka Suture	BS
Bureau de Recherches Géologiques et Minières (France)	BRGM
Bureau du Cadastre Minier de Madagascar	BCMM
Canadian Institute of Mining	CIM
Caracle Creek International Consulting	CCIC
Carbon	C
Centimetres	cm
Closed side setting (crusher)	CSS
Cubic metres	m ³
Cubic metres per hour	m ³ /h
Cubic metres per minute	m ³ /min
Degrees	°
Degrees Celsius	°C
Diamond Drill Hole	DDH
Directly On Line	DOL
Dry metric tonnes	dmt
Dry metric tonnes per hour	dmt/h
Earth Moving Vehicle	EMV
Environmental Commitment Permit	RIM
Feasibility Study	FS
Feet	' or ft
Front End Loader	FEL
Free air delivery	FAD
Global Positioning System	GPS
Greater than	>
Greater than or equal to	≥
Grams	g
Grams per tonne	g/t
Hectares	ha

Abbreviations, Symbols, Acronyms and Units of Measure

High density polyethylene	HDPE
Hours	h
Inductively coupled plasma (assay)	ICP
Intermediate bulk container	IBC
Internal diameter	ID
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
Madagascar Minerals & Resources SARL	MMR
Madagascar Ministry of Environment's Office National pour l'Environnement (the National Office for the Environment)	ONE
Mine Rehabilitation and Closure Programme	MRCP
Mesh size (of screen or sieve)	mesh
Metres	m
Metres above mean sea level	mamsl
Metric tonnes	t
Micrometres	µm
Millilitres	ml
Millimetres	mm
Million tonnes	Mt
Million tonnes per annum	Mtpa
Million Watts	MW
National Environmental Action Plan	NEAP
National Instrument 43-101	NI-43-101
Net Smelter Royalty	NSR
Notified Maximum Demand	NMD
Non-sulphide gangue	NSG
Original equipment manufacturer	OEM

Abbreviations, Symbols, Acronyms and Units of Measure

Particle size distribution where 80% of particles in a stream are larger than the size indicated	P80
Parts per million	PPM
Pollution Control Dam	PCD
PEG Mining Consultants Inc.	PEG
Percent	%
Percent graphitic carbon	% C(g)
Percent total carbon	% C(t)
Preliminary Economic Assessment	PEA
Projet de Gouvernances des Ressources Minérales	PGRM program
Projet de Reforme du Secteur Minier	PRSM program
Qualified Persons	QPs
Quality Assurance / Quality Control	QA/QC
Rock Quality Description	RQD
Run of Mine	ROM
Special Advisory Committee	SAC
Square metres	m ²
Stirred media detritor	SMD
Storm Water Management Plan	SWMP
Taiga Consultants Ltd.	TAIGA
Tailings storage facility	TSF
Thousand Pascals	kPa
Thousand tonnes per annum	ktpa
Thousand tonnes per month	ktpm
Thousand Watts	kW
Three dimensional	3D
Tonnes (1000kg)	t
Tonnes per cubic metre	t/m ³
Toronto Stock Exchange	TSX
Universal Transverse Mercator	UTM
Uranium Star Minerals SARL	USM
Variable Speed Drive	VSD
Whole Rock Analysis	WRA
X-Ray Fluorescence	XRF

1 SUMMARY

1.1 INTRODUCTION

The Company is a mineral exploration and development company based in Toronto, Canada. The Company is currently focused on the exploration and development of its 100% owned, flagship Molo Project.

The Molo deposit is situated 160 km southeast 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.

Geologically, Molo 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 (then Energizer) in late 2011 and announced in early January 2012. The 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 early December of the same year. This maiden mineral resource estimate formed the basis for a PEA, which was undertaken by DRA Projects in 2013.

The positive outcome of this PEA led the Company to undertake another phase of exploratory drilling and sampling in 2014, which was done under the supervision of CCIC. This phase of exploration was aimed at improving the geological confidence of the deposit and its contained mineral resources and included an additional 32 diamond drill holes (totalling 2,063 metres) and 9 trenches (totalling 1,876 metres).

CCIC were subsequently engaged to update the geological model and resource estimate. The entire database on which this new model and resource estimate is based contains 80 drill holes (totalling 11,660 metres) and 35 trenches (totalling 8,492 metres). This new resource formed the basis of the Molo 2015 FS which targeted 860ktpa of ore processing capacity.

This Report utilises the knowledge base of both the Molo 2015 FS and Molo 2017 240ktpa FS technical reports.

Anticipating the future demand for industrial minerals such as those held by the Company (Graphite, Vanadium and Cobalt) is complex. The demand for these minerals is, to a large extent, driven by the development of the battery market which remains uncertain. Significant research has been completed by various analysts and the consensus view is that an explosive increase in demand can be expected. The uncertainty, however, is the timing of such increase in demand.

In order to ensure that the Company remains ahead of the competition and to appropriately plan for future market demand, the Company has opted for a flexible development approach which comprises a modular solution yielding optimal cashflow and return metrics with suitable flexibility to enable them to rapidly respond to market changes.

The Company has an off-take agreement in place with a Japanese Trader and is in the process of formalizing an additional sales agreement with a European Trader. As such, the Company requested feasibility-level analysis of a phased development approach: Phase 1 – 240,000 Tonnes per annum (240ktpa), and Phase 2 – after 240ktpa for the first 2 years of production, ramping up to 720ktpa in the third year to accommodate additional sales.

The Feasibility Study (hereinafter referred to as the “Molo 2019 720ktpa (Ph 2) FS”) as detailed in this Report considers in some detail the development of a greenfields graphite mine with modular beneficiation plant and supporting infrastructure. Engineering, costing and Project planning for Phase 1 of the proposed mine development has been completed to a level as required for this Report to comply to the requirements as defined by the TSX in terms of the NI 43-101. The numbers confirmed during the Molo 2017 240ktpa FS for Phase 1 of the Project are updated as part of this Report to incorporate the effects of escalation and market realities, as currently relevant.

Although detailed engineering has not been completed in order to define an optimal solution for a larger throughput mine facility, costing for Phase 2 (which adds an additional two modules of the beneficiation plant and increases the mining and infrastructure in proportion), is costed as part of this Report. Phase 2 costing is merely factored and therefore deemed accurate to FS level but will be optimised through certain economies of scale which are not considered in this Report. The increased capital expenditure is included in the financial model to provide comprehensive financial analysis of the Project. The increased throughput requires increased water sourcing which has been considered in the engineered solution for water supply. Some of the infrastructure has been increased to support the increased workforce.

The Company has every intent to develop Phase 2 in close succession to 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.

1.2 PROJECT LOCATION

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

1.3 PROJECT DESCRIPTION

The proposed development of the Project includes the construction of a green fields open pit mine, a Phase1 processing plant with a capacity of 240,000 tonnes of ore per annum and all supporting infrastructure including water, fuel, power, tailings (co-disposed), buildings and permanent accommodation. This Project will be augmented with expansion to Phase 2 processing plant with a capacity of 720,000 tonnes of ore per annum based on market conditions.

1.4 SUMMARY OF FINANCIAL RESULTS

Table 1 below summarizes the financial results of Phase 1 (240ktpa for the first two years of production and Phase 2 (ramp-up to 720ktpa production in year three). These are based on a discounted flow analysis of the Project using real cash flows, which do not include the effect of inflation.

TABLE 1: SUMMARY OF FINANCIAL RESULTS

Description	Phase 1 and 2	
	Pre-Tax	Post-Tax
Post-tax: NPV (8% Discount Cash Flow)(1)(2)	\$237.1m	\$184.3m
Post-tax: IRR (1)(2)	43.1%	36.2%
Payback (2)	3.4 years	3.8 years
Capital cost ("CAPEX")	\$60,082,340	
Owners Contingency	\$6,670,430	
On-site Operating Costs ("OPEX") per tonne of concentrate, (year 3 onward)	Mining \$82.69	
On-site Operating Costs ("OPEX") per tonne of concentrate, (year 3 onward)	Processing \$270.27	
Transportation per tonne of concentrate (from mine site to Madagascar Port year 3 onward)	\$133.01	
Average annual production of concentrate	45,136 tonne	
Life of Mine ("LOM")	30 years	
Graphite concentrate sale price (US\$/tonne at Start Up - 2017)	\$1,208	
Average Head Grade	7.1%	
Average ore mined per annum over Life of Mine	720,000 tonne	
Average stripping ratio	0.53:1	
Average carbon recovery	88.30%	

Notes

- Note 1: Assumes Project is financed with 100% equity
- Note 2: Values shown are based on real graphite sales pricing Table 2 below summarizes key mine and process data.

TABLE 2: MINE & PROCESS DATA

	Phase 1	Phase 2
Proven reserves (t)	14,048,733	
Probable reserves (t)	8,207,458	
Grade (% graphitic carbon)	8.05%	7.10%
Waste to ore ratio	0.53:1	
Processing rate (tpa)	240,000	720,000
Mine life (years)	30	
Recovery (%)	88.30%	
Average annual product tonnes	17,000	45,000

1.5 PROPERTY DESCRIPTION AND OWNERSHIP

1.5.1 Property Description

The Project includes 790 claims and an area totalling 308.6 km².

The Project is centred on UTM coordinates 495,289 Easting 7,345,473 Northing (UTM 38S, WGS 84 datum), and is located 11.5 km east-northeast of the town of Fotadrevo.

The property is within Exploitation/Mining Permit PE #39807 which covers an area of 175 km² or 17,500 hectares ("ha"), and Exploration Permits PR #39806 and PR #39810 which cover areas of 96.1 km² (9609 ha) and 37.5 km² (3750 ha), respectively.

1.5.2 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 MOU with Malagasy to acquire the remaining 25% interest in the land position.

On April 16, 2014, Energizer 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’ one and two. 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 five years, with the possibility of two renewals of an additional three 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 titleholder on an Exploration Permit are minimal. A titleholder 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 predetermined 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 have to 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.

1.5.3 The Company's Royalties

Malagasy retains a 1.5% net smelter return royalty on the Project.

1.5.4 Permits

Exploitation Permit PE #39807 (175 km²) 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 the 240ktpa (Phase 1) Project on April 8, 2019 after reviewing the following:

- Exploitation Permit PE #39807
- Environmental & Social Impact Assessment ("ESIA") and Relocation Action Plan ("RAP") to International Finance Corporation (IFC) Performance and World Bank Standards
- Completion of local and regional stakeholder and community engagement, with overwhelming support from both the local community and local government, as well as regional government
- Signed agreements with all potentially affected land occupants to accept compensation for any affected crops and grazing land and relocation if needed
- Approved capital investment certification from the BCMM
- Receipt of Cahier des Charges Miniér (mining specification) from the BCMM as pre-requisite to submitting the ESIA & RAP to ONE for review
- Successful completion of the ONE's technical evaluation process which consisted of a site visit and four separate community consultations
- Joint agreement and signature of the Cahier des Charges Environnementales (environmental specification) with the ONE

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) 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.

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 ($\pm$ biotite, garnet) quartzo-feldspathic gneiss, marble, chert, quartzite, and amphibolite gneiss. The metaplutonic

rocks include migmatitic ($\pm$ hornblende / diopside, biotite, garnet) feldspathic gneiss of monzodioritic to syenitic composition, biotite granodiorite, and leucogranite.

1.7 MINERAL RESOURCE ESTIMATE

The Project hosts the following resources:

- Measured mineral resource of 23.62 Mt grading 6.32% Carbon ("C")
- Indicated mineral resource of 76.75 Mt grading 6.25% C
- Inferred mineral resource of 40.91 Mt at 5.78% C
- The effective date of the Mineral Resource tabulation is 14 August 2014. The Mineral Resources are classified according to the Canadian Institute of Mining, Metallurgy and Petroleum definitions. A cut-off grade of 4% C was used for the "higher grade" zones and 2% C for the "lower grade" zones. It is important to note that while the 'high' grade resource occurs within the 'low' grade resource, each was estimated and reported separately.
- A relative density of 2.36 tonnes per cubic meter was assigned to the mineralized zones for the resource estimation. The resource remains open along strike and to depth. The Mineral Resources above are inclusive of the Mineral Reserves below.
- The current mineral resource estimate for Molo is summarised in
- Table 3 below. The mineral resources are classified in the Measured, Indicated and Inferred categories as defined by the Canadian Institute of Mining, Metallurgy and Petroleum definition standards.

TABLE 3: MINERAL RESOURCE STATEMENT FOR THE MOLO GRAPHITE DEPOSIT - SEPTEMBER 2014

Classification	Material Type	Tonnes	Grade - C%	Graphite - T
Measured	"Low Grade"	13 048 373	4.64	605 082
Measured	"High Grade"	10 573 137	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

Classification	Material Type	Tonnes	Grade - C%	Graphite - T
Inferred	"Low Grade"	24 233 267	4.46	1 080 677
Inferred	"High Grade"	16 681 453	7.70	1 285 039
Total Inferred		40 914 721	5.78	2 365 716

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

Notes:

- Mineral Resources are classified according to the Canadian Institute of Mining definitions.
- Mineral Resources are reported Inclusive of Mineral Reserves.
- "Low Grade" Resources are stated at a cut-off grade of 2% C.
- "High grade" Resources are stated at a cut-off grade of 4% C.
- Eastern and Western high-grade assays are capped at 15% C.
- A relative density of 2.36 tonnes per cubic metre (t/m³) was assigned to the mineralised zones for the resource tonnage estimation.

The total Measured and Indicated Resource is estimated at 100.37 million tonnes, grading at 6.27% carbon. Additionally, an Inferred Resource of 40.91 million tonnes, grading at 5.78% carbon 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 million tonnes, grading at 6.32% carbon, have been upgraded by infill drilling from the Indicated to Measured Resource category.

1.8 EXPLORATION

No further exploration is currently planned.

1.9 MINERAL RESERVE ESTIMATE

The mineral reserves declared in this Molo 2019 (720ktpa) FS are declared as per Table 4 below.

TABLE 4: MINERAL RESERVES

Category	Tonnage	C Grade (%)
Proven	14 169 741	7.00
Probable	8 266 944	7.04
Proven and Probable	22 436 685	7.02

Proven reserves are reported as the Measured Resources inside the designed open pit and above the grade cut-off of 4.5% C. Similarly, the Probable Reserves are reported as the Indicated Resources inside the designed open pit and above the grade cut-off of 4.5% C.

1.10 METALLURGICAL TEST WORK

The FS analyses are based on a full suite of metallurgical test work performed by SGS Canada Metallurgical Services Inc. in Lakefield, Ontario, Canada. These tests included laboratory scale metallurgical work and a 200-tonne 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 of concentrate grades between 88.8% total carbon and 97.8% total carbon. Optical mineralogy on representative concentrate samples identified interlayered 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 87.8% based on the metallurgical response of composites using samples from all drill holes within the five-year pit design of the original FS at the higher concentrate production rate of 53,000 tpa. The average composition of the combined concentrate grade is presented in Table 5. The size fraction

analysis results were converted into a grouping reflecting a typical pricing matrix, which is shown in Table 6.

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 uncertainty of up to 1.7% relative for graphite concentrate grades greater than 90% total carbon.

TABLE 5: 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 6: PRICING MATRIX - FLAKE SIZE DISTRIBUTION GROUPING AND PRODUCT GRADE

Product Size	% Distribution	Product Grade (%) Carbon
>50 mesh	23.6	96.9
-50 to +80 mesh	22.7	97.1
-80 to +100 mesh	6.9	97.2
-100 mesh	46.8	97.6

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

1.11 RECOVERY METHODS

The process design is based on an annual Phase1 feed plant throughput capacity of 240 kilotonnes at a nominal head grade of 8.05% C(t) producing an estimated average of 17 kilotonnes per annum (kt/a) of final concentrate. The same process design has been applied to an annual Phase 2 feed plant throughput capacity of 720 kilotonnes at a nominal head grade of 8.05% C, which would produce an estimated average of 45 kt/a of final graphite concentrate.

The ore processing circuit consists of three stages of crushing which comprises jaw crushing in the primary circuit, followed by secondary cone crushing and tertiary cone crushing; the secondary and tertiary crushers operate in closed circuit with a double deck classification

screen. 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 $\pm 55\%$ utilization. The crushed product (P80 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. A single stage primary ball milling circuit is employed, incorporating a closed-circuit classifying screen and a scalping screen ahead of the mill. The scalping screen undersize feeds into a flash flotation cell before combining with the mill discharge material. Scalping and classification screen oversize are fed to the primary mill.

Primary milling is followed by rougher flotation which, along with flash flotation, recovers graphite to concentrate from the main stream. 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/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 final tailings thickener. 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 MIBC (methyl isobutyl carbinol) frother are used 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 (methyl isobutyl carbinol) 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 (Magnaflow 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 (Magnaflow 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.

1.12 INFRASTRUCTURE

The project is located in a relatively remote part of South Western Madagascar, approximately 13 km NE of the local village of Fotadrevo. There is currently limited infrastructure on site and project infrastructure will have to be constructed.

The following elements are all part of the project scope:

- Raw water supply (from a network of bore holes extracting ground water)
- Power supply (temporary during construction) and then a permanent diesel power station to supply the plant and permanent camp
- Sanitation for the plant, permanent camp, and temporary during construction
- Storm water control and management
- All permanent buildings (offices, workshops, stores, laboratory)
- All buried services (potable water, sewage, stormwater, electrical reticulation)
- In plant roads
- Haul road

- Waste, high and low grade -Rock dumps.

As the proposed Phase1 plant is a small plant, and with the Company's intention to rapidly expand to a larger Phase 2 process plant and mining operation, the brief from the Company was to develop a "fit for purpose" and cost-effective design without compromising on safety or quality.

1.13 GEOTECHNICAL

The geotechnical investigation conducted by SRK Consulting in 2014 was used as reference document for the design and planning of this phase of the project. (Report 479297/Plant Geotech/Final).

In summary, transported soils are present across all areas investigated to shallow depths not exceeding a maximum depth of 0.6 m. From the consistencies noted during test pit excavations the transported soils are anticipated to have a maximum allowable bearing capacity of 100 kPa, limiting total consolidation settlement to 25 mm.

Residual soils were noted in the majority of the test pits excavated and comprised dense to very dense silty and/ or clayey sands. The residual soils are expected to have a maximum allowable bearing capacity of 200 kPa, limiting total consolidation settlement to 25 mm (differential settlement expected to be half this value).

As rock is located at a shallow depth at most locations it is recommended that structures generally be founded on rock rather than the overlying thin soils. However, light structures with loads of less than 100 kPa could be founded on the soils if necessary.

1.14 CONCRETE

Concrete grades and mix design were selected taking into consideration durability requirements. Particular attention will be given to wet process plant areas and wash down slabs. All foundations were designed as pad or raft type foundations with load bearing pressures not exceeding 150kPa. Foundations were designed to minimize settlement.

1.15 STORM WATER

Storm water runoff within the process plant areas are dealt with by a minimum slope on the terrace platform. Runoff is then collected in concrete lined V-drains.

Storm water within the process plant area will be collected through dedicated storm water containment channels and then handled accordingly.

1.16 PRODUCT PRICING

As an industrial mineral, flake graphite pricing is determined by three factors: 1) flake size, 2) carbon purity and 3) industry-specific technical attributes of the flakes (Benchmark, 2017a; Roskill, 2017). Flake sizing is broadly classified into four ranges: small (-100 mesh, or <75µm) medium (-80 to 100 mesh, or 75µm to 180µm), large (-50 to 80 mesh, or 180µm to 300µm),

and extra-large or jumbo (+50 mesh, or >300µm). These flake sizes are in turn classified by carbon content ("C"), and are typically sold in ranges of 88-93% 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. There are however subscription pricing services that provide monthly graphite pricing for various flake sizes and carbon purities based upon input from graphite purchasers. The Company utilized the average pricing for the past 12 months for flake graphite sold on an FOB China basis, provided by UK-based Benchmark Mineral Intelligence, with the flake size distribution of Molo graphite to arrive at a "basket" sale price of US \$1207.55 per tonne as outlined in Section 19.3.

1.17 LOGISTICS

The Port of Ehoala at Fort Dauphin is a modern (2009) port developed by Rio Tinto for the QMM project. It has a 15m draft with shipping lines calling on a regular basis. There are however no crane facilities and vessels require their own cranes.

The following equipment are available at the port.

- 1 x 3.5T Telehandler
- 5 x Trailers for container movement (2x40ft, 3x20ft)
- 1 x Tractor
- 2 x Reach stacker
- 6 x Forklifts (1 x 2.5T; 2 x 5T; 3 x 7T)

The port is fenced and there is a security service (G4S) for days and nights port guarding. Despite the presence of a national airport, the port of Ehaola is mainly connected to the hinterland destinations by road. All types of trucks can obtain access to the port and his berth for cargo off loading, however the majority are container trucks (20ft and 40ft).

Customs are available on site and clearance can be streamlined via pre-clearance in order to lessen standing time of the containers once arrived. It is to be noted that all cargo items imported into the Republic of Madagascar, needs to have a BSC online cargo tracking note. Failing to submit the BSC certificate, cargo cannot be cleared, and the shipment will be sent back to origin and be subject to a fine of 2500 USD per Bill of lading, plus regulations charges. All containers, vehicles, bulk commodities, including airfreight requires a BSC certificate.

The route from Molo to Fort Dauphin runs either via the RN 10 or the RN 13. Both these routes are in relatively poor condition and trucks are expected to take between four and five days to make the round trip. A truck was run over the route by a Madagascan trucking contractor to gauge cycle times and they managed to complete the journey in two long days each way. This was in the dry season and in the wet season there may be periods of time when the roads become impassable. No money has been budgeted for roads repairs or upgrades.

Due to the poor road conditions, majority of cargo would have to be transported to site during the dry season. Cargo transport limitations include:

- 12m (L) x 3.5m (W) x 2.8m (H) at a maximum of 35 T per 3-axle trailer.
- 12m (L) x 2.5m (W) x 3.5m (H) at a maximum of 26 T per 2-axle trailer.

Cargo exceeding 4m width pose problems to transport due to the Manambaro Bridge, as there is no possibility to divert. Some access areas would also need to be adjusted for items holding a width of 2.3m – 3.6m. (Ex. Raft of Bevilana). Any cargo exceeding the above-mentioned limitations would have to be considered on a case-by-case basis prior to importation.

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

1.18 CAPEX AND OPEX

The Phase 1 initial CAPEX is estimated at \$21.0 M, including a 10% contingency, with an additional \$3.1 M required to cover the first 3 months of working capital. Phase 2 CAPEX is estimated at \$67.4 M, including a 12.5% contingency, with an additional \$7.3 m required to cover the first 3 months of working capital. Over the life of the mine, sustaining capital of \$3.3 M will be required for equipment replacement and for rehabilitation at the end of the project. Table 7 summarizes the capital requirements.

The base date for the capital costs is May 2019 and no provision has been made for escalation. The accuracy of capital costs is considered to be with +/- 10%.

TABLE 7: CAPITAL COSTS

Capital Cost Breakdown	Phase 1 (240ktpa)	Phase 2 (720ktpa)
Process Equipment	\$8,438,609.00	\$25,315,827.00
Civil & Infrastructure	\$2,103,672.21	\$6,661,016.63
Tailings	\$0.00	\$0.00
Mining	\$2,574,143.85	\$4,913,341.38
Buildings	\$1,154,609.43	\$2,886,523.59
Electrical Infrastructure	\$128,804.10	\$386,412.30
Project Services/EPCM	\$931,481.79	\$2,794,445.38
Construction Services	\$1,474,775.11	\$3,686,937.78
Indirect Costs	\$372,750.00	\$1,118,250.00
Environmental & Permitting costs	\$729,827.94	\$1,459,655.89
Owner's Costs	\$1,197,000.00	\$4,189,500.00
Sub-total	\$19,105,673.44	\$53,411,909.93
Contingency (10%/12.5%)	\$1,910,567.34	\$6,676,488.74
3 Months Working Capital	\$3,100,000	\$7,300,000
CAPEX TOTAL	\$24,116,241	\$67,388,398.67
Sustaining CAPEX over Life of Mine		\$3,300,000

The operating costs per tonne of finished graphite flake concentrate delivered on a FOB basis at the Port of Fort Dauphin, Madagascar are outlined in

Table 8.

TABLE 8: OPERATING COSTS PER TONNE OF FINISHED GRAPHITE CONCENTRATE

Category	Phase 1	Phase 2
	Operating cost	
Mining (US\$/T)	102.81	65.34
Processing (US\$/T)	265.82	265.82
Trucking to local port / Ft. Dauphin (US\$/T)	133.01	133.01
General and Administration (US\$/T)	64.29	50.00
TOTAL	\$565.93	\$514.17

Please note that these operating costs assume that the plant is able to successfully handle the variability in the ore body, as shown by the SGS test work discussed in detail in Section 13. Should the plant not perform as expected this could have a material impact on operating costs as:

- The flake size distribution could be worse than expected
- The product grade could be lower than expected

- The recoveries could be lower than expected or a combination of all of these

1.19 ECONOMIC ANALYSIS

Table 9 below summarizes the economic analysis of the project using discounted cash flow methods.

TABLE 9: ECONOMIC ANALYSIS OF THE PROJECT

Metric	Unit	Value
Before Tax		
Total Project Cash Flows	USDm	841
NPV @ 8%	USDm	237.1
NPV @ 10%	USDm	182.9
NPV @ 12%	USDm	143.3
IRR	%	43.10%
Payback Period	year	3.4
After Tax		
Total Project Cash Flows	USDm	671.6
NPV @ 8%	USDm	184.3
NPV @ 10%	USDm	140.5
NPV @ 12%	USDm	108.4
IRR	%	36.20%
Payback Period	year	3.8

Note

All values in the above table do not account for inflation in costs or product pricing.

The assumptions used in the financial model are as follows:

- 13 South African Rand (ZAR) to US\$1

- 1.1 Euro to US\$1
- 12% Import Duties and Taxes

1.20 ENVIRONMENTAL & PERMITTING

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 the 240ktpa (Phase 1) Project on April 8, 2019 after reviewing the following:

- Exploitation Permit PE #39807
- Environmental & Social Impact Assessment ("ESIA") and Relocation Action Plan ("RAP") to International Finance Corporation (IFC) Performance and World Bank Standards
- Completion of local and regional stakeholder and community engagement, with overwhelming support from both the local community and local government, as well as regional government
- Signed agreements with all potentially affected land occupants to accept compensation for any affected crops and grazing land and relocation if needed
- Approved capital investment certification from the BCMM
- Receipt of Cahier des Charges Miniér (mining specification) from the BCMM as pre-requisite to submitting the ESIA & RAP to ONE for review
- Successful completion of the ONE's technical evaluation process which consisted of a site visit and four separate community consultations
- Joint agreement and signature of the Cahier des Charges Environnementales (environmental specification) with the ONE.

1.20.1 Environmental and Social Impact Assessment

A comprehensive Environmental and Social Impact Assessment was completed and submitted to Malagasy government as part of the Environmental Permit process.

Early integration of environmental and social sensitivities and risks ensured that the final impact assessment component revealed that there are no fatal flaws from an environmental and social perspective. The significance levels of impacts range from minor to major before any mitigation measures are applied and from minor to average with mitigation measures included. Notably, all major risks require significant reduction in risk via stringent controls. These controls have been incorporated into the Project design and planning with additional operational controls specified within the various environmental and social management plans.

To this end, the ESIA contains a chapter which details specific management measures which either remove the risks completely or reduce their significance to an acceptable level.

In addition, each specific environmental and social component has a prescribed monitoring plan which will be followed during each Project developmental phase. This is aimed at monitoring compliance against various specifications such as the baseline environment and predicted impact removal and reduction measures.

1.21 CONCLUSIONS

1.21.1 Geology

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.

1.21.2 Mining

Maiden mineral reserves of 22 300 000 tonnes have been declared for the Molo 2019 720ktpa (Phase 2) FS at an average grade of 7.0% and based on the information contained in the FS, it is possible to economically mine this deposit.

1.21.3 Tailings

Due to the substantially reduced tonnages for the project as envisaged, tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining. Despite this co-disposal approach, a detailed design has been completed, complete with environmental and social impact assessment and closure to allow for the upgrade to a more conventional, cyclone facility, should the throughput be increased during the life of the mine with an expansion to Phase 2 production. This approach has been pursued to ensure that sufficient flexibility is built into the project development strategy to accommodate the anticipated increase in market demand.

1.21.4 Risks

In addition to the qualitative risk assessment completed during the Molo 2015 FS, a comprehensive HAZID study was completed as part of this Molo 2019 720ktpa (Phase 2) FS.

1.21.5 Permitting

The Mining and Environmental Permits have been obtained for the project, but supplementary sectoral permits will be required.

1.21.6 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.

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.

1.22 RECOMMENDATIONS

1.22.1 Geology

No further recommendations.

1.22.2 Mining

The Project will allow for potential optimisation of drilling and blasting designs during execution that could reduce operating costs slightly.

From a pure mining perspective, the Project is robust and provided reasonable levels of short-term planning are applied it should have very few challenges in delivering the required tonnages at the required grade to meet the production targets set out in this study.

1.22.3 Metallurgical Test Work

The following recommendations are made for the detailed engineering stage:

- Investigate the metallurgical impact of different attrition mill technologies such as stirred media mills or attrition scrubbers;
- Evaluate a range of different grinding media (e.g. different size, shape, material) to determine if flake degradation can be reduced without affecting the concentrate grade;
- Develop a grinding energy versus concentrate grade relationship for the best grinding media. This will allow a more accurate prediction of the required attrition mill grinding energy as a function of the final concentrate grade;
- Conduct attrition mill vendor tests to aid in the sizing of the equipment;
- Carry out vendor testing on graphite tailings using the optimized reagent regime proposed by the reagent supplier.

- Complete a series of flotation tests on samples covering mine life intervals for the Molo 2019 720ktpa (Phase 2) FS pit design.

1.22.4 Recovery Methods

The process plant has been designed to easily optimize the final product grade, this is achieved by having two options in the attrition cleaning step. It is however recommended that additional laboratory test work be conducted to test the current plant configuration for treatment for higher feed grade material. Provision is to be made for attritioning circuit tailings to be recycled back into the process.

1.22.5 Infrastructure

The following are recommended prior to the detailed design stage:

- Additional geotechnical investigations at the proposed new construction and permanent camp site, particularly at the location of the new potable water storage tanks
- A detailed geotechnical investigation will need to be undertaken to identify and confirm suitable sources of concrete aggregate and concrete sand materials at the location of the project site. This testing will need to include for concrete material testing and the production of concrete trial mixes with the material identified
- The geotechnical information will also need to confirm the suitability for construction of all the material to be excavated from the Return Water Dam. It is proposed that all the material excavated from the Return Water Dam is utilised in the works as processed fill material
- Confirmation as to whether the material from the proposed borrow pit near Fotadrevo (which will be used to supply all fill material for the TSF starter wall construction) can be utilised as fill material, or if this material can be stabilized in some manner and used in the works
- A detailed topographical survey will need to be undertaken of the proposed construction site, borrow pit areas and the access road between Fotadrevo and the mine site. This information is required prior to the final detailed design of the plant layout and associated earthworks

1.22.6 Water

The following is recommended during the detailed design phase:

- 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. Additionally, proposed 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.

- The installation of a weather station on the Project site should be done as soon as possible.
- Quantitative and predictive water balance, groundwater and geochemical analyses should be undertaken on regular intervals in order to update the water management plan.

1.22.7 Environmental, Social

- The installation of a suitable weather station at or as near as possible to the proposed project site, even before construction commences, is recommended. Accurate, local weather data is almost non-existent in Madagascar. This data will prove invaluable for model calibration, improvement in baseline understanding and for future energy supply options which could utilise wind and or solar power generation.
- Clean and or renewable energy supply should be considered as a medium to long term target.
- Appointment of a community representative and the establishment of a mandate to sensitise the local communities prior to any project activities.
- Monitoring and auditing to commence at project preparation phase.
- Compilation of Standard Operating Procedures for Environmental and Social aspects requiring direct management and intervention.
- It is recommended that actual activity data, (e.g. kilometres travelled, or litres of diesel consumed) for a financial year is used when a GHG Assessment is being calculated. Given that this project involves an estimation of a future GHG assessment for activities yet to begin, a series of assumptions have been made in order to obtain the activity data required to undertake this calculation.
- Community recruitment, skills development and training should begin at project preparation phase.

1.22.8 Permitting

- Security of land tenure is a process and is estimated to take 6-9 months, thus this process should be commissioned as early as possible.
- Application for all other necessary permits (water use, construction, mineral processing, transportation, export, labour and so forth should be undertaken.
- Compilation of a comprehensive legal register.
- Application for an amendment of the environmental approval would be required for the expansion to 720ktpa (Phase 2).

2 INTRODUCTION

2.1 THE ISSUER

This Technical Report (the “Report”) has been prepared for the Company (<http://www.NextSourceMaterials.com/>) (“NEXT”, or “the Company”), a mineral exploration and development 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 the exploration and development of its Molo graphite deposit (“Molo” or “the Project”) in south-western Madagascar.

2.2 TECHNICAL TEAM

This Technical Report has been prepared by a combined technical team made up of people from the following organizations:

- Caracle Creek International Consulting (Proprietary) Limited – Geology
- GCS Water and Environment (Pty) Ltd / Agetipa – Water
- Globesight (Pty) Ltd - Environmental, Social, Permitting
- SGS Lakefield – Metallurgical Test work
- Metpro – Metallurgical Test work analysis
- EPOCH – Tailings Storage Facility
- Erudite Projects (Pty) Ltd – Mining design & mining capital and operating costs
- EVH Consulting – Earthworks, Civils and Infrastructure
- Erudite Projects (Pty) Ltd - Process Engineering
The process engineering was completed, as part of the Molo 2017 204ktpa FS, by Met63 with the competent person being Mr. Paul Harvey supported by Mr A Mokwena. Neither of these experts are in the employ of Met63 at this stage and were not able to further comment on the process as part of the 2019 update. Mr Pono Mogoera, Pr.Eng who holds specific graphite processing knowledge and is currently in the employ of Erudite Projects (Pty) Ltd, has reviewed the process design in consultation with Mr Mokoena and SGS Lakefield and his comments are incorporated as appropriate. As a result, P. Mogoera is the signatory for the process design in this update.
- SDV, Panalpina, Unitrans, Strang – Logistics
- OHMS – Mine Geotech
- RLH – Port Trade Off Study (From Molo 2015 FS)

2.3 REPORT HISTORY

The report history for this Project is as follows:

Date	Description of Event
May 2019	Technical Report NI 43-101 report of 2017 updated to reflect escalation and FS for increase from 240ktpa to 720ktpa throughput
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 240ktpa Feasibility Study (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	PEA Technical Report NI 43-101 issued
2013	PEA undertaken by DRA
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

2.4 TERMS OF REFERENCE AND PURPOSE

The terms of reference for this phase were to update the Molo 2017 240ktpa FS on the Molo graphite deposit to reflect current pricing. The work covered the following key activities:

- To compile Capex and Operating Cost estimates for the project
- To build a financial model and perform an economic analysis of the project

2.5 SOURCES OF INFORMATION

The following sources of information have been used in this Report. See Table 10 below.

TABLE 10: SOURCES OF INFORMATION

Report of Document	Author / Organization
Physical Data	
Site Topographical Data	The Company
Exploration Permit 3432	Malagasy Government
Civil Geotechnical Reports	EPOCH Consulting
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	Erudite Projects
Equipment Tenders	Various
Process Plant and Metallurgy	
Process Design Criteria (PDC)	Modular Supplier
Process Flow Diagrams (PFD)	Modular Supplier
Mechanical Equipment List (MEL)	Modular Supplier
Piping and Instrumentation Diagrams (P&ID)	Modular Supplier
Metallurgical Test Work Reports	SGS
Metallurgical Test Work Analysis	Metpro
Vendor Test Work Reports	Modular Supplier
Hazop Studies (Hazop 1 and Hazop 2 Reports)	The Company
Plant Layout Drawings	The Company
Block Plan	The Company
Rheology Test Work	SGS
Production Schedule	DRA

Report of Document	Author / Organization
Tailings Disposal and Storage	
Site Selection Report	Erudite Projects (Pty) Ltd
TSF Report (including TSF Geotechnical Study)	Erudite Projects (Pty) Ltd
Infrastructure	
Site Plan	The Company
Area Plan	The Company
Architectural Drawings	The Company
Infrastructure Drawings	The Company
Infrastructure Equipment List (IEL)	Modular Supplier
Plant and Camp Geotechnical Report	Erudite Projects (Pty) Ltd
Water Supply Dam Geotechnical Report	InRoads Consulting
Surface Water Supply Report - ANT_2 Dam	GCS
Power	
Power Plant Layout Drawings	Modular Supplier/MET 63
Running Load Estimate	Modular Supplier/MET 63
Single Line Diagram	Modular Supplier/MET 63
Vendor Quotes	Various
Fuel quotes	Various
Water	
Detailed Water Report	GCS Water and Environment
Water Balance	GCS Water and Environment
Dam Site Trade-off Study	GCS Water and Environment
Environmental and Social	
Specialist Studies	GCS Water and Environment & Agetipa
ESIA (Environmental & Social Impact Assessment)	GCS Water and Environment & Agetipa
MOU (Memorandum of Understanding)	Agetipa
TOR (Terms of Reference)	Agetipa
Certification of Investment Amount	GCS Water and Environment & Agetipa
Sensitivity Maps	GCS Water and Environment
Permitting and Stakeholders	
Permit Register	Agetipa
Stakeholder Register	Agetipa
HR and Operational Readiness	
Malagasy Legislation	The Company

Report of Document	Author / Organization
Malagasy Labour Rates	The Company
Staffing Plan	DRA
Transport and Logistics	
Route Surveys	Unitrans SDV

2.6 PERSONAL INSPECTIONS

The following personal inspections were done on the property:

- Dr. Philip John Hancox between the 8th and 11th of 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 19th and 21st of 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 15th and 19th of 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 11-13 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.
- P. Mogoera has not visited the site.
- Although not an author of this report, a principal engineering geologist, Mr Colin Wessels, visited the site from 5th to 13th April and 17th to 21st 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 staff from GCS although not QP's on the project also spent time on site and contributed to the study in their various areas of expertise.
 - Ferdi Pieterse : 11- 13 March 2014
 - Alkie Marais : 11 - 13 March 2014
 - Alistair Main : 11 - 14 March 2014
 - Alvar Koning : 11 - 15 March 2014 and 1 July – 12 July 2014
 - Hassen Khan : 11 - 15 March 2014 and 1 July – 12 July 2014

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 31st of May 2019);
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information as supplied by the Company and other third-party sources.

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.

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.

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. To the extent possible under the mandate of a NI 43-101 compliant report, the data have been verified with regard to the material facts.

The Company has received specialist input from the following organizations in the preparation of this report and place full reliance on this information:

- EPOCH (Tailings Storage Facility)
- GCS Water and Environmental (Pty) Ltd & Agetipa (ESIA, Water, Geohydrology, Geochemistry, Water Balance, Environmental, Social, Permitting)
- SDV, Unitrans (Logistics)
- OHMS (Mine Geotechnical)
- Africa Business Solutions (Taxation)
- Metpro Management (Process analytics)
- Caracle Creek International (Resource definition)
- Erudite Projects (Pty) Ltd (Mining and Process)

This reliance applies to the following sections of the technical report as per competent persons sign off.

Except for the purposes legislated under provincial securities laws any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 MADAGASCAR OVERVIEW

The Republic of Madagascar (“Madagascar”) is not a traditional or particularly well-known exploration and mining destination and is mostly under-explored, with the country subject to only very little modern era systematic exploration.

4.2 COUNTRY OVERVIEW

Madagascar is the largest island in the Indian Ocean (Figure 1) and the 4th largest island in the world (after Greenland, New Guinea and Borneo). It is located to the southeast of the African continent, from which it 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 km² and a 5,000 km long coastline.



FIGURE 1: MAP OF MADAGASCAR

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

The capital of Madagascar is Antananarivo, a city of approximately 1,500,000 people. It is located in the central eastern area of the island approximately 150 kilometres inland from the central-east coast, at an elevation of just over 1,200 metres above mean sea level (“mamsl”).

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

Madagascar is one of the world’s hotspots of endemism and is recognised 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.

4.2.1 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 Reforme 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 five-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.

4.3 LOCATION

The Project is located approximately 160 km southeast of Madagascar's port city of Toliara, in the Tulear region and roughly 220 km northwest of Fort Dauphin as per Figure 2.

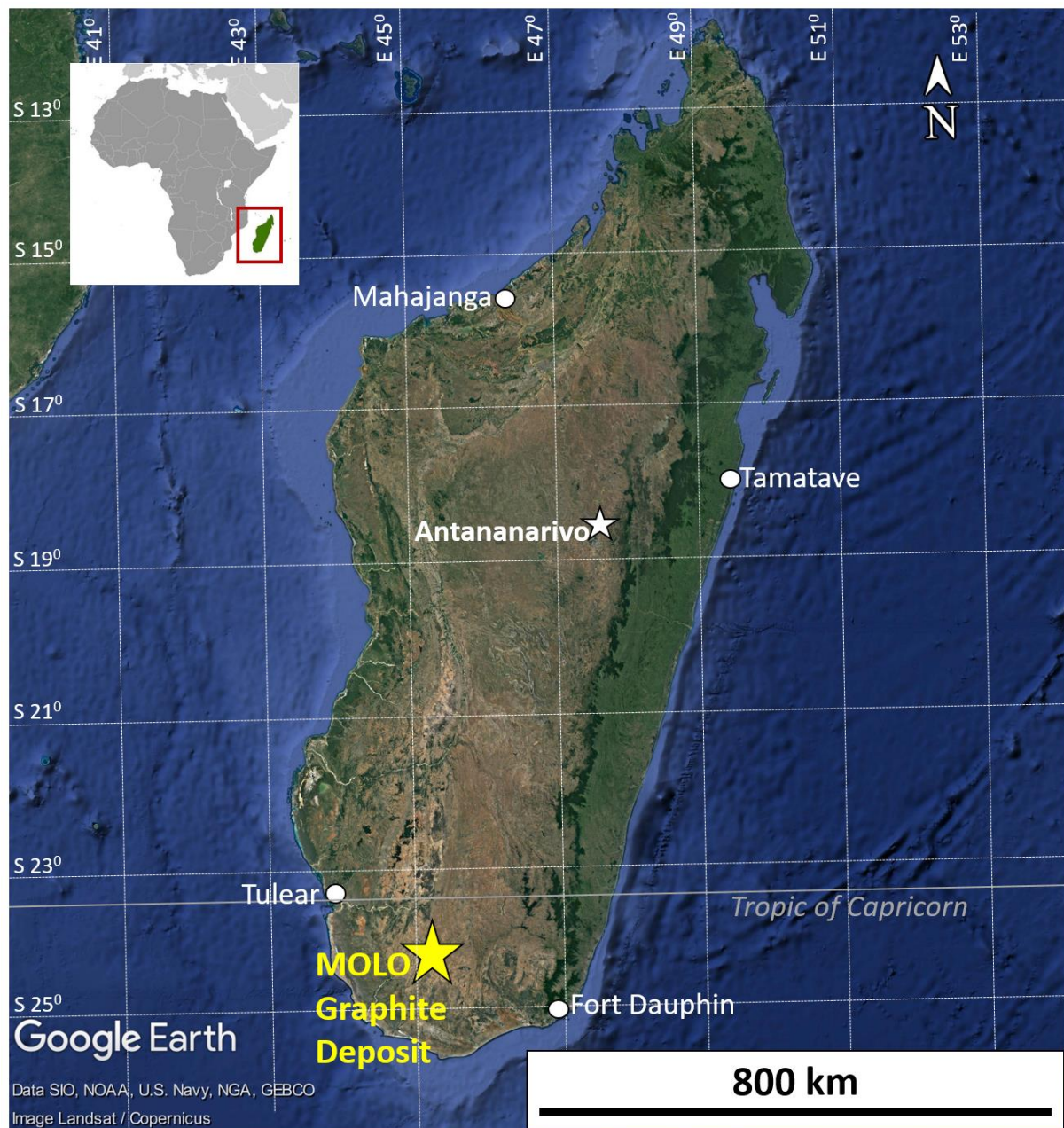


FIGURE 2: MOLO PROJECT LOCATION

4.4 PROPERTY AREA

The Project includes 790 claims and an area totalling 308.6 km². See Figure 3 below. The Project is centred on UTM coordinates 495,289 Easting 7,345,473 Northing (UTM 38S, WGS 84 datum). The Project is located 11.5 km east-northeast of the town of Fotadrevo within Exploitation/Mining Permit PE #39807 which covers an area of 175 km² or 17,500 hectares ("ha"), and Exploration Permits PR #39806 and PR #39810 which cover areas of 96.1 km² (9609 ha) and 37.5 km² (3750 ha) respectively. The Government of Madagascar designates individual claims by a central LaBorde UTM location point, comprising a square measuring 625 X 625 m.



FIGURE 3: MOLO PROJECT - EXPLOITATION PERMIT PE #39807

4.5 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 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 one and two. 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 five years, with the possibility of two renewals of an additional three years each. Payments of the administration fees are due each year on 31 of March of each year, 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 titleholder on an Exploration Permit are minimal. A titleholder 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 predetermined 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 have to 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.

4.6 ROYALTIES

Malagasy retains a 1.5% net smelter return royalty on the Project.

4.7 PERMITS

Exploitation Permit PE #39807 (175 km²) 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 the 240ktpa (Phase 1) Project on April 8, 2019 after reviewing the following:

- Exploitation Permit PE #39807
- Environmental & Social Impact Assessment (“ESIA”) and Relocation Action Plan (“RAP”) to International Finance Corporation (IFC) Performance and World Bank Standards
- Completion of local and regional stakeholder and community engagement, with overwhelming support from both the local community and local government, as well as regional government
- Signed agreements with all potentially affected land occupants to accept compensation for any affected crops and grazing land and relocation if needed
- Approved capital investment certification from the BCMM
- Receipt of Cahier des Charges Miniér (mining specification) from the BCMM as pre-requisite to submitting the ESIA & RAP to ONE for review

- Successful completion of the ONE's technical evaluation process which consisted of a site visit and four separate community consultations
- Joint agreement and signature of the Cahier des Charges Environnementales (environmental specification) with the ONE

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 4). From Andranovory, 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.

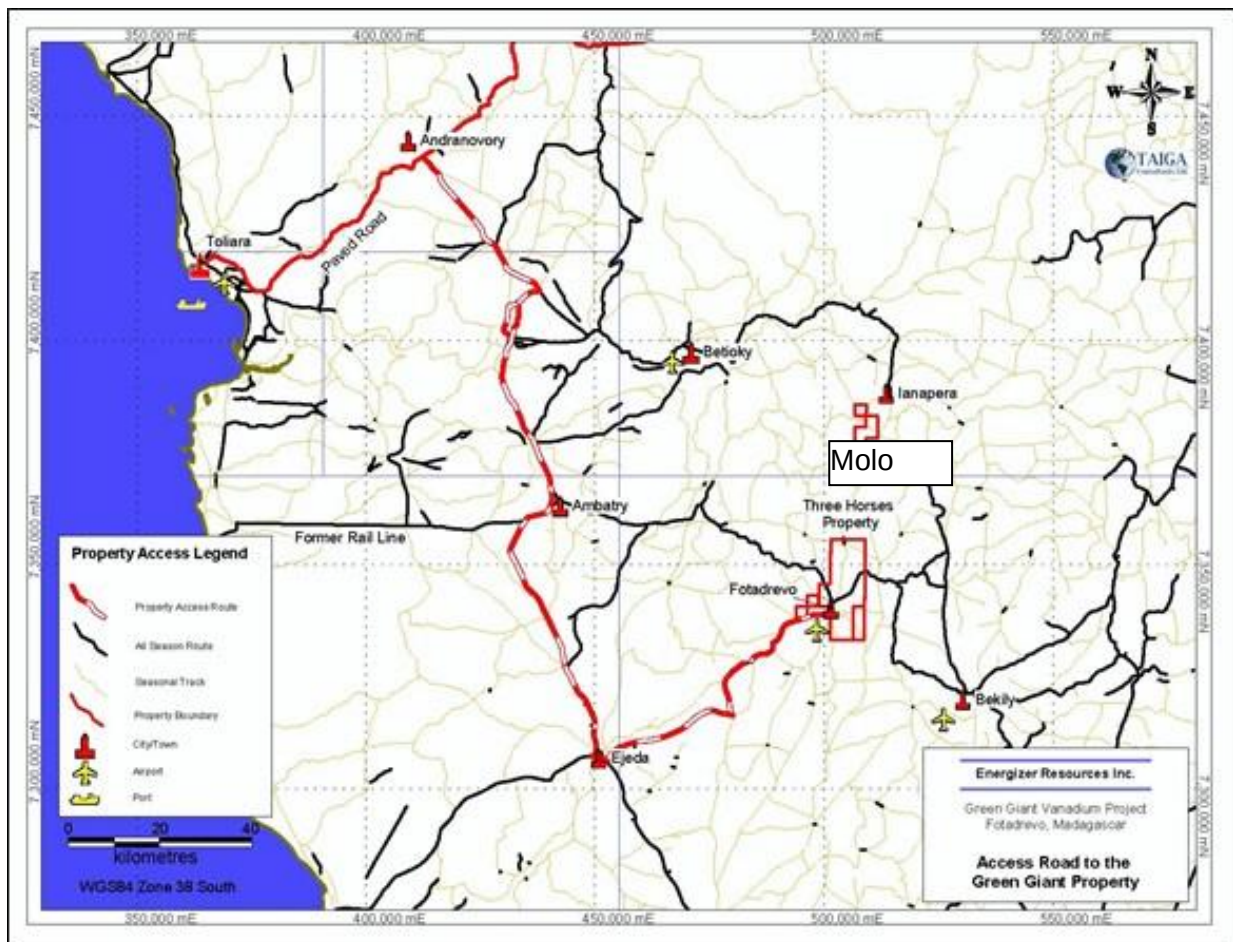


FIGURE 4: ROAD ACCESS TO THE MOLO AREA FROM THE TOWN OF TOLIARA

With the upgrading of an existing airstrip at Fotadrevo (Figure 5) 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 min from Toliara.



FIGURE 5: AIRSTRIP AT FOTADREVO

Antananarivo is currently serviced by Air France (from Paris), South African Airways (from Johannesburg) and Air Mauritius (Sir Seewoosagur Ramgoolam International Airport in southeast Mauritius). Air Madagascar also provides services to Paris, Johannesburg, Mauritius, Nairobi, and Réunion from Ivato International Airport, which is located 16 km from Antananarivo, the capital of Madagascar. 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 (Figure 6) Grass cover is widespread and trees are widely spaced overall, with accumulations near drainage lines and streambeds. In areas of lower relief, alluvial cover is generally shallow, and bedrock and/or float are readily observable. Elevations range between 536-565 metres above mean sea level (“mamsl”).



FIGURE 6: 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 10 m 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 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. Total rainfall is sparse within the Molo area, with yearly precipitation ranging from 30 cm to 50 cm. The rainy season causes difficulty in travelling off the main highways and for exploration, effectively limiting drilling to the dry season.

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

The village of Fotadrevo (where the Company has its base camp) is located to the west of the Molo deposit area. The village has been a labour source during the exploration programmes on Molo and will most likely provide a portion of the workforce during any future development. A few basic goods are commercially available in the village, however, the main centre for support of exploration and development are the cities of Toliara and Antananarivo. Two 40 kVA diesel-powered generators provide power to the Company's base camp.

A cellular telephone tower is located in Fotadrevo, which provides convenient coverage. No potable water is currently available within the project area. A 123 mm diameter well has been drilled to a depth of 42m within the base camp compound, which provides non-potable water.

5.5 SECURITY

As Madagascar is an island 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, few police, or other security patrols are 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.

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 11 shows a summary of the historical exploration activities previously on the property.

Note that only exploration in 2011 is directly related to the exploration of the graphite prospects, and only Phase II of that year concerns the Molo graphite deposit.

TABLE 11: HISTORICAL ACTIVITIES ON THE PROJECT

Date	Activity	Company Responsible
2011	Prospecting (538 grab samples) over areas of historical graphitic occurrences (BRGM) on the Project	The Company
	Diamond Drilling (7 holes). Metallurgical samples selected from Molo-11-07	Boart Longyear
	Trenching programme (1 trench)	The Company
	Geologic mapping at 1:1000 scale	The Company
	EM-31 ground geophysical survey (52.2 line kms)	The Company

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 remainder of this Technical Report.

7 GEOLOGICAL SETTING AND MINERALISATION

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), whereupon the two land masses split apart. On a regional scale Madagascar can be described as being formed by two geological entities, a Precambrian crystalline basement, and a much younger Phanerozoic sedimentary cover (Figure 7) 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 supercontinent of Gondwana.

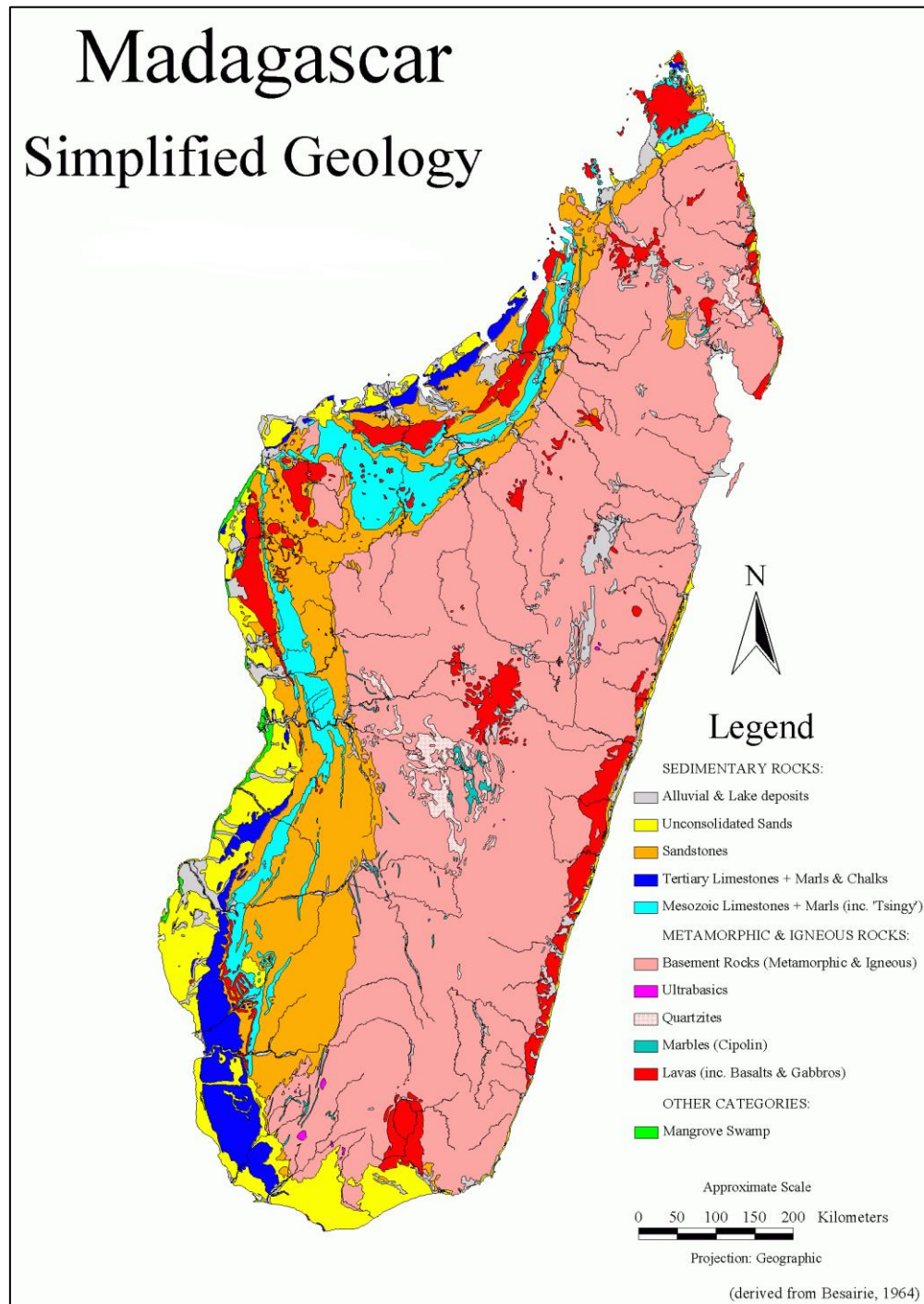


FIGURE 7: GEOLOGICAL MAP OF MADAGASCAR SHOWING THE DISTINCTIVE CRYSTALLINE BASEMENT AND SEDIMENTARY BASINS IN THE WEST

Source: Besarie (1964).

The geology of the basement of Madagascar is composed of intercontinental tectonic blocks made up of ancient poly-deformed, high-grade metamorphic rocks and later igneous intrusions. The tectonic and metallogenic basement framework was originally subdivided into four blocks (Besarie, 1967), these being the: northern Bemarivo Block; northeastern Antongil Block; central Antananarivo Block; and the southern Bekily Block. The Molo deposit lies entirely within the bounds of the Bekily Block (Figure 8). 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 9). 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 interchangeably.

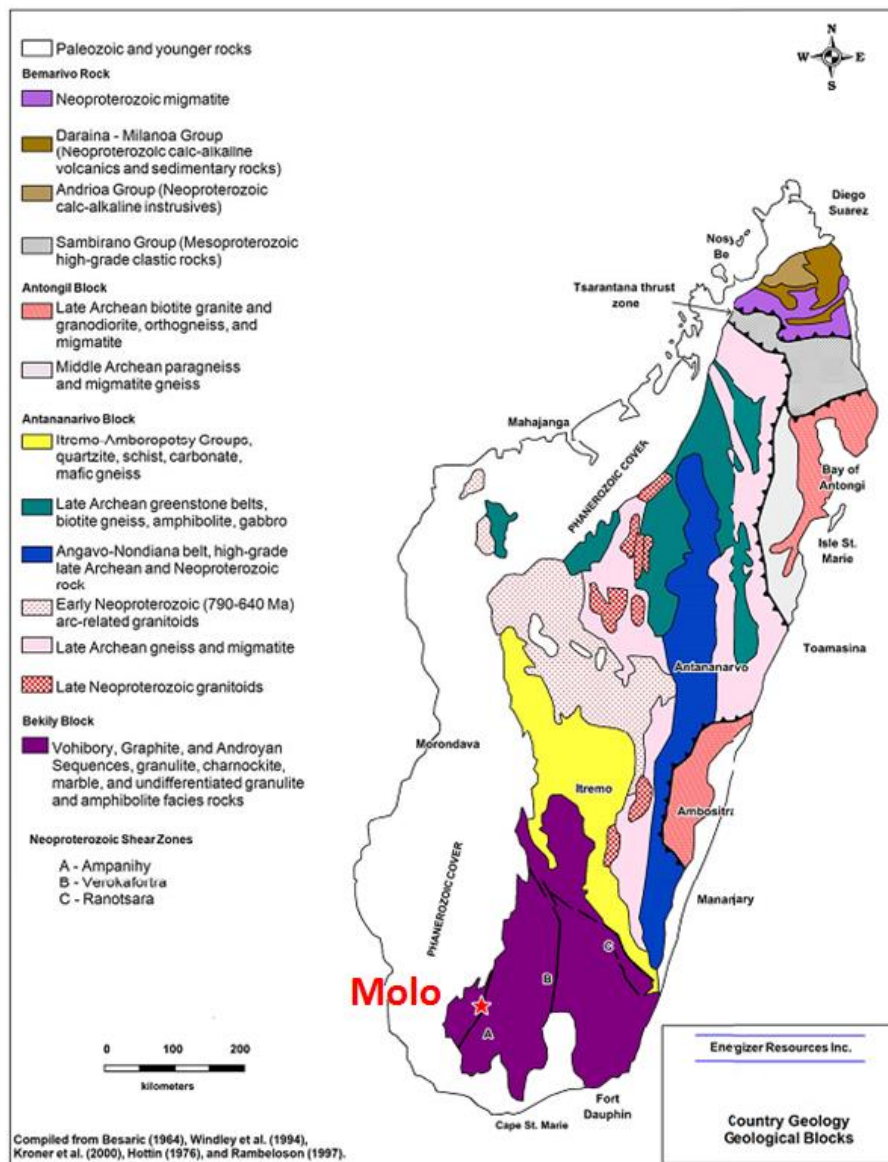


FIGURE 8: COUNTRY GEOLOGY: GEOLOGICAL BLOCKS

Source: AGP Mining Consultants (2011).

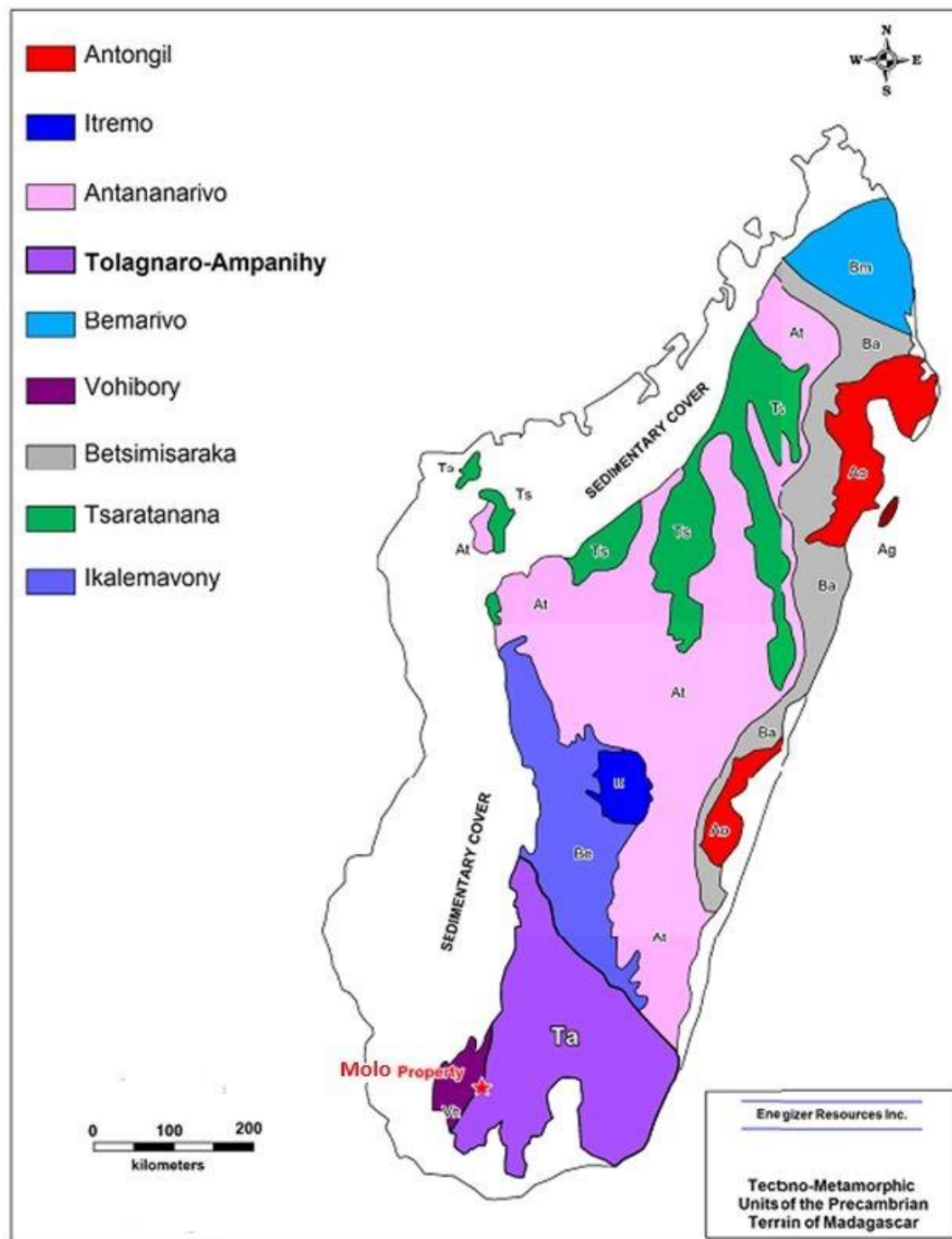


FIGURE 9: TECTONO-METAMORPHIC UNITS OF THE PRECAMBRIAN TERRAIN OF MADAGASCAR

(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 10). 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 3-11 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, cross-cut the region. A third set of en-echelon shears forms part of the early Palaeozoic Ranotsara Shear Zone that cuts the basement in a northwest-southeast 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 D1), during which northeast verging recumbent sheath folds and ductile thrusts were formed, are dated between 647-627 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 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 620-610 Ma. Between 609 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 and 520 Ma suggests that this thermal event is part of an extended period of low-pressure (3 to 5 k 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 to 490 Ma, prevailing low-pressure, high-temperature amphibolite-granulite facies rapidly gave way to greenschist facies conditions.

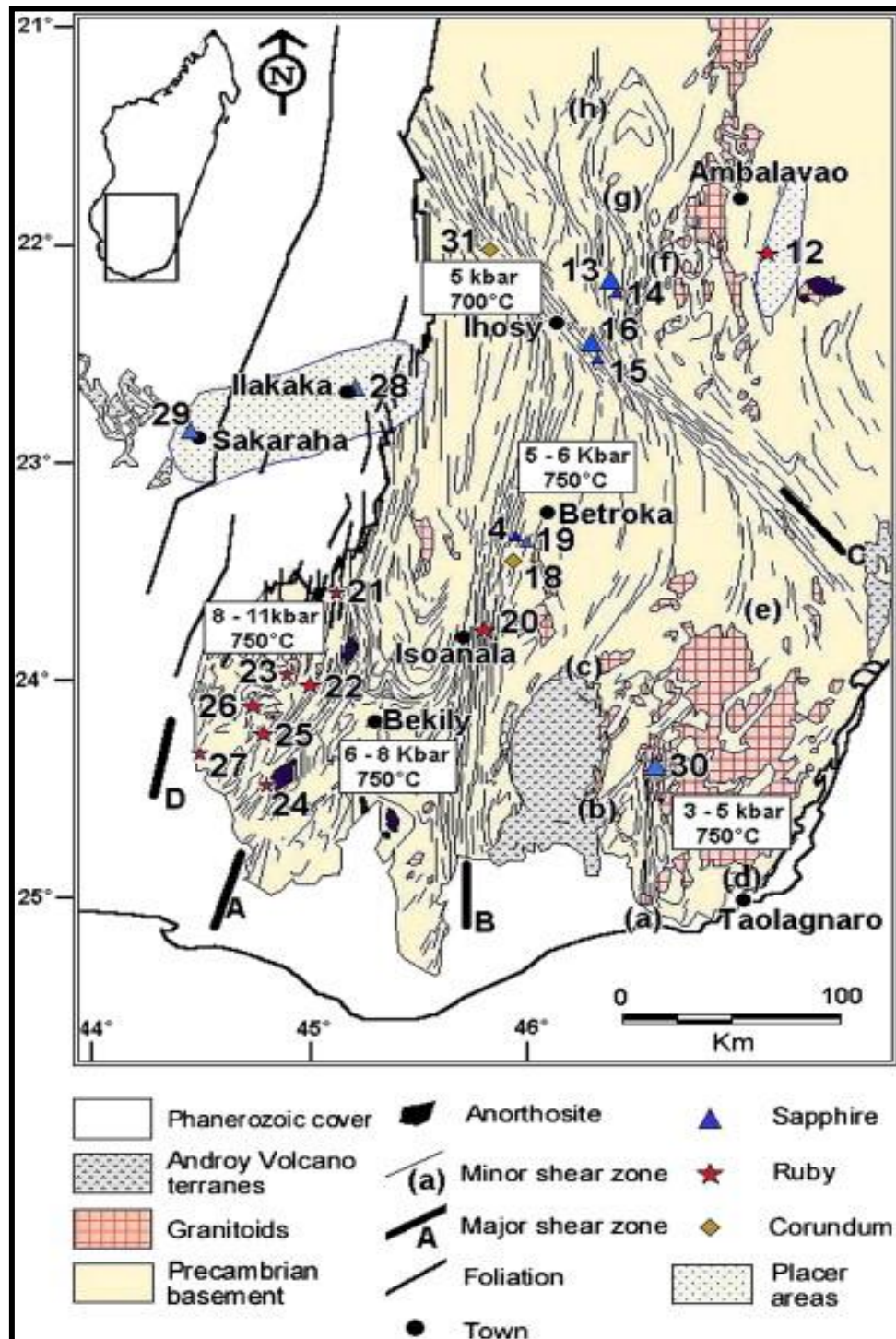


FIGURE 10: STRUCTURAL AND LITHOLOGICAL SKETCH MAP OF SOUTH-EAST MADAGASCAR SHOWING THE POSITIONS OF THE MAJOR SHEAR ZONES

(From: 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 11). 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 11 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 ($\pm$ biotite, garnet) quartzo-feldspathic gneiss, marble, chert, quartzite, and amphibolite gneiss. The metaplutonic rocks include migmatitic ($\pm$ hornblende / diopside, biotite, garnet) feldspathic gneiss of monzodioritic to syenitic composition, biotite granodiorite, and leucogranite.

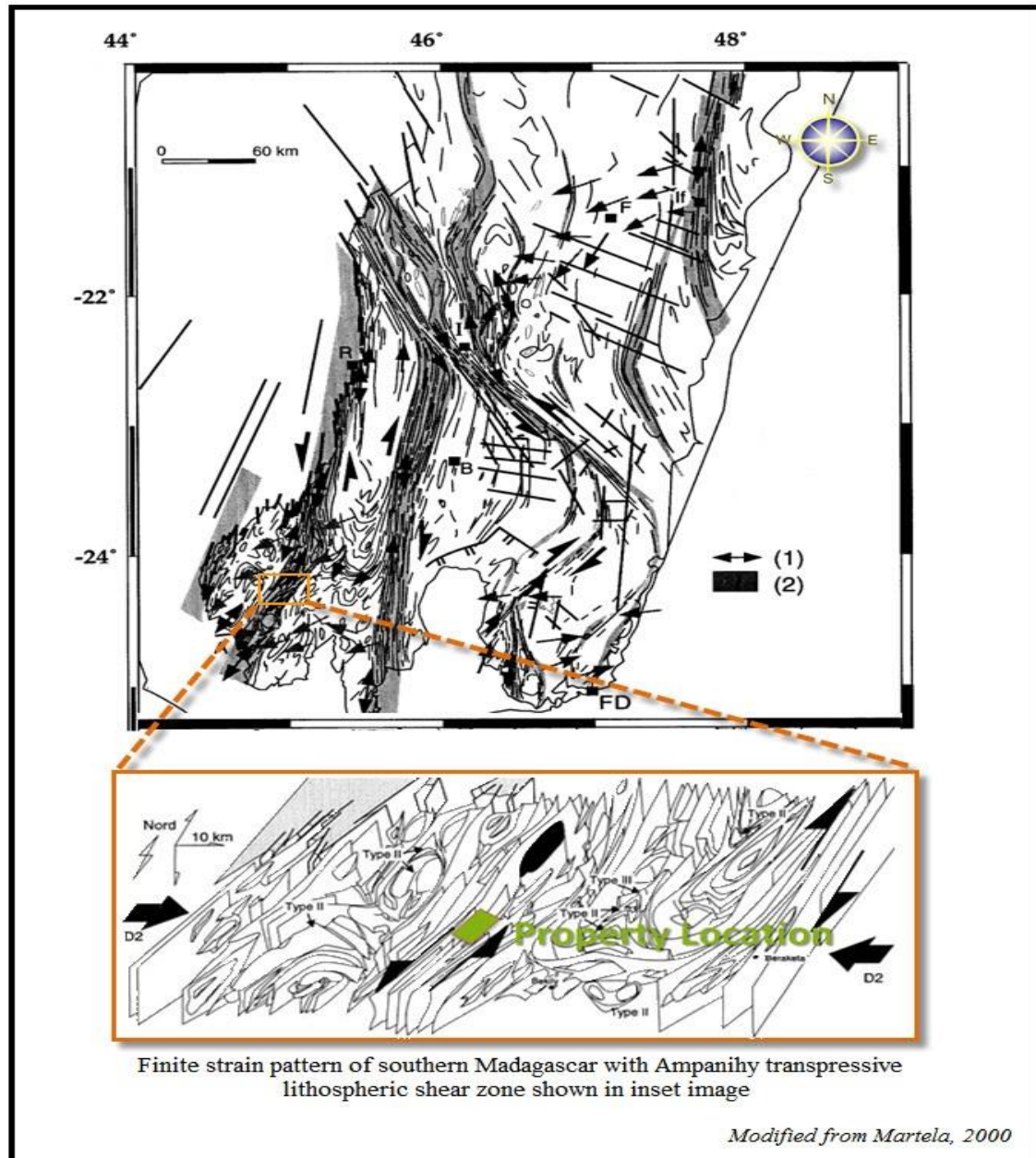


FIGURE 11: POSITION OF THE MOLO PROJECT AREA WITHIN THE OVERALL STRAIN PATTERN DOCUMENTED FOR SOUTHERN MADAGASCAR

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, subequigranular 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 ($\leq 1\%$) 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 breccia;

- Concordant layers intercalated 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-feldspathic 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 ($\pm$ hornblende/diopside, biotite, garnet) feldspathic gneiss. The feldspathic gneiss is comprised of a mixture of quartz-poor constituent lithologies.

7.2.2 **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 ($\pm$ 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 consists of multi-folded graphitic strata with a surficially exposed strike length of over two kilometres (Figure 12). Outcrop mapping and trenching on Molo has shown the surface geology to be dominated by resistant ridges of graphitic schist (Figure 13) and graphitic gneiss, with fracture-lined vanadium mineralisation, as well as abundant graphitic schist float.

Geological modelling (Figure 14) has shown that the deposit consists of various zones of mineralised graphitic gneiss, with a barren footwall composed of garnetiferous gneiss (Figure 15). The host rock of the mineralised zones is graphitic gneiss.

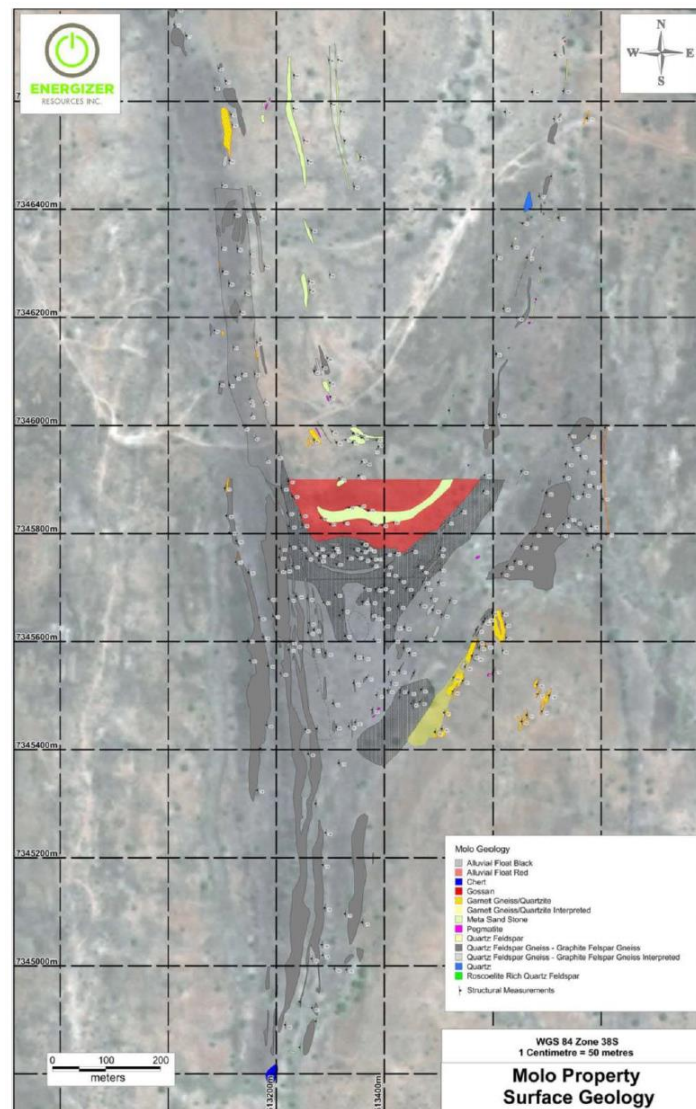


FIGURE 12: MAP SHOWING THE SURFACE GEOLOGY OF THE MOLO DEPOSIT

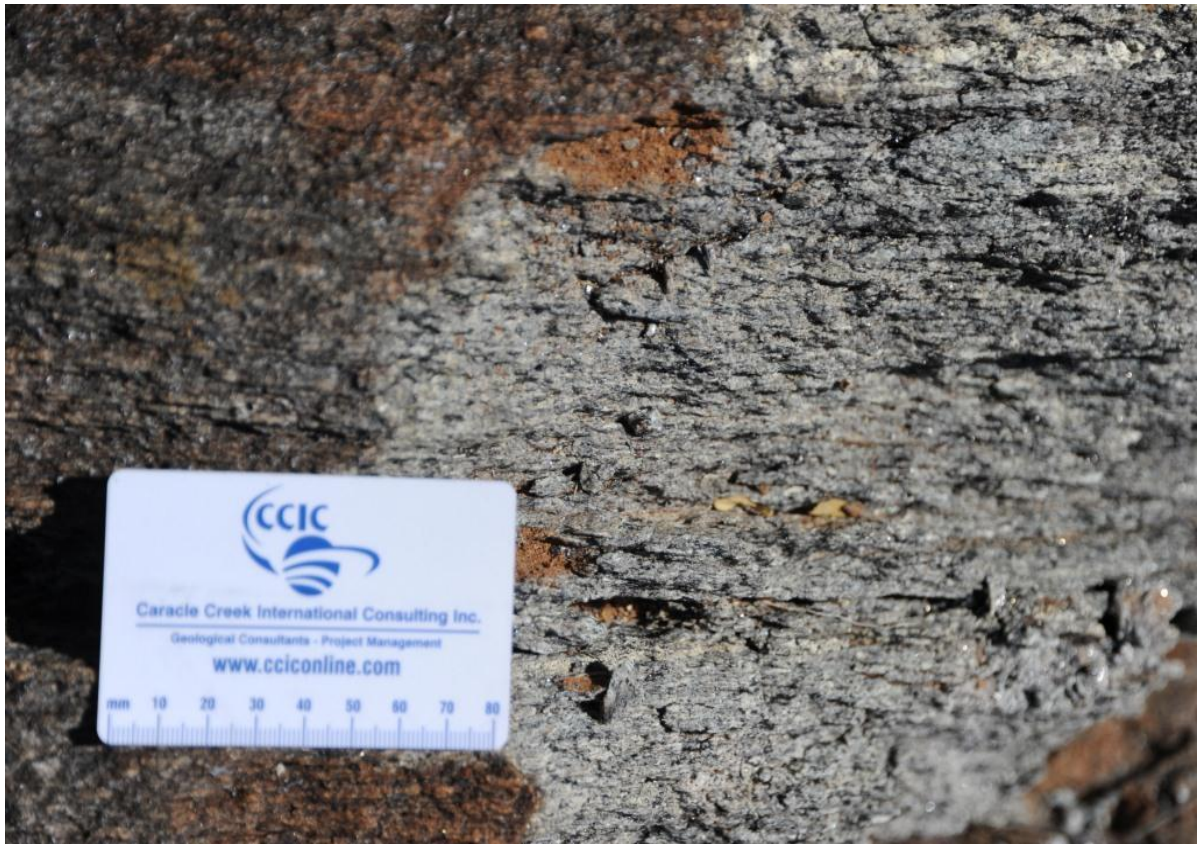


FIGURE 13: OUTCROP EXPOSURE OF GRAPHITIC SCHIST ON THE MOLO PROJECT

Photograph taken during the site visit of the 10th of May 2012

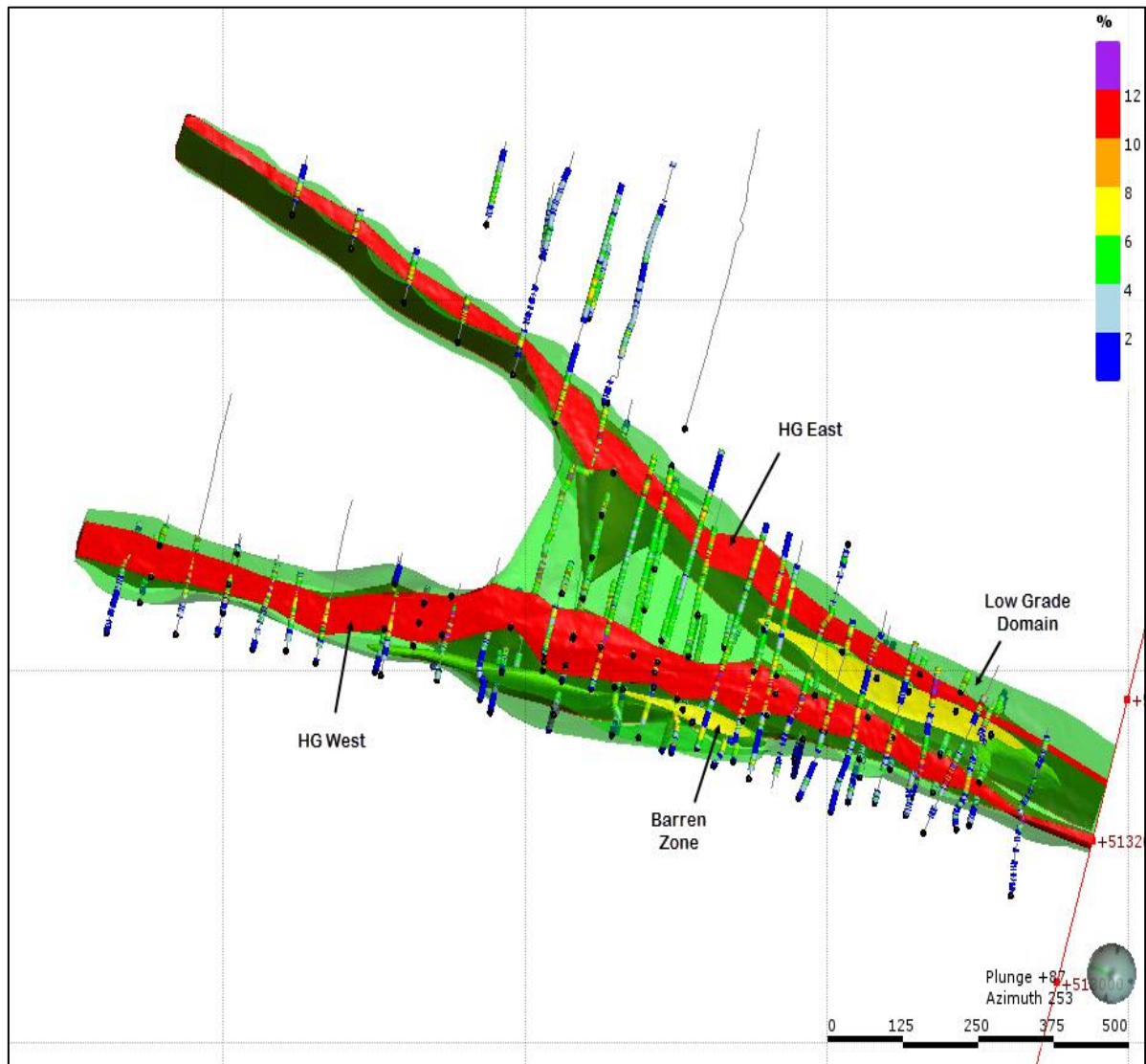


FIGURE 14: GEOLOGICAL MODEL OF THE MOLO DEPOSIT SHOWING THE NATURE OF THE TWO TIGHTLY OPPRESSED ‘HIGH’ GRADE (HG EAST AND HG WEST) MINERALIZED ZONES



FIGURE 15: GARNET GNEISS IN THE FOOTWALL OF MOLO-4 AT A DEPTH OF 132 M

No academic studies have been undertaken on the graphitic schists and gneisses of the Molo deposit and at present the deposit is still not fully understood. There is, however, no indication of secondary hydrothermal, or other transported, post-metamorphic graphitic mineralisation or upgrading and the present distribution and crystallinity of the graphite zones seem to be primarily due to regional metamorphic and structural events. The length, width, depth and continuity of mineralisation for the Molo deposit are more fully described under Section 14 “Mineral Resource Estimates”.

8 DEPOSIT TYPES

The Green Giant-JV Project hosts at least two different deposit types:

- 1) Metamorphosed black shale / roll front redox vanadium deposits, and deposit types.
- 2) Flake graphite deposits. The metamorphosed black shale / roll front redox vanadium deposits are well described in previous reports (e.g. Scherba and Chisholm, 2008) and are not repeated here.

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 be from natural source. Most natural sources are considered to be 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

Economic deposits of graphite include five main geological types:

Flake graphite disseminated in metamorphosed, silica-rich

- 1) Sedimentary rocks
- 2) Flake graphite disseminated in marble
- 3) Amorphous deposits formed by metamorphism of coal or
- 4) Carbon-rich sediments
- 5) Veins filling fractures, fissures, and cavities in country rock
- 6) Contact metasomatic or hydrothermal deposits in metamorphosed

Natural graphite of economic value can be divided into two main classes, these being:

- 1) Disseminated flake
- 2) Crystalline vein (fibrous, or columnar)

Most, if not all, of the world's deposits of flake graphite occur in metamorphic rocks of Precambrian age. Flake graphite is a lamellar form found in metamorphic rocks, such as marble gneiss, and schist. Each flake is separate, having crystallized as such in the rock. In many cases, pegmatitic veins have intruded the rocks.

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. Here 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 programme. 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 programme with the goal of delineating new graphitic trends and comparing them to those associated with vanadium mineralisation. During the course of this reconnaissance exploration programme 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 <http://www.mintek.co.za/> 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.

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 consequence of the 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 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).

During the 2011 field season mapping activities increased and the identification of the graphitic prospect areas led to smaller scale mapping (1:1000 scale) over areas which appeared to have the greatest graphitic potential. Areas included were Molo (Figure 3), Fondrana, Fotsy and Seta.

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 graphitic 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 (Figure 16). Subsequently an additional nine trenches (totalling 1,876 metres) have been excavated as part of the 2014 exploration programme.



FIGURE 16: 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 below as Figure 17.

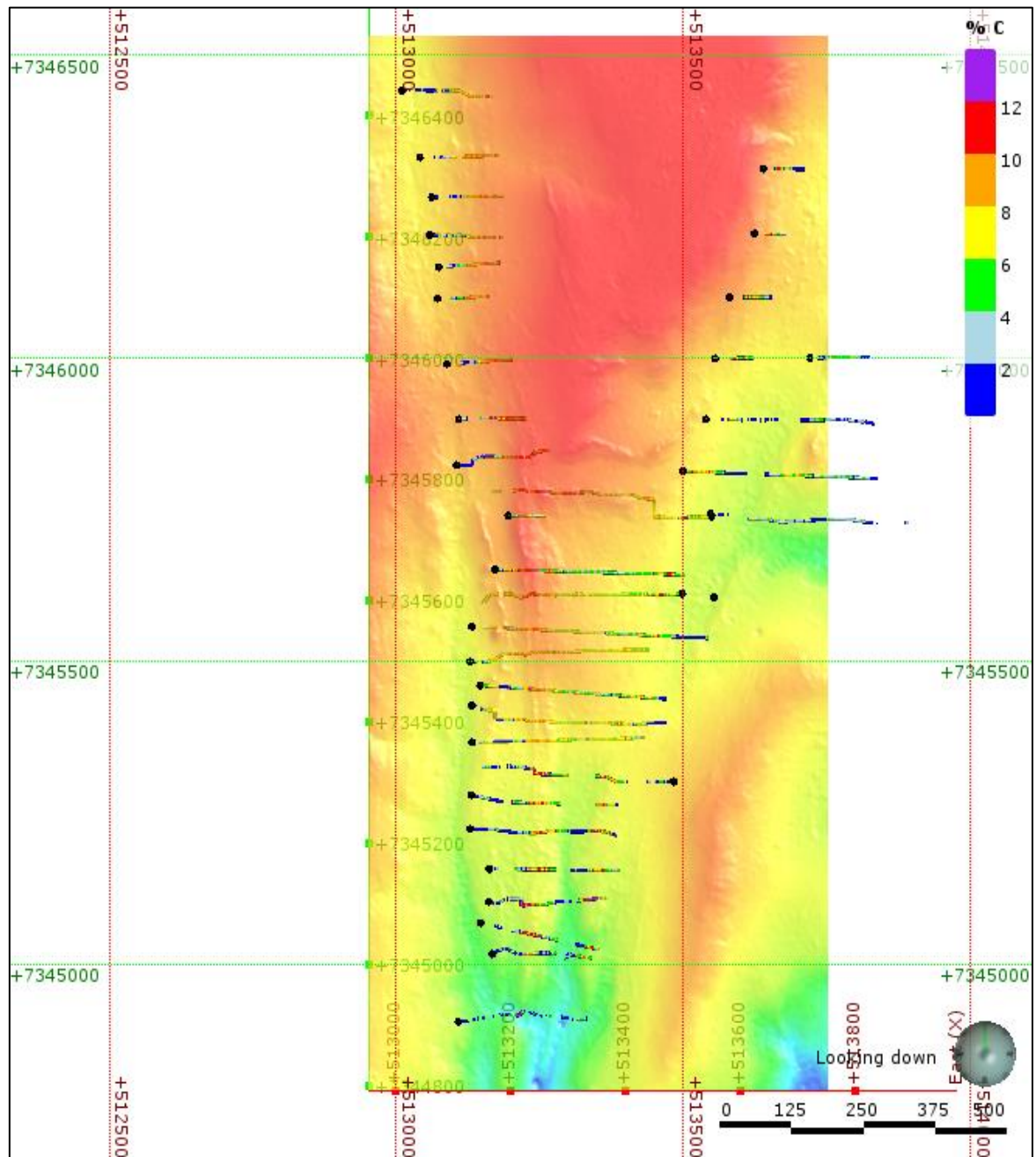


FIGURE 17: PLAN SHOWING THE POSITIONS AND GRADES OF ALL THE TRENCHES EXCAVATED ON THE MOLO DEPOSIT, OVERLAIN ON THE TOPOGRAPHIC ELEVATION MAP

9.2.1 Trench Sampling

Standardized sampling methods include two-meter-long continuous chip samples approximately 4 cm wide being collected along the northern edge of the trench floor, consisting of about 3 kg 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, mineralisation, depth and any other notable aspects
- All samples are brought back to the camp each night for storage in the secure 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 12 below. This data was used in the estimation process.

TABLE 12: 2012/2014 MOLO TRENCH RESULTS

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
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

9.3 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, 160.5 line kilometres of EM31 surveying was completed over five target areas, full details of which are available on the Company’s web page:

(<http://energizerresources.com/projects/green-giant-graphite.html>).

This data was used to plan the original (2011) boreholes drilled on Molo (Figure 19).

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 18) 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.

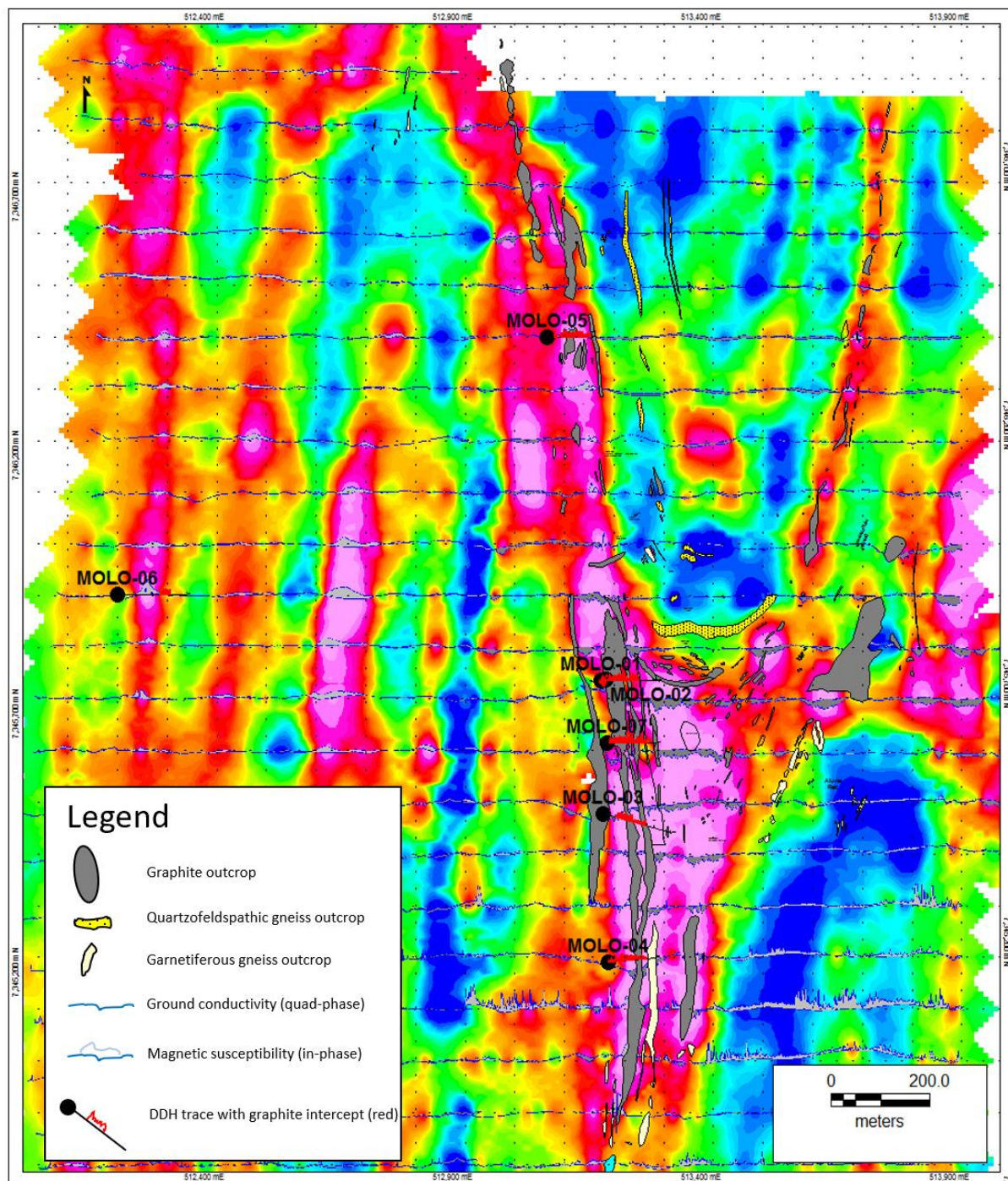


FIGURE 18: EM31 GEOPHYSICAL SURVEY MAP OF THE MOLO DEPOSIT SHOWING THE POSITIONS OF HOLES MOLO-01 TO MOLO-07 DRILLED DURING 2011

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.

9.4 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 of 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, Molo 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 Green Giant Property. Numerous diamond drilling programmes have been conducted over the Green Giant (and formerly the Three Horses) property in the past seven years 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. This resulted in 19 holes being drilled by Boart Longyear over the Molo, Fotsy, Fondrana and Seta areas, for a combined 1701 m of diamond drilled core. Of these 19 holes, one larger (PQ) diameter hole was completed and sent to Activation Labs (www.actlabs.com/) ("Actlabs") in Ancaster, Ontario, Canada for metallurgical analysis.

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 were completed on Molo during 2012. During 2014 an additional 32 diamond drill holes (totalling 2,063 metres) were completed. With this most recent drill programme, a total of 80 diamond drill holes (Figure 19), (totalling 11,660 meters) have now been completed on Molo, and these were used for the mineral resource estimations.

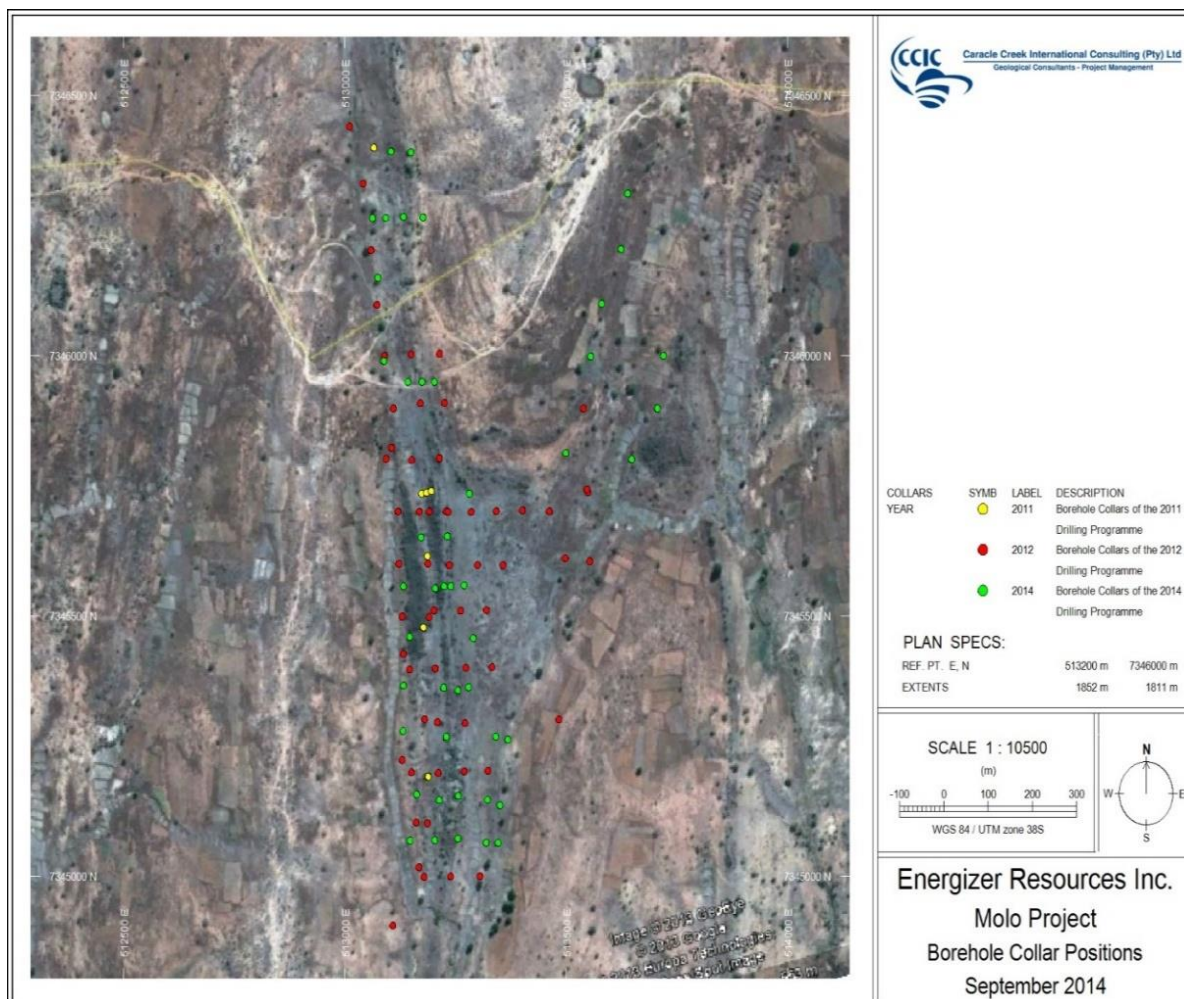


FIGURE 19: BOREHOLE COLLAR POSITIONS FOR ALL OF THE KNOWN BOREHOLES DRILLED ON THE MOLO DEPOSIT

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 70 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 in order 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.

10.1.1 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-hour 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.

10.1.2 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.

10.1.3 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.

10.1.4 Core Photography

The core is photographed in groups of two boxes (Figure 20) and then forwarded for cutting. Core is typically photographed wet.



FIGURE 20: CORE PHOTOGRAPHS OF BOREHOLE MOLO-05 CORE (BOXES 17 AND 18) AS SUPPLIED TO CCIC BY THE COMPANY

10.1.5 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 ± 3 m. 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 one metre 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 (Figure 21).



FIGURE 21: CONCRETE MARKER SHOWING THE COLLAR POSITION OF BOREHOLE MOLO 12-02

10.1.6 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.

10.1.7 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:

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

10.1.8 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 campsite until shipment

10.1.9 Diamond Drill Results

All results from the 2011, 2012 and 2014 diamond drill programmes are presented in Table 13 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 13: 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
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

BHID	EASTING	NORTHING	ELEV	LENGTH	DIP	AZIMUTH
MOLO-12-27	513226	7345702	554.22	212	-50	270
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, ANALYSES AND SECURITY

11.1 SAMPLE PREPARATION, ASSAY AND ANALYTICAL PROCEDURES

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

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

11.2 QA/QC

In order 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.

11.2.1 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 Figure 22 represents the previous and current campaigns respectively). Blanks for the 2014 campaign shows that the majority of samples have concentrations that lie within the blank + 3 times detection limit threshold, with the maximum outlier being 0.57% C.

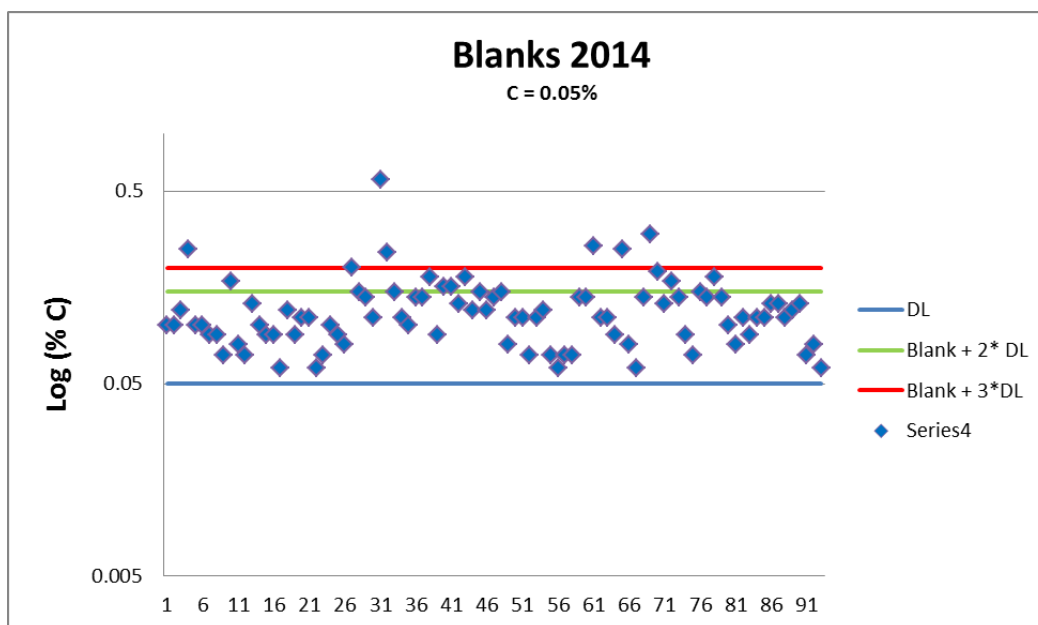


FIGURE 22: PLOT OF LOG %C VERSUS THE DATE OF THE ANALYSIS 2014 FOR THE BLANKS

11.2.2 Standards

Because of the difficulty in sourcing certified standard reference material (“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 (Figure 23, Figure 24 and Figure 25).

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 indicate a slight positive bias when compared to the recommended value. For GGC-01, all of 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.

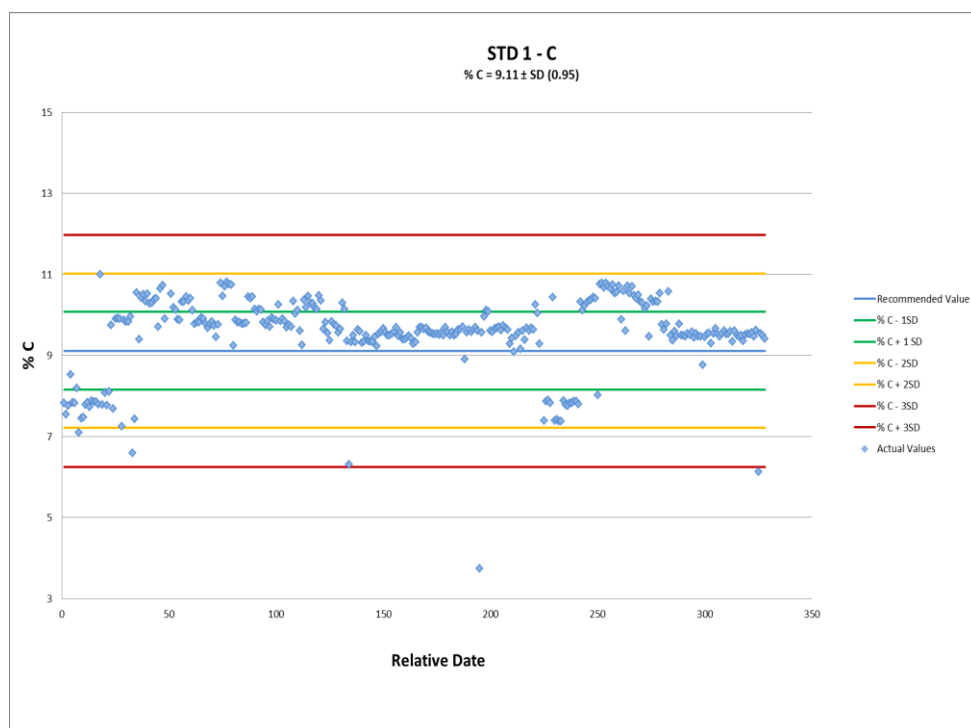


FIGURE 23: GRAPH SHOWING CARBON CONCENTRATION AS ANALYSED IN STD 1C

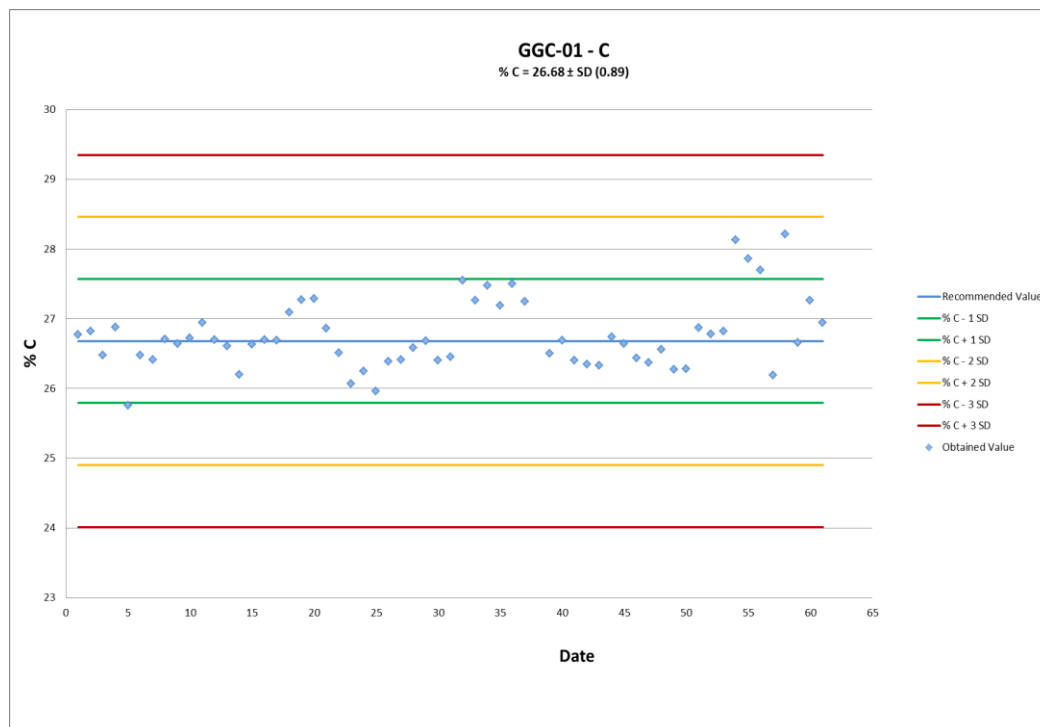


FIGURE 24: GRAPH SHOWING CARBON CONCENTRATION AS ANALYSED IN GGC-01

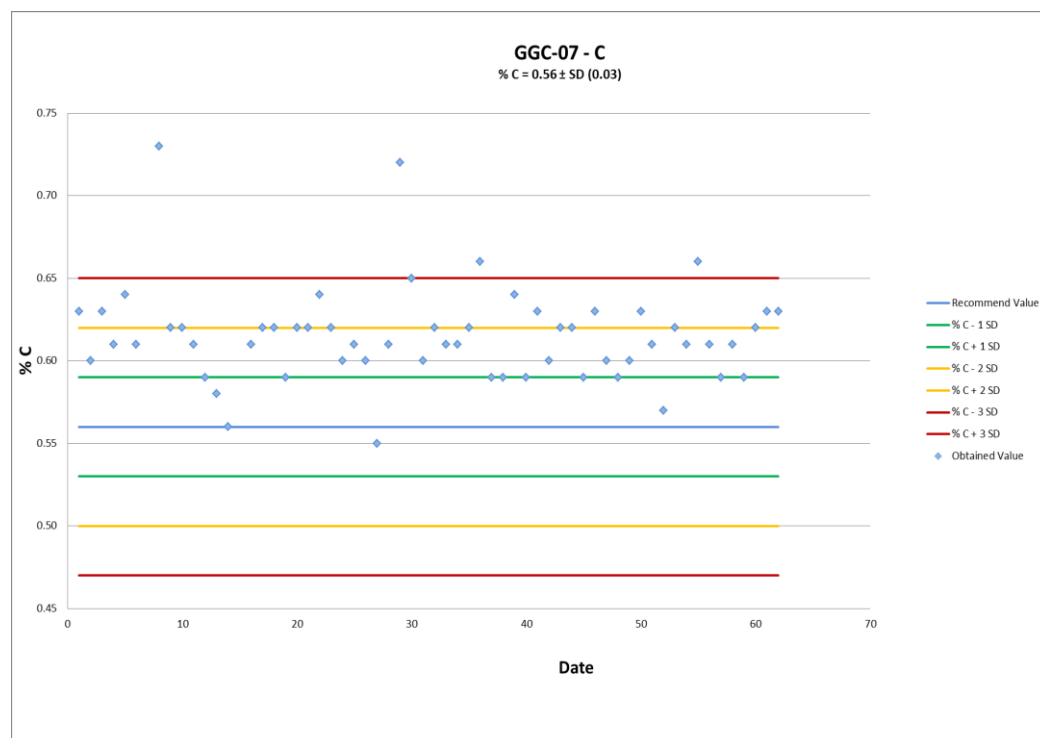


FIGURE 25: GRAPH SHOWING CARBON CONCENTRATION AS ANALYSED IN GGC-07

11.3 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. (Figure 26) 2012 and 2014 samples are shaded in blue and red respectively. The majority 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.

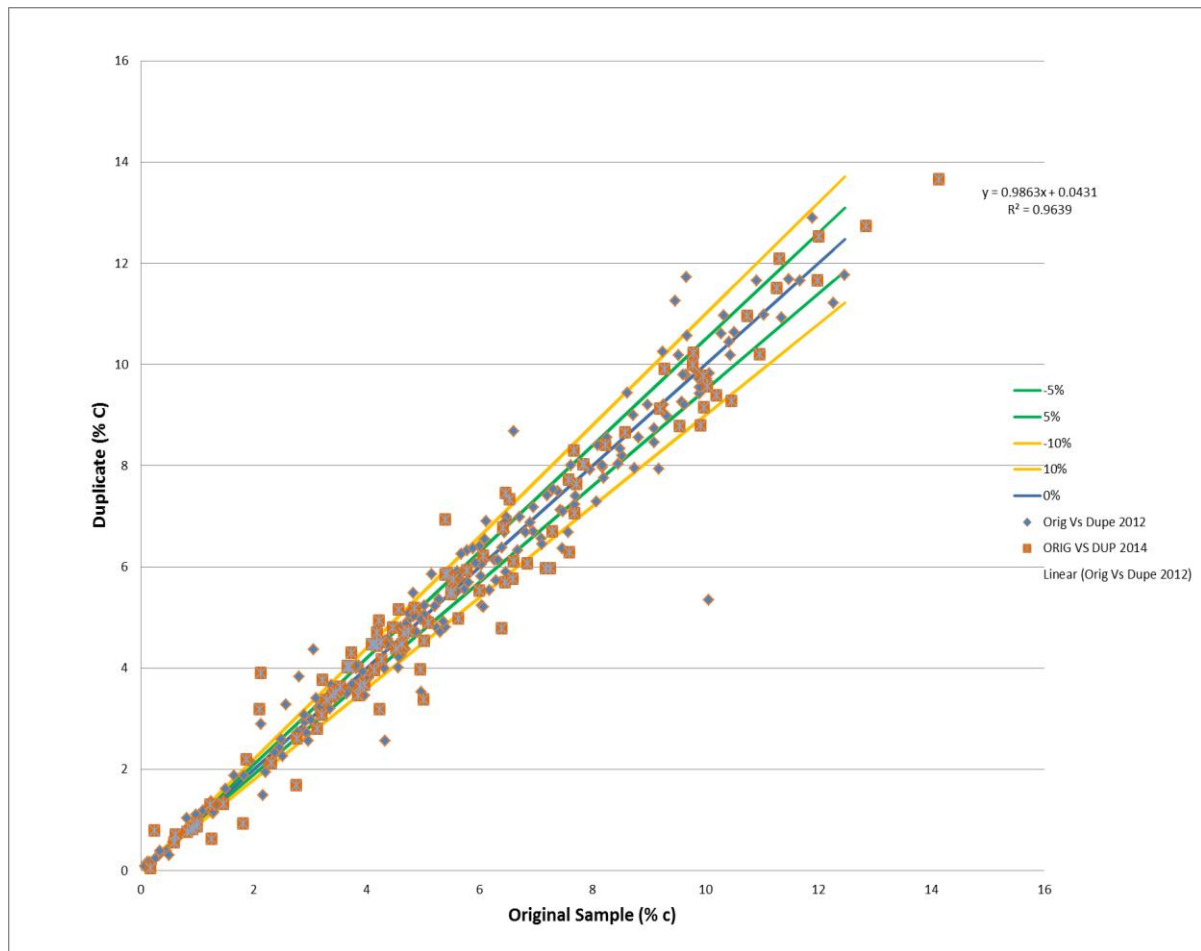


FIGURE 26: ORIGINAL (ORIG) VERSUS DUPLICATE (DUPE) PLOTS

11.4 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 is as follow:

- Pieces of whole core were collected, and the rock types documented
- The selected section of core is then dipped into wet paraffin wax and allowed to dry. This seals the core to avoid the absorption of moisture

The pieces of core are weighed dry, followed by weighing in a water bath.

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

12 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 the Company contained 80 drill holes totalling 11,660 m 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.

12.1 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 (Figure 27). 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.

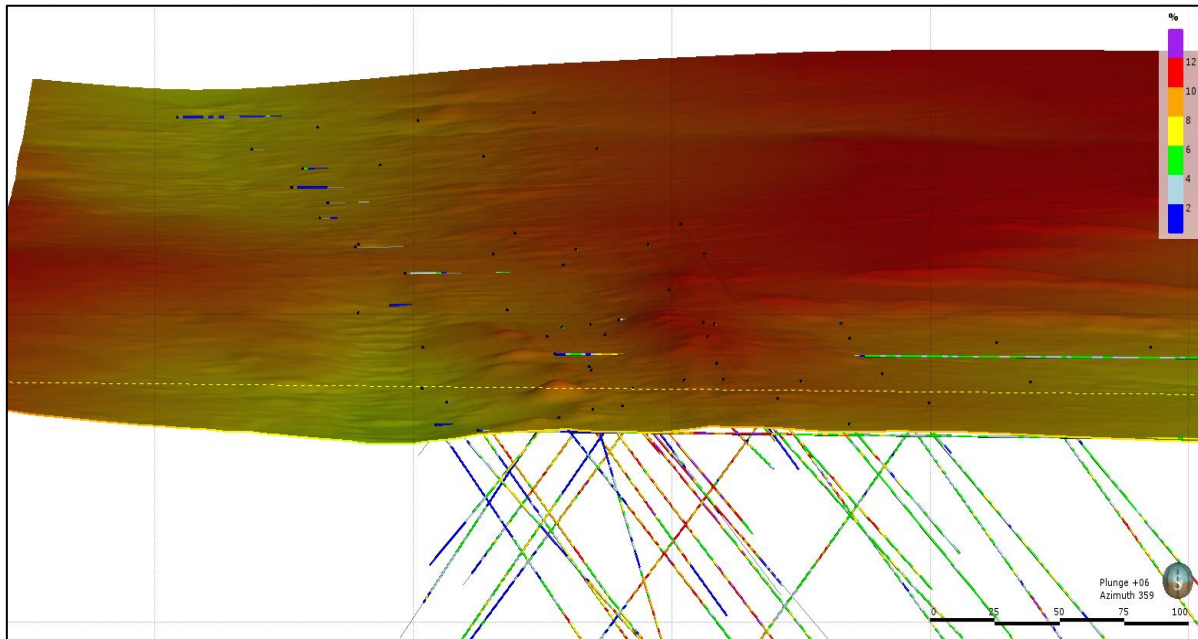


FIGURE 27: EAST-WEST CROSS-SECTION SHOWING THE COLLAR POSITIONS WITH RESPECT TO TOPOGRAPHY

12.2 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 (Figure 28). 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 were also examined.



FIGURE 28: BOREHOLE FOTSY-06 - ONE OF THE BORE HOLES CHECK LOGGED DURING MAY 2012 SITE VISIT

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 here 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 testwork. 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, and

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 856,071 T 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 Phase 2 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 market for graphite concentrates grading as low as 80% carbon, the price of a 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 a concentrate, while minimizing the percentage of small flakes.

It is pertinent that the decisions made in 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 29. 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.

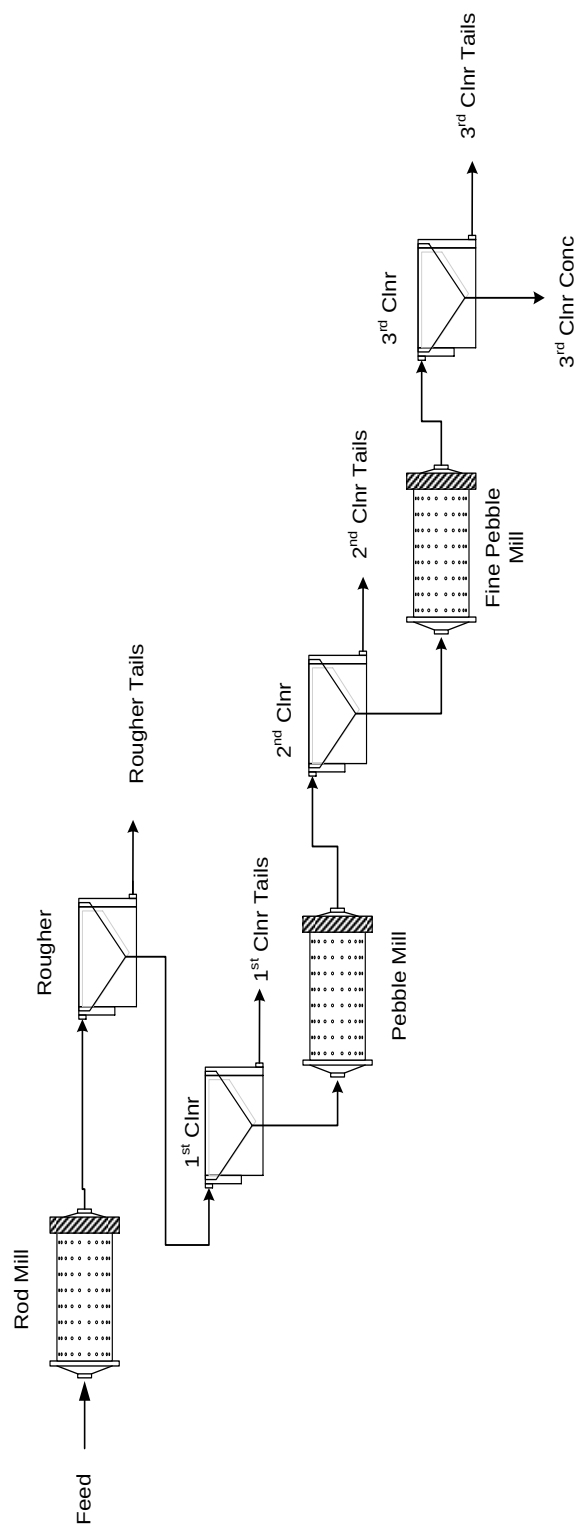


FIGURE 29: FLOWSHEET FOR MOLO MINERALIZATION – MINTeK

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 200-tonne bulk trench sample that was generated for pilot plant testing. For every 2 metres of the total trench length of 160m, approximately 2.5 tonnes of material were extracted for piloting. A 5 kg sub-sample was then removed from each 2.5 tonne 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 P80 = 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 3rd 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 30 and the size fraction analysis results of the combined concentrate are presented in Table 14. 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.



TABLE 14: 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 tonne 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 10.8 tonnes 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 100 tonnes 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 31. 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 2 m 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 were removed from each interval to generate the material for the SGS laboratory scale flowsheet development program described above.

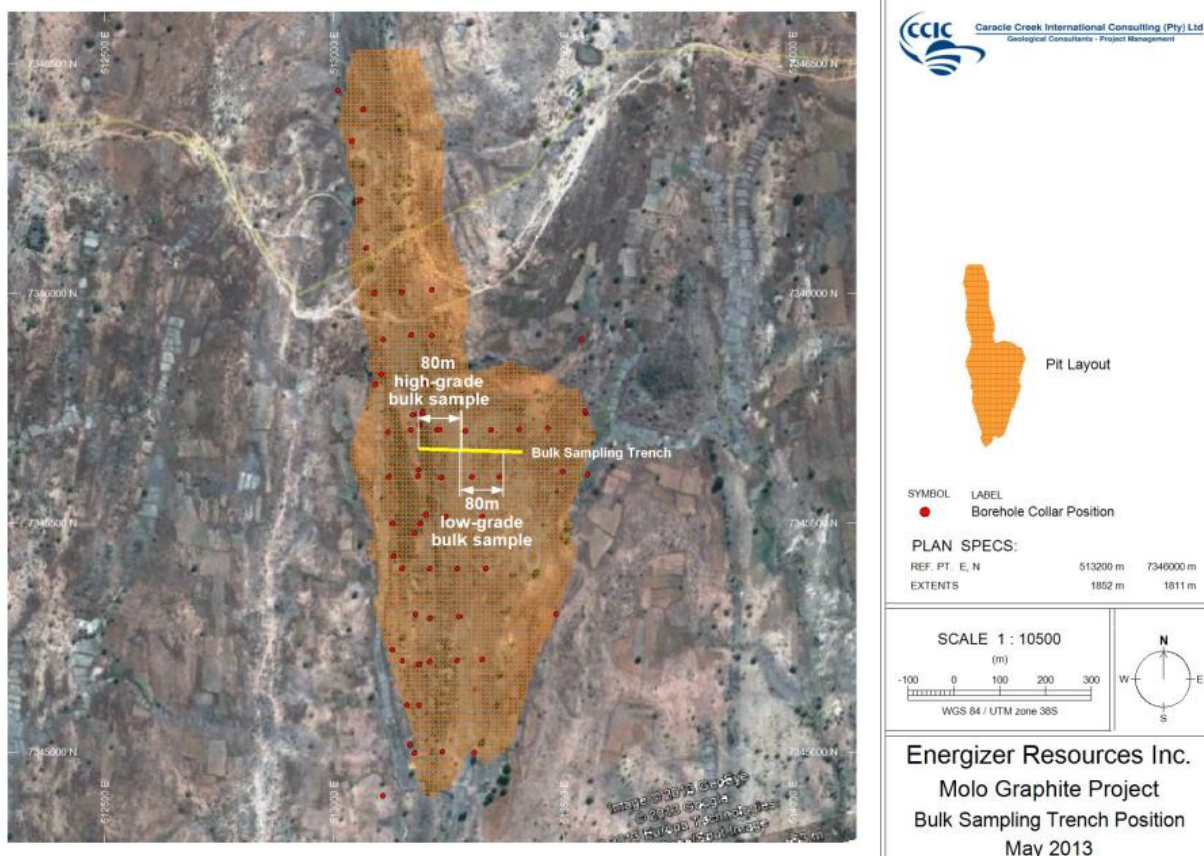


FIGURE 31: MAP SHOWING POSITION OF BULK SAMPLING TRENCH

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 15.

TABLE 15: SUMMARY OF COMMINATION TEST RESULTS FOR PILOT PLANT COMPOSITES

Sample Name	Relative Density	JK Parameters			RWI (kWh/t)	BWI (kWh/t)	AI (g)
		A x b ¹	A x b ²	T _a			
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 tonnes of the Molo bulk composite using the flowsheet depicted in Figure 32. 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.



A total of fifteen (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 were assayed for total carbon and the low-grade tailings streams also for graphitic carbon.

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

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

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

The average feed sizes to the flash and rougher flotation circuits were P80 = 835 microns and P80 = 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 16.

TABLE 16: 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 P80 = 268 microns.

The average size distribution and total carbon grade of each size fraction are presented in Table 17.

TABLE 17: 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 18.

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 18: 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 constitute 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 require 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 19.

TABLE 19: 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 (4) comminution composites were tested at SGS and two (2) additional composites were shipped to Mintek in South Africa for testing. The make-up of the six (6) comminution composites is shown in Table 20.

TABLE 20: 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 21 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 21: 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 5-year pit layout were 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 22: Summary of Comminution Test - Mintek 2014. The Bond rod and ball mill test results are consistent with the results obtained for the four composites tested at SGS Lakefield.

TABLE 22: SUMMARY OF COMMINTION TEST - MINTEK 2014

Sample Name	Grindmill Feed Top Size -6.7mm		Grindmill Feed Top Size -9.5mm		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 (0 to 14m depth) and the deep section (28 to 56m depth) of the deposit.

Two drill holes of each the high-grade (HG) and the low-grade (LG) mineralization were 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 P80 = 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 twenty different drill hole composites, which are specified in Table 23.

TABLE 23: VARIABILITY COMPOSITES – PHASE I

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

The optimized flowsheet and conditions shown in Figure 33 were 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.

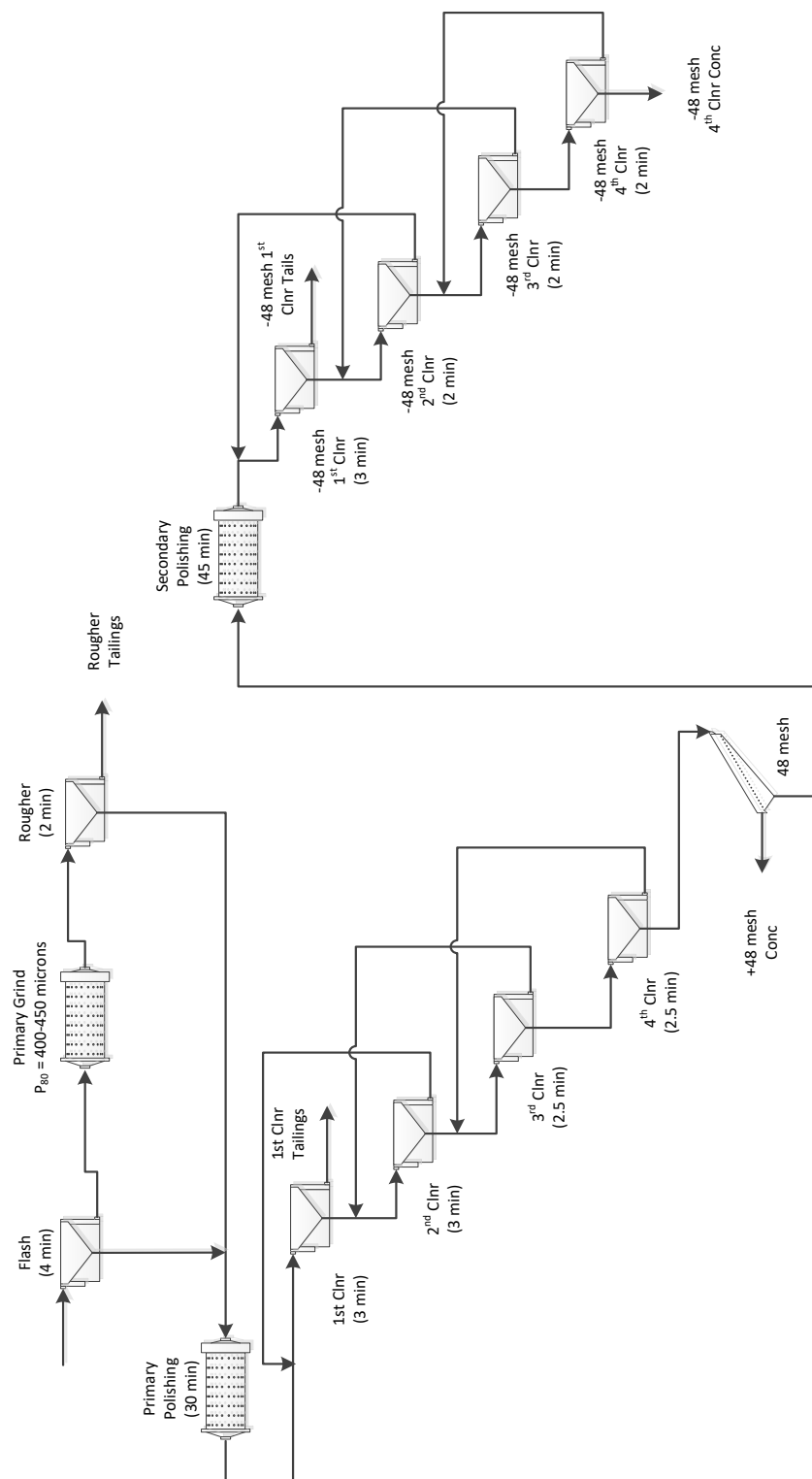


FIGURE 33: OPTIMIZED FLOWSHEET

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 Figure 34, 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 to 14m) and medium (14 to 28m) depth composites was almost identical at 44.6% and 44.4%, respectively. Only the concentrate from deep (28 to 56m) composites contained a slightly higher percentage of +80 mesh material at 50.2% mass recovery.

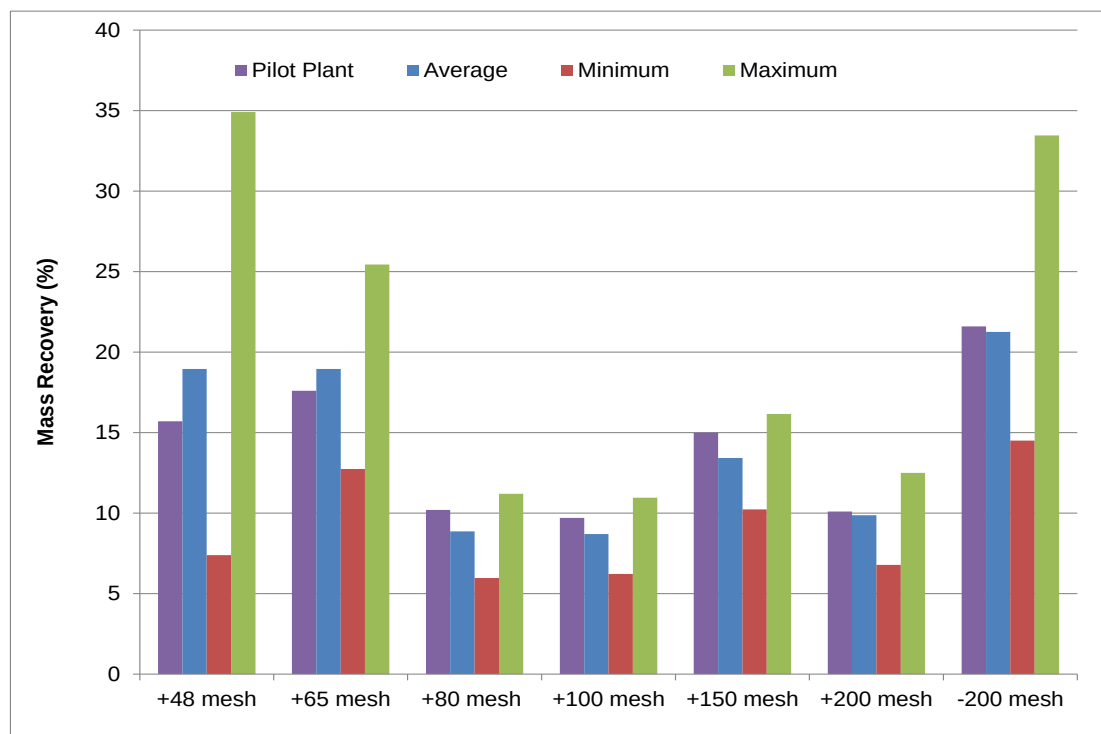


FIGURE 34: RANGE OF MASS RECOVERY INTO SIZE FRACTIONS (V1 TO V32 AND PILOT PLANT)

The average, minimum, and maximum total carbon grades of tests V1 to V32 for each size fraction are depicted in Figure 35 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.

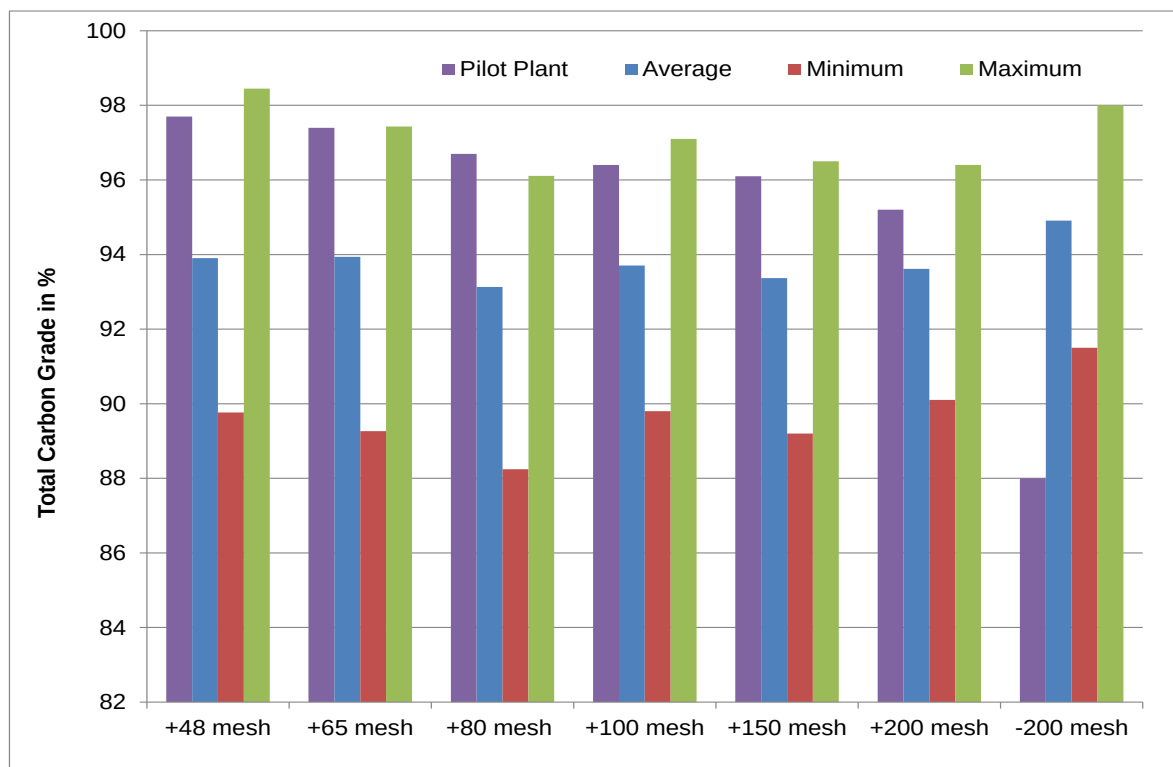


FIGURE 35: RANGE OF CARBON GRADES OF SIZE FRACTIONS (V1 TO V32 AND PILOT PLANT)

The variability in the concentrate grade of the variability composites is further illustrated in Figure 36 which depicts the combined concentrate grade of each of the thirty-two (32) variability flotation tests.

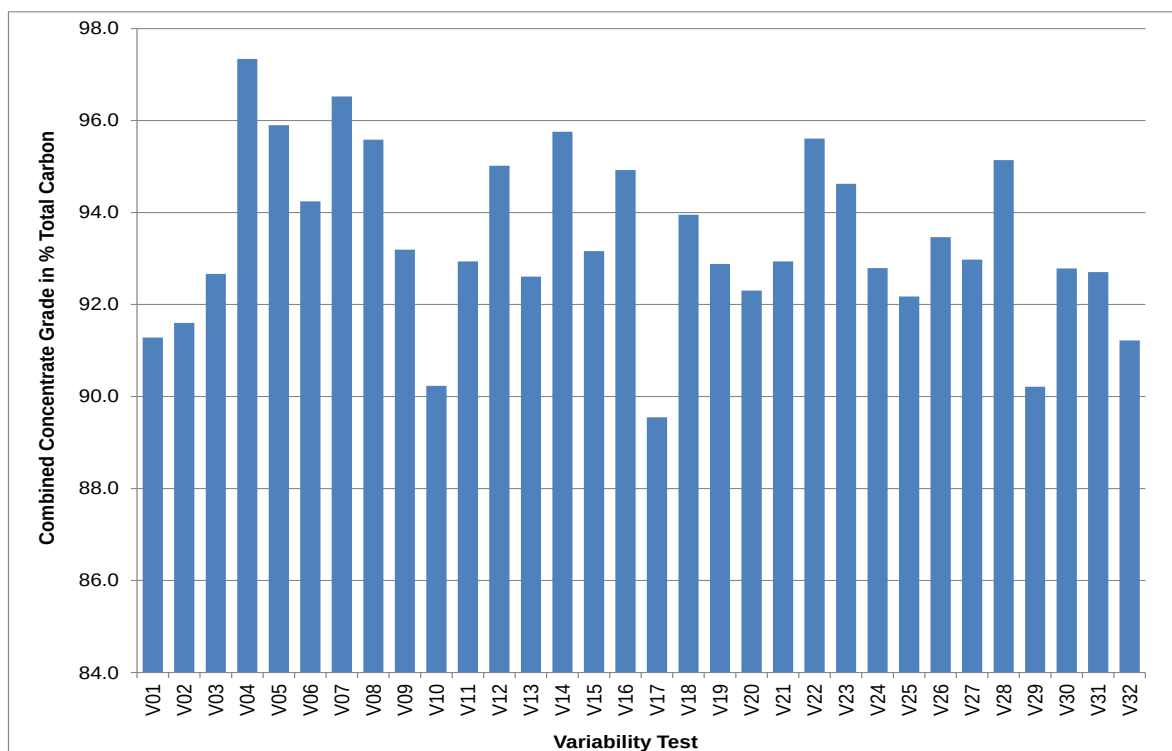


FIGURE 36: COMBINED CONCENTRATE GRADE OF TESTS ON VARIABILITY COMPOSITES (V1 TO V32)

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 37. The pit outline is demarked by the blue line. The three intersections for each drill hole represent the depth intervals 0 to 14m, 14 to 28m, and 28 to 56m starting from top to bottom.

The colour coding of each depth interval was conducted based on the legend in the figure. The results show that only seven drill hole intersections within the Molo 2015 FS 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.

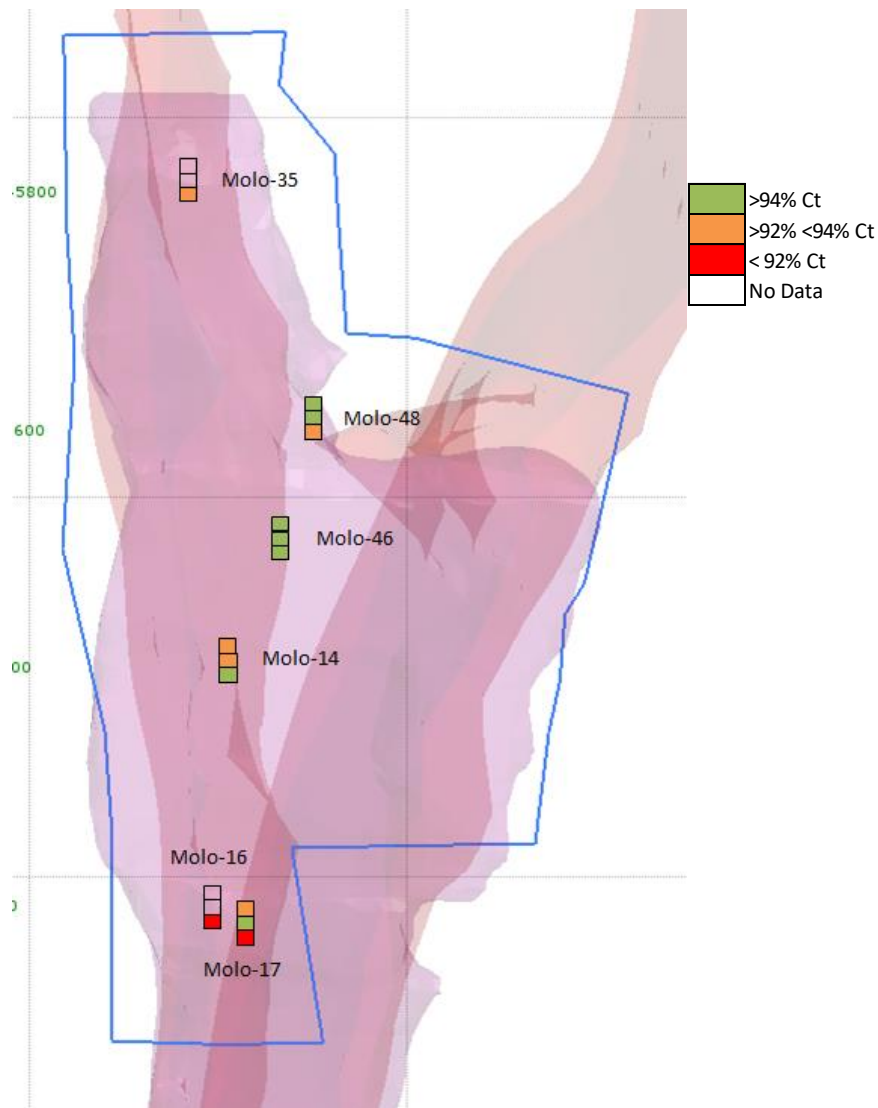


FIGURE 37: LOCATION AND PERFORMANCE OF PHASE I VARIABILITY DRILL HOLES

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 were 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 but contained impurities of non-sulphide gangue (NSG) minerals.

The NSG minerals were fine-grained between 10 and 200 microns but were interlayered 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 interlayered graphite particles is displayed in Figure 38. The non-sulphide minerals indicated by the green arrows were interlayered with graphite (red arrows).

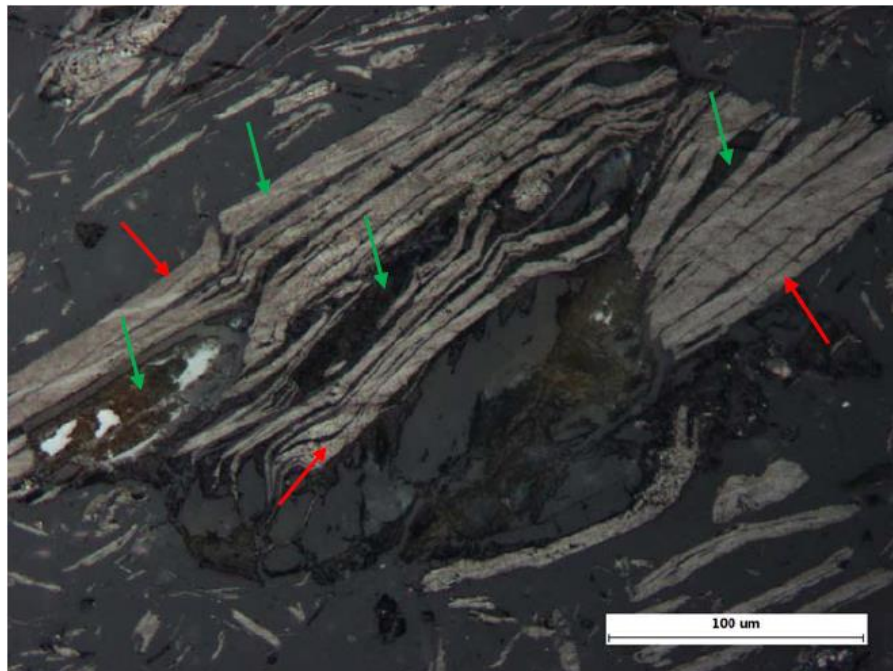


FIGURE 38: INTERCALATED GRAPHITE

13.7 VARIABILITY FLOTATION – PHASE II

A second variability flotation program was completed on the Molo deposit on a number of drill hole 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 interlayering 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 3.5 tonnes 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 seven drill holes with three 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 two 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 24.

TABLE 24: VARIABILITY COMPOSITES – PHASE II

Drill Hole	Depth of Drill Hole Interval		
	0-14m	14-28m	28-56m
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 twenty-one open circuit flotation tests.

The location and metallurgical performance of the Phase II drill hole composites are depicted Figure 39 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.

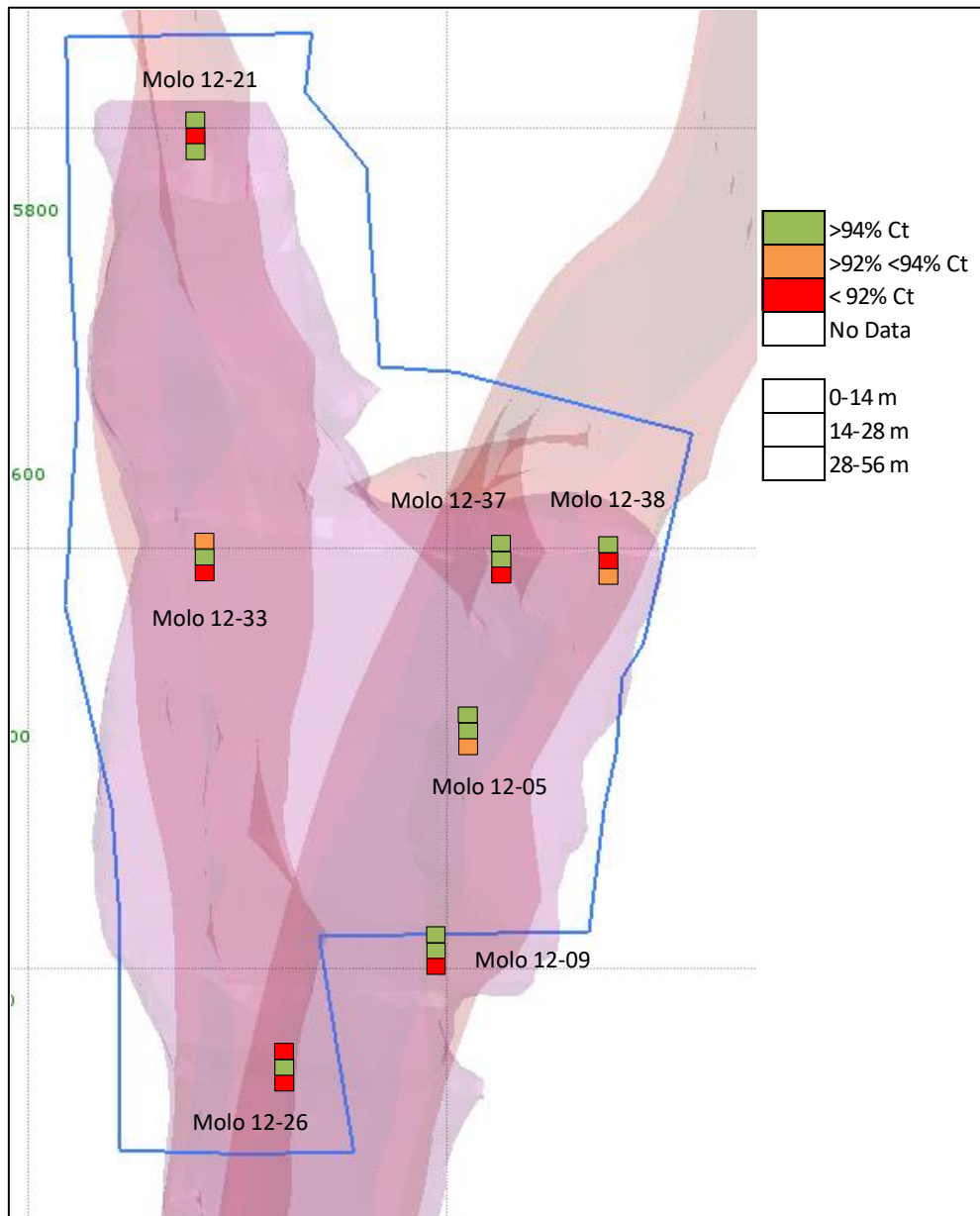


FIGURE 39: LOCATION AND PERFORMANCE OF PHASE II VARIABILITY DRILL HOLES

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 0 to 14 m, 14 to 28 m, and 28 to 56 m were maintained for the area composites i.e. a total of fifteen area composites were generated. A summary of the drill holes that were included in each area composite is provided in Table 25.

TABLE 25: 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 26. 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 26: 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 Figure 40. 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 (0 to 14 m) with a concentrate grade of 95.0% total carbon, while the worst performing composite was Area 4 (14 to 28 m) which produced a concentrate grading of only 89.4% total carbon.

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

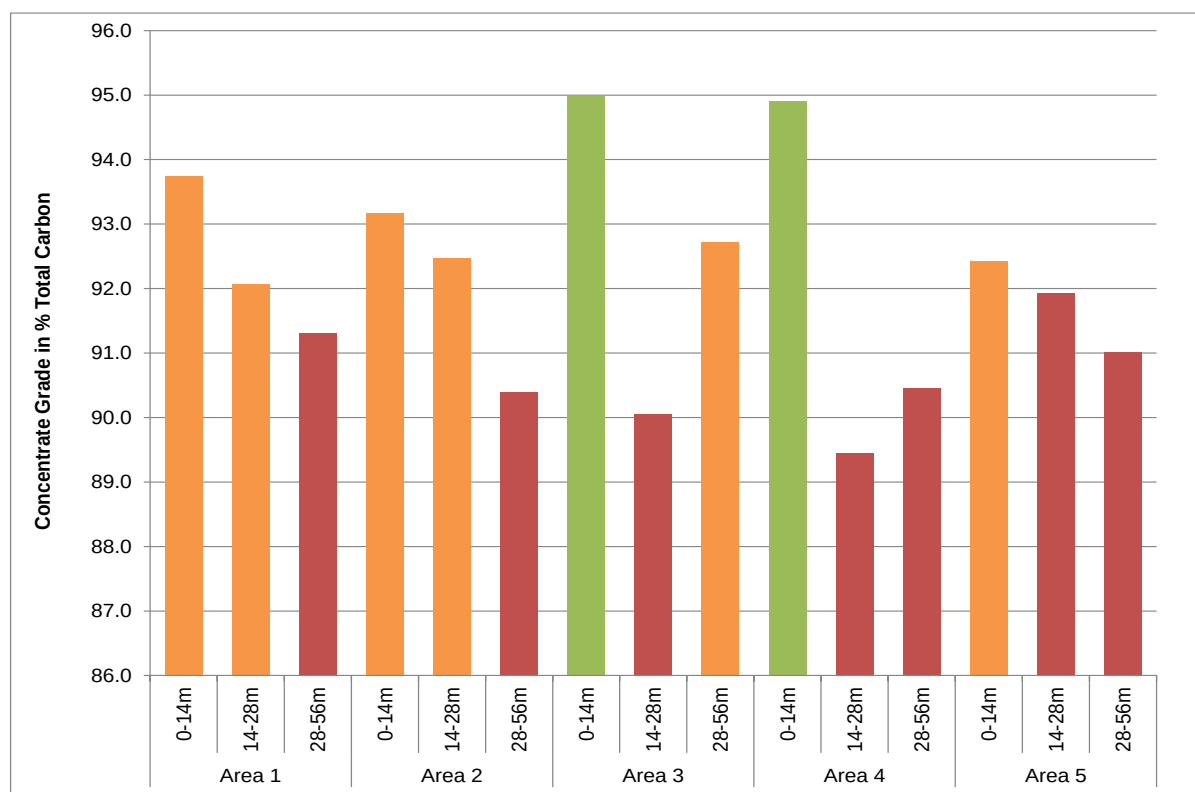


FIGURE 40: COMBINED CONCENTRATE GRADES OF TWENTY-ONE AREA COMPOSITES

The location of the five areas and the metallurgical response of the fifteen area composites are shown in Figure 41. Only the top 14 m 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.

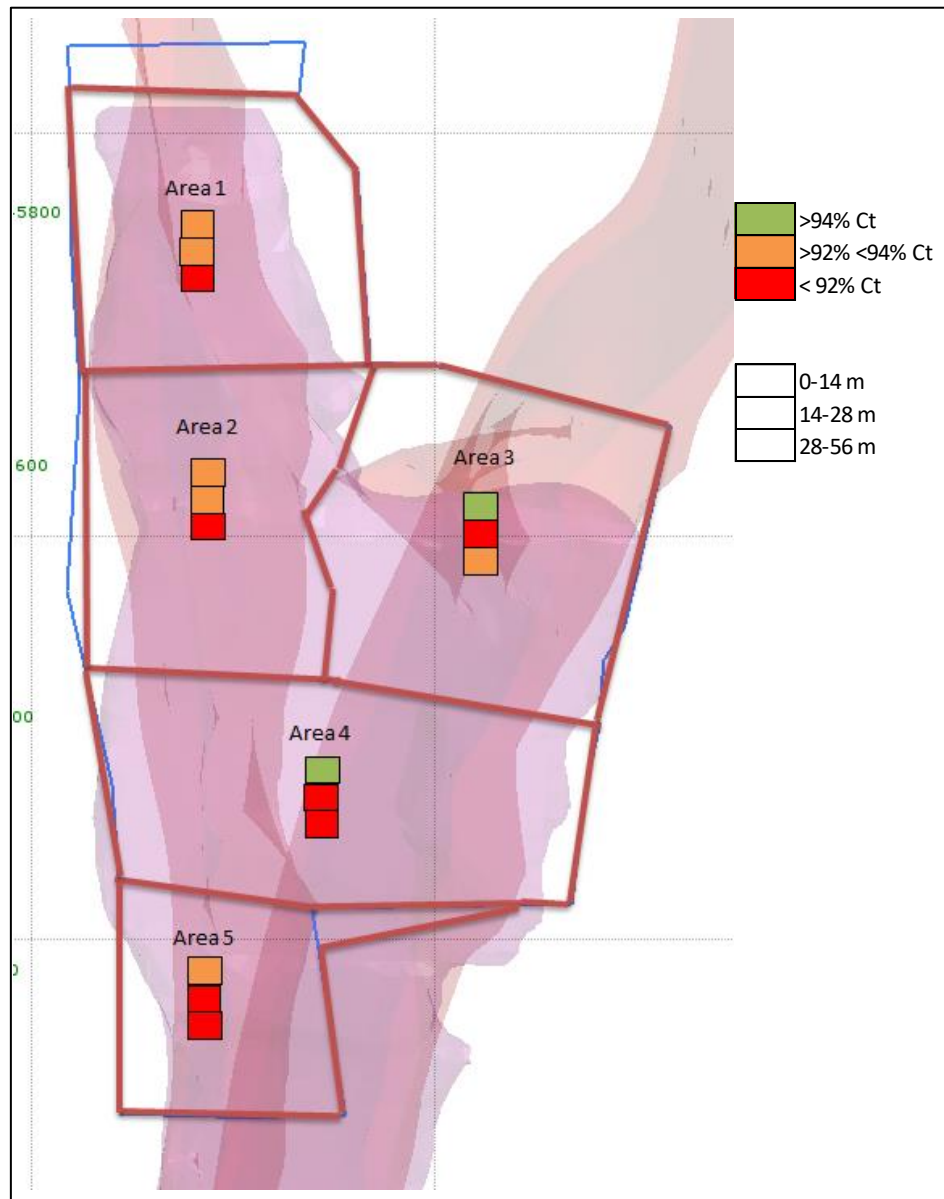


FIGURE 41: LOCATION AND PERFORMANCE OF AREA COMPOSITES

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

Since open circuit tests treat the intermediate cleaner tails as final tails, the recoveries are lower compared to closed circuit operation. In order 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 fifteen 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 intercalated 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 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 ten 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 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 Figure 42 and Figure 43 and which depict the mass recovery into the size fractions of the final concentrates of tests conducted with the stirred media mill and attrition scrubber, respectively. In order 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.

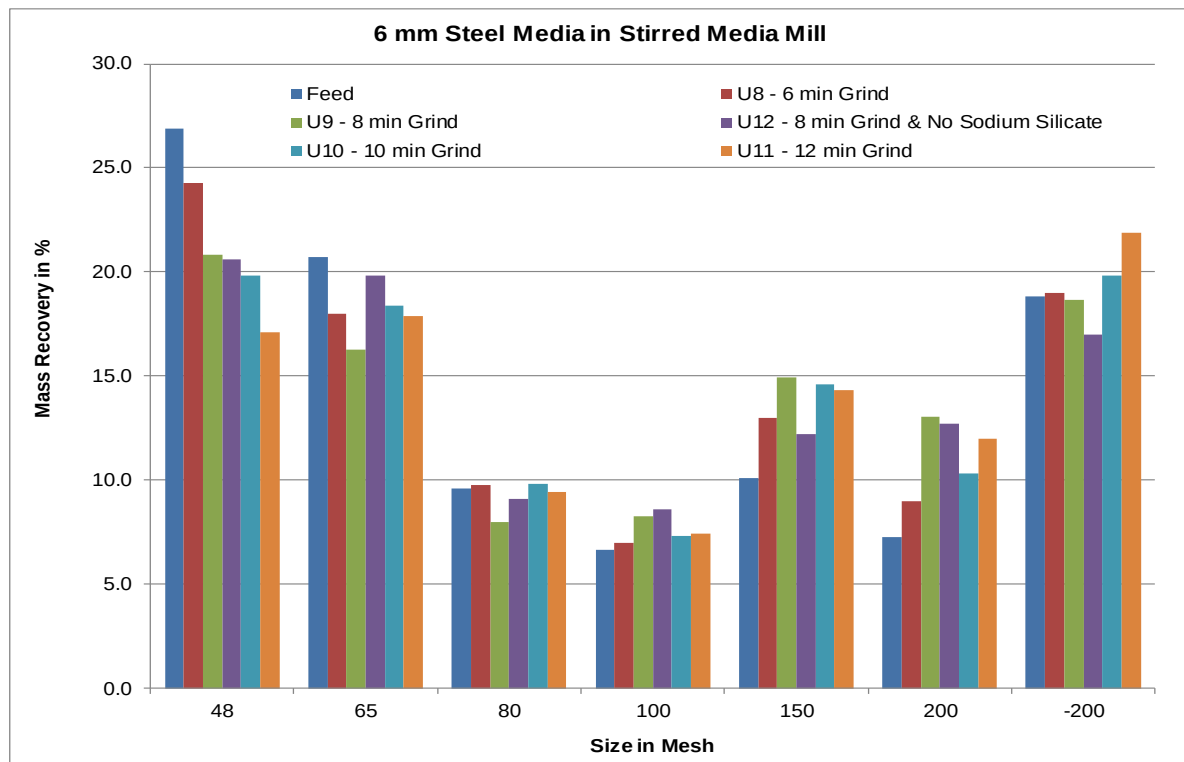


FIGURE 42: STIRRED MEDIA MILL SIZE FRACTION ANALYSIS – MASS RECOVERY

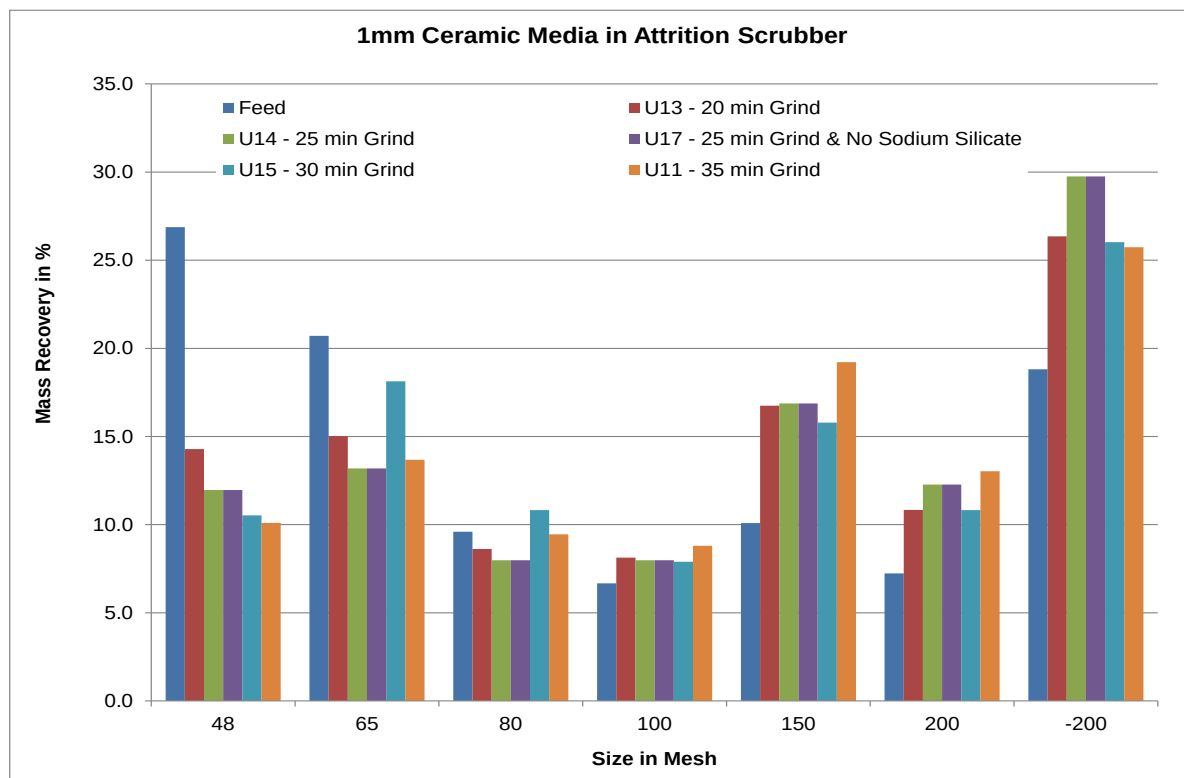


FIGURE 43: ATTRITION SCRUBBER SIZE FRACTION ANALYSIS – MASS RECOVERY

The total carbon grades into the size fractions of the combined concentrate for the five tests using the stirred media mill are presented in Figure 44. 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 five 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 per tonne 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.

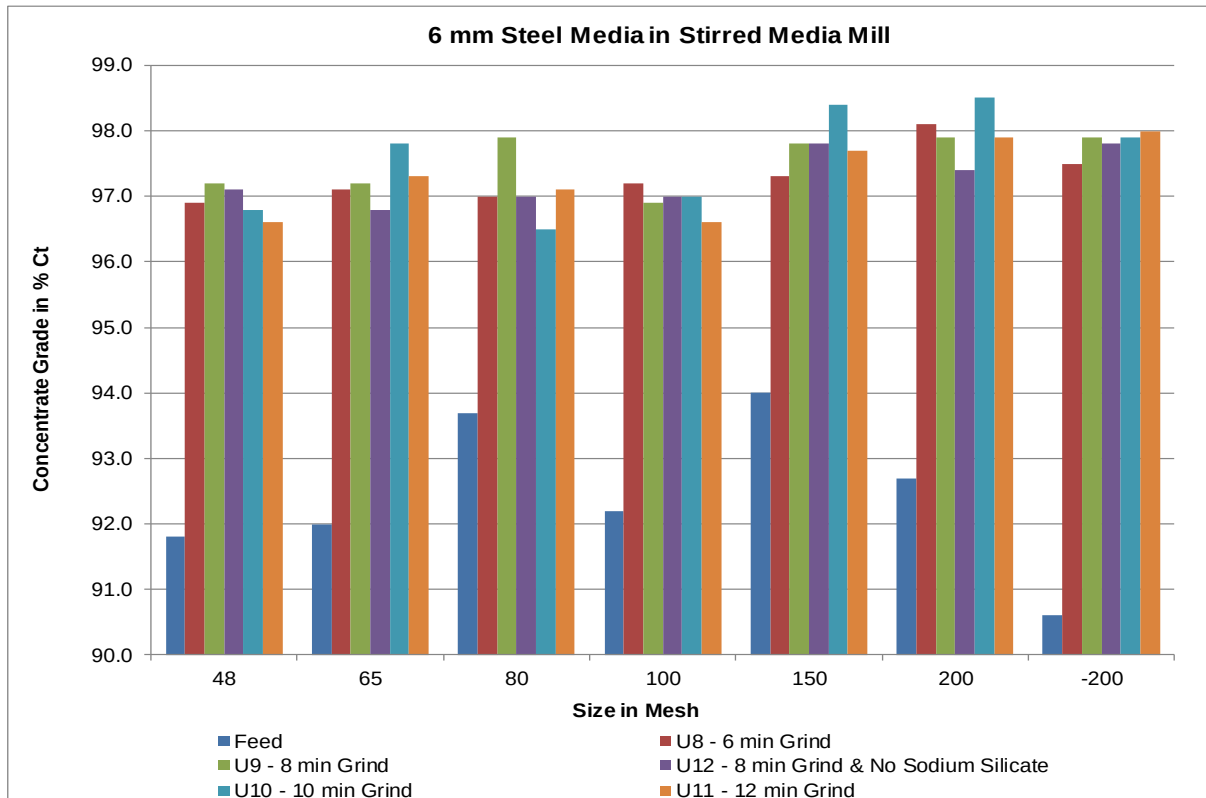


FIGURE 44: CONCENTRATE GRADES OF SIZE FRACTIONS – STIRRED MEDIA MILL

The complete Molo flowsheet with the attrition circuit is depicted in Figure 45.



Since the concentrate used in the upgrading tests was a weighted composite of all area 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 27.

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 27: 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 seven 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 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 interlayered 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 3rd cleaner concentrate is depicted in Figure 46 to illustrate the interlayering. The image shows graphite (red arrows) that is largely liberated in the sample. However, non-sulphide gangue minerals (green arrows) are mainly interlayered within graphite.

While the interlayering 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 interlayering is not required.

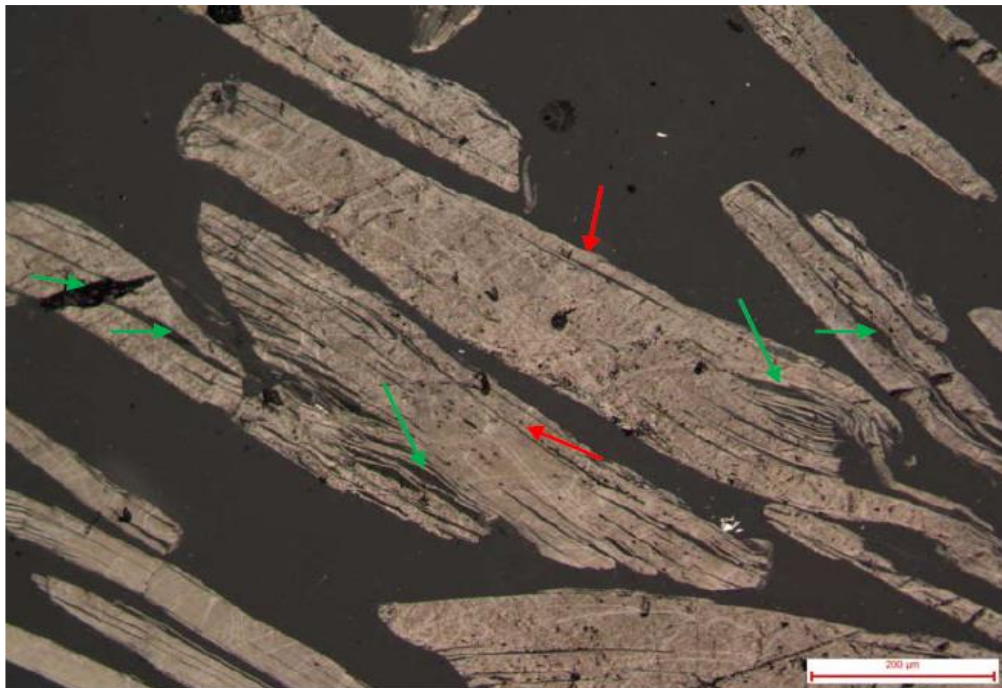


FIGURE 46: OPTICAL PHOTOMICROGRAPHS (PPRL) FROM THE U-8 SFA 3RD CLNR CONC +48 MESH

13.9 TOTAL CARBON AND GRAPHITIC CARBON ASSAY METHODS – SGS

Carbon occurs as organic carbon, carbonate carbon and graphitic carbon. The three 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 analyzed with a LECO 844, while samples higher than 30% carbon are analyzed 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 3-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 testwork 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/m²/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 a solids loading rate of 0.75 t/m²/h.

Vendor B quantified the solids loading rate for the concentrate thickener at 0.64 t/m²/h at a flocculant SNF 905 VHM dosage of 20 g/t. The solids loading rate for the graphite tails was 0.46 t/m²/h at a flocculant SNF 934 VHM dosage of 140 g/t. The concentrate and tailings thickener underflow solids concentration at 2-hour retention time were 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 – 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 to 417 kgDS/m²/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 to 218 kgDs/m²/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 kgDs/m²/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 kgDs/m²/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 32-39% 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, 80, 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 t/h per meter of screen width.

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 to 2.2 t/h per meter of screen width.

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 fairly high enough grade 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.

Since the FEED study was completed some improvements have been made. These will be incorporated into the detailed engineering ahead of construction. The changes are not significant but will improve product quality.

14 MINERAL RESOURCE ESTIMATES

This Mineral Resource Statement was produced by CCIC in September 2014. The approach and methodologies applied in this resource estimation are in accordance with international resource reporting guidelines, including NI 43-101. Leapfrog™ software was used to construct volumetric solids for the zones of mineralisation. The three-dimensional resource modeling, 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.

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 0.5 m 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 47). Elevations within the area of study range from 570 mamsl to 543 mamsl, with an average gradient of 2.0‰. The high-grade domain on the western side creates a ridge up to 2.0 m in elevation.



Z_Coord
(topo/rt/topo)

[ABSENT]
[543.365,546.999]
[546.999,550.633]
[550.633,564.268]
[564.268,567.902]
[567.902,561.536]
[561.536,565.17]
[565.17,568.804]
[568.804,572.538]

1:14000
0 40 100 180 260 340

Green Giant Graphite Deposit
August 2014

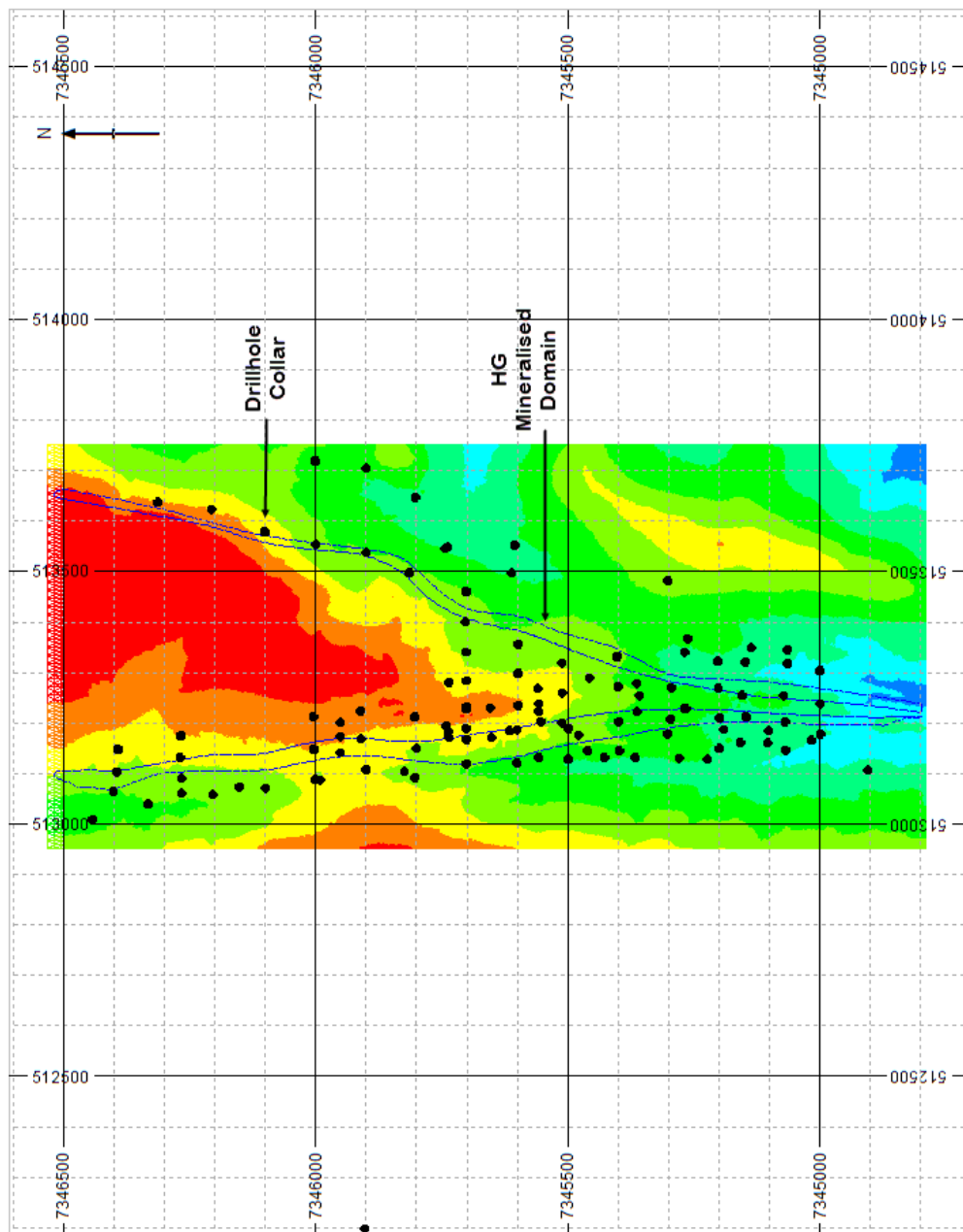


FIGURE 47: TOPOGRAPHIC CONTOURS; ELEVATIONS IN MAMSL

14.1.2 Drill Holes

This Mineral Resource estimate is based on 80 drill holes (total 11,660 m) and 35 trenches (total 8,492 m) drilled by the Company. Drill spacing varies from 100 m * 100 m in poorly informed areas to 50 m * 50 m in well informed areas. Figure 48 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 28.

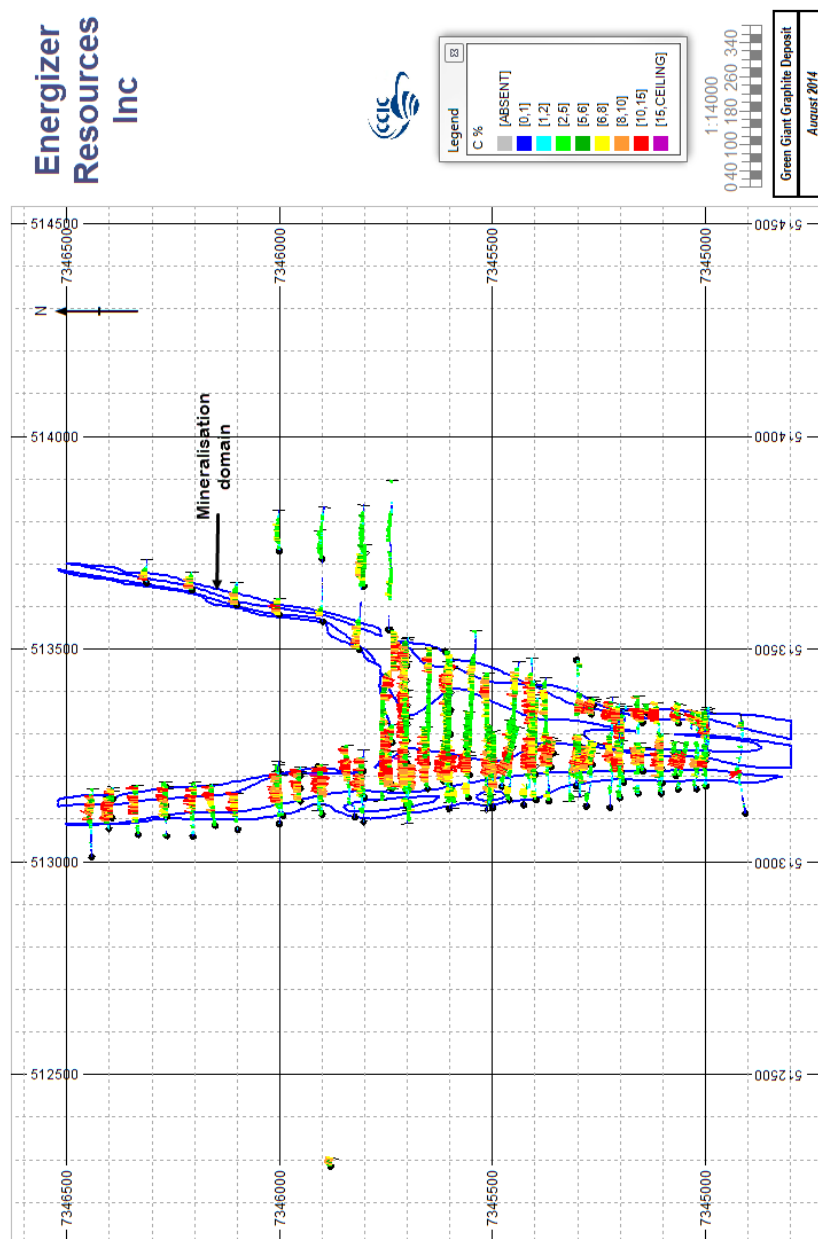


FIGURE 48: DRILL HOLE POSTING PLAN COLOURED ON C (%) GRADES.

TABLE 28: SUMMARY OF LITHOLOGICAL CODES USED.

Logging Code	Description
Gp Gn	Graphite Gneiss
Gt Gn	Garnet Gneiss
Mb	Marble
NR	No Return
SAPR	Saprolite
PEG	Pegmatite
OVB	Overburden

Spatial and statistical comparisons of % C between drill holes and trenches are presented in Figure 49 and Figure 50. 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 because the infilling drilling program focussed on upgrading the higher-grade portions of the deposit.

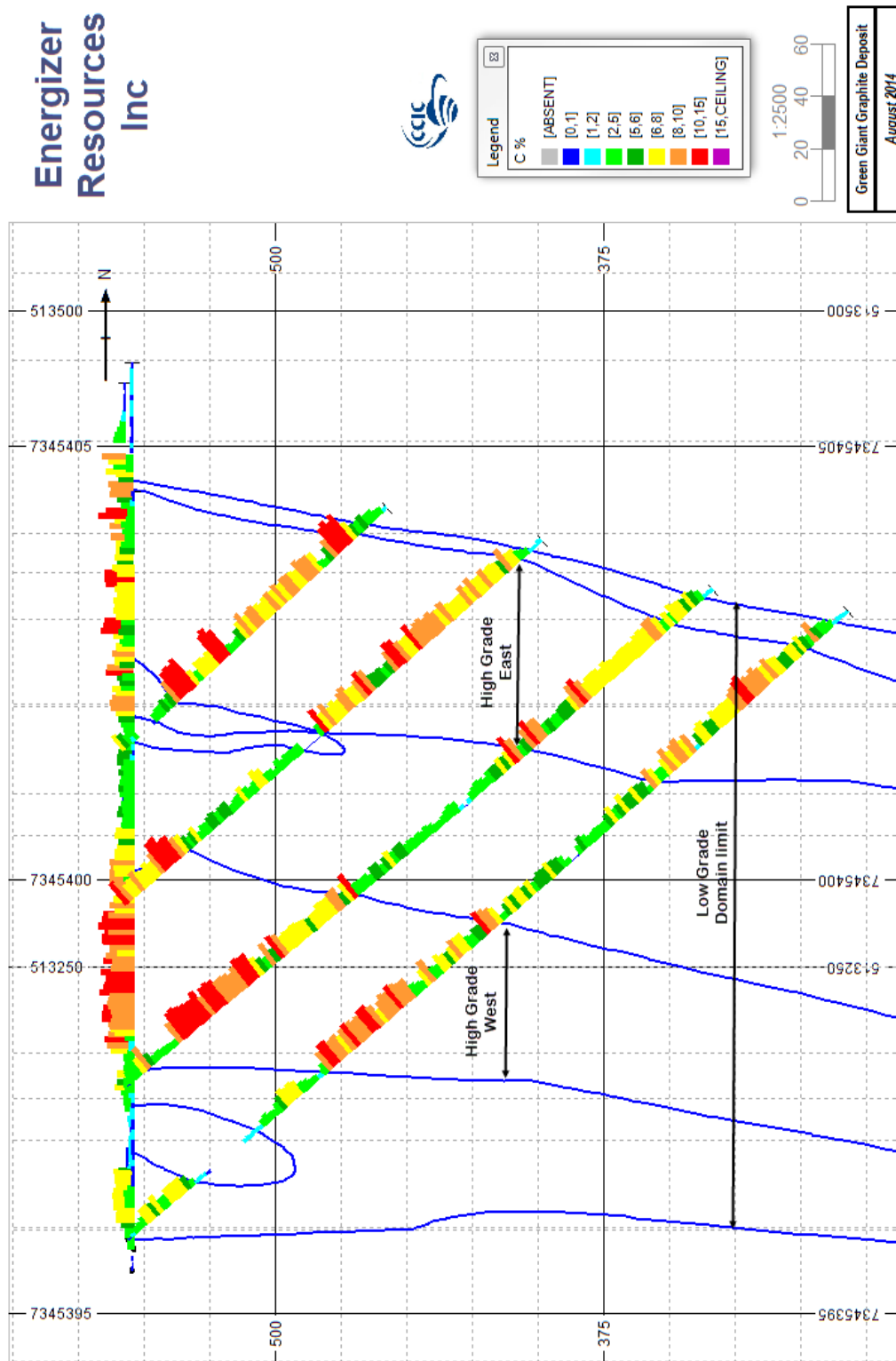


FIGURE 49: CROSS SECTION SHOWING THE GRADE DISTRIBUTION (C%) IN THE DRILL HOLES AND TRENCHES.

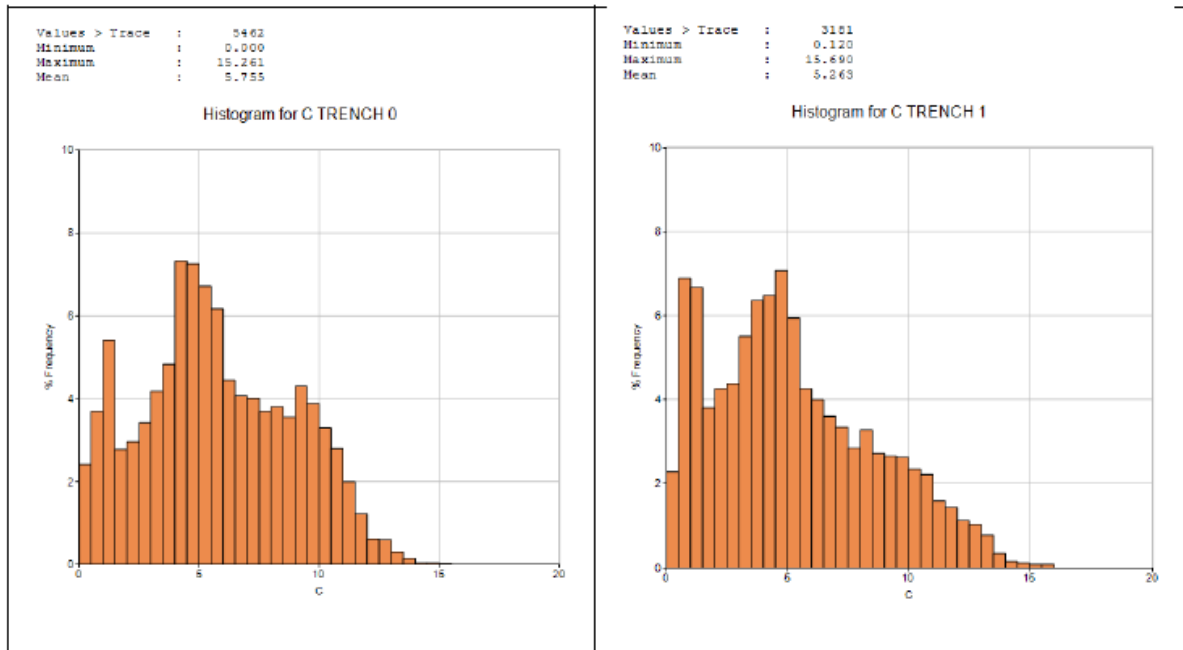


FIGURE 50: STATISTICAL COMPARISONS BETWEEN DIAMOND DRILL HOLE (DD) AND TRENCH (TH) SAMPLES.

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 Figure 51 below. The average RD for all 226 readings is 2.39 t/tm³. RD values within the GpGn range from 1.59 t/tm³ to 2.95 t/tm³, with an average of 2.35 t/tm³.

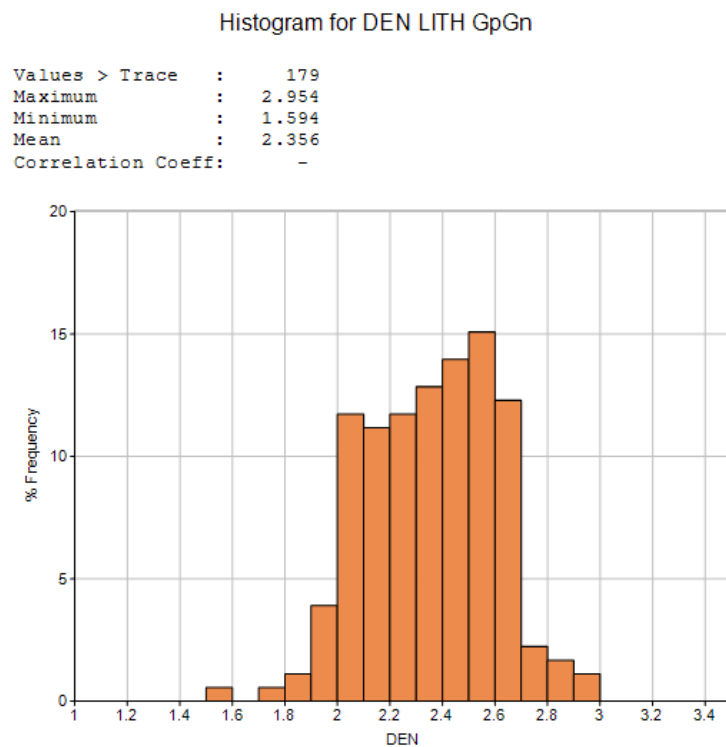


FIGURE 51: HISTOGRAM OF RELATIVE DENSITY READINGS WITHIN THE GRAPHITIC GNEISS.

14.2 GEOLOGICAL MODEL ON WHICH THE GRADE ESTIMATION IS BASED

14.2.1 Grade Domaining

The deposit is split into three domains of mineralisation, namely:

- A “barren” or “unmineralised” domain. This is a lithological boundary that separates the unmineralised Garnet Gneiss from the mineralised Graphitic Gneiss. This boundary was treated as a “hard” boundary during grade domaining and estimation.
- The mineralised 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.

Values > Trace : 6915
Minimum : 0.000
Maximum : 15.690
Mean : 6.450

Histogram for C

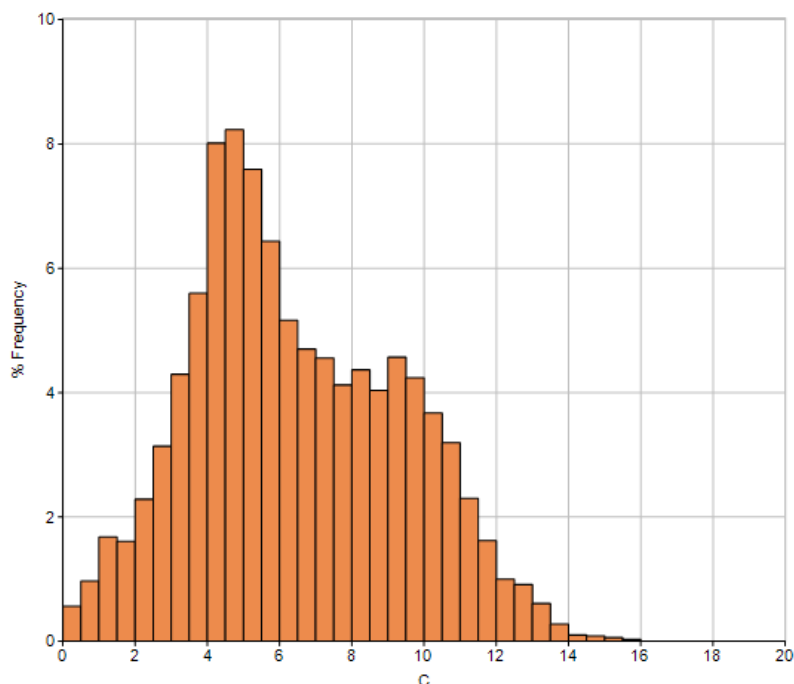


FIGURE 52: HISTOGRAM OF C DISTRIBUTION.

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 wireframe. 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; and
- 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 53 and Figure 54 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 60.0 m, where only the “high grade west” domain is developed, to more than 260.0 m in the central portions. “High grade” domains are generally 60.0 m thick, thinning to the south.

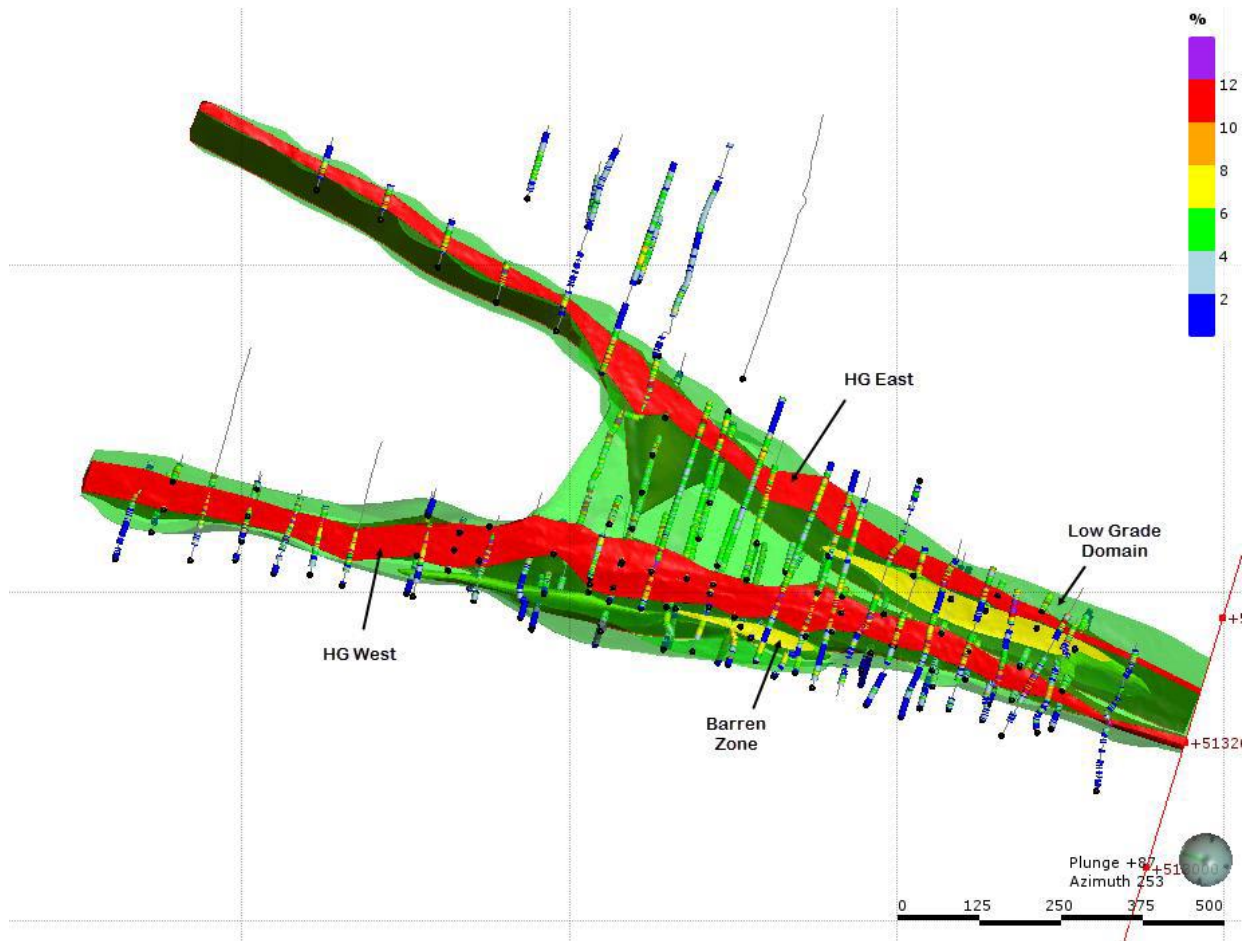


FIGURE 53: ISOMETRIC VIEW SHOWING “LOW” AND “HIGH” GRADE DOMAINS.

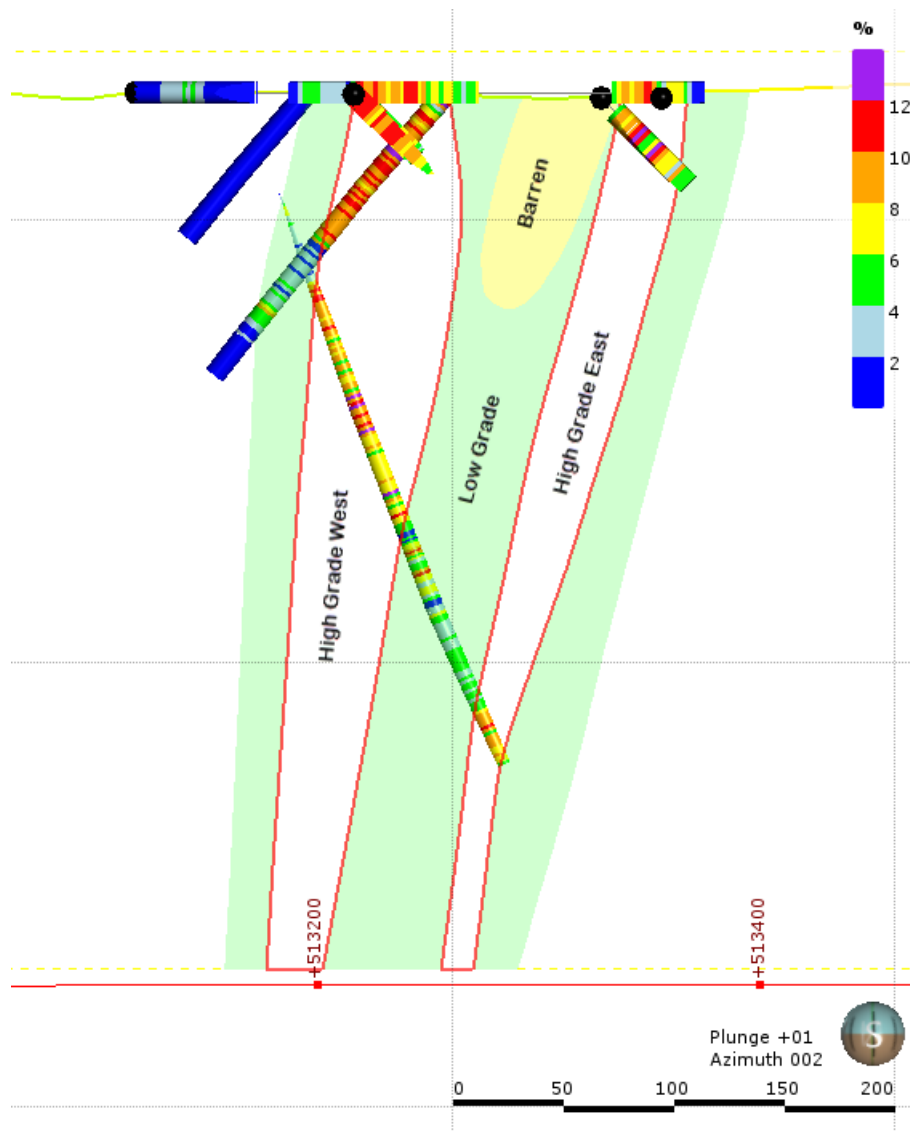


FIGURE 54: WEST-EAST SECTION SHOWING THE "LOW" AND "HIGH" GRADE DOMAINS.

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 Figure 60, the contact between the “low” and “high” grade zones was treated as a “soft” boundary. A transition of 5.0 m was used to flag samples from the “high” into the “low” grade zones. The boundary was treated as “hard” from “low” into “high” grade.

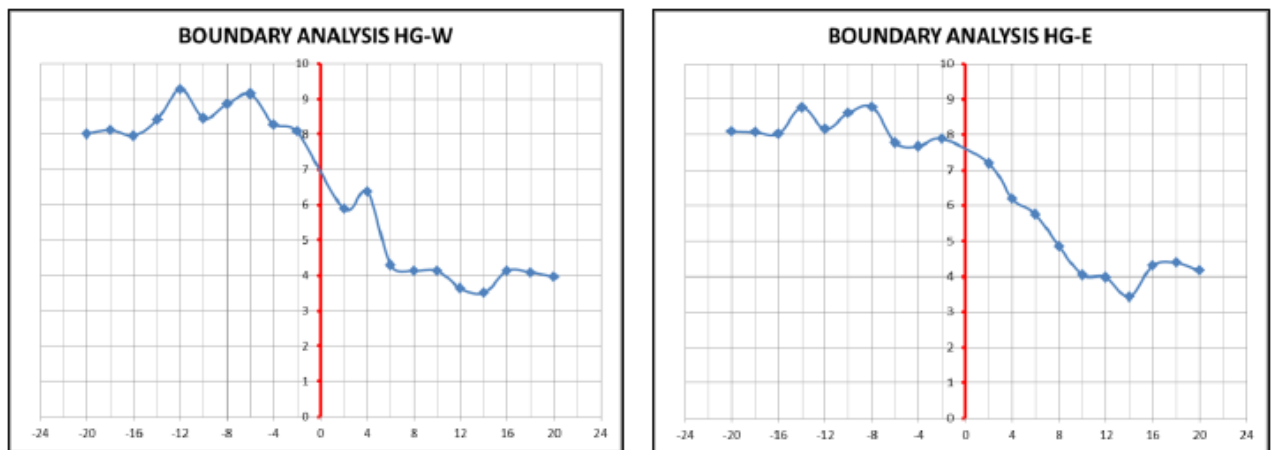


FIGURE 55: “HIGH” AND “LOW” GRADE BOUNDARY ANALYSIS.

Zonal flagging in Datamine uses a field called KZONE to distinguish the different grade domains during geostatistical analysis and estimation (Figure 56).

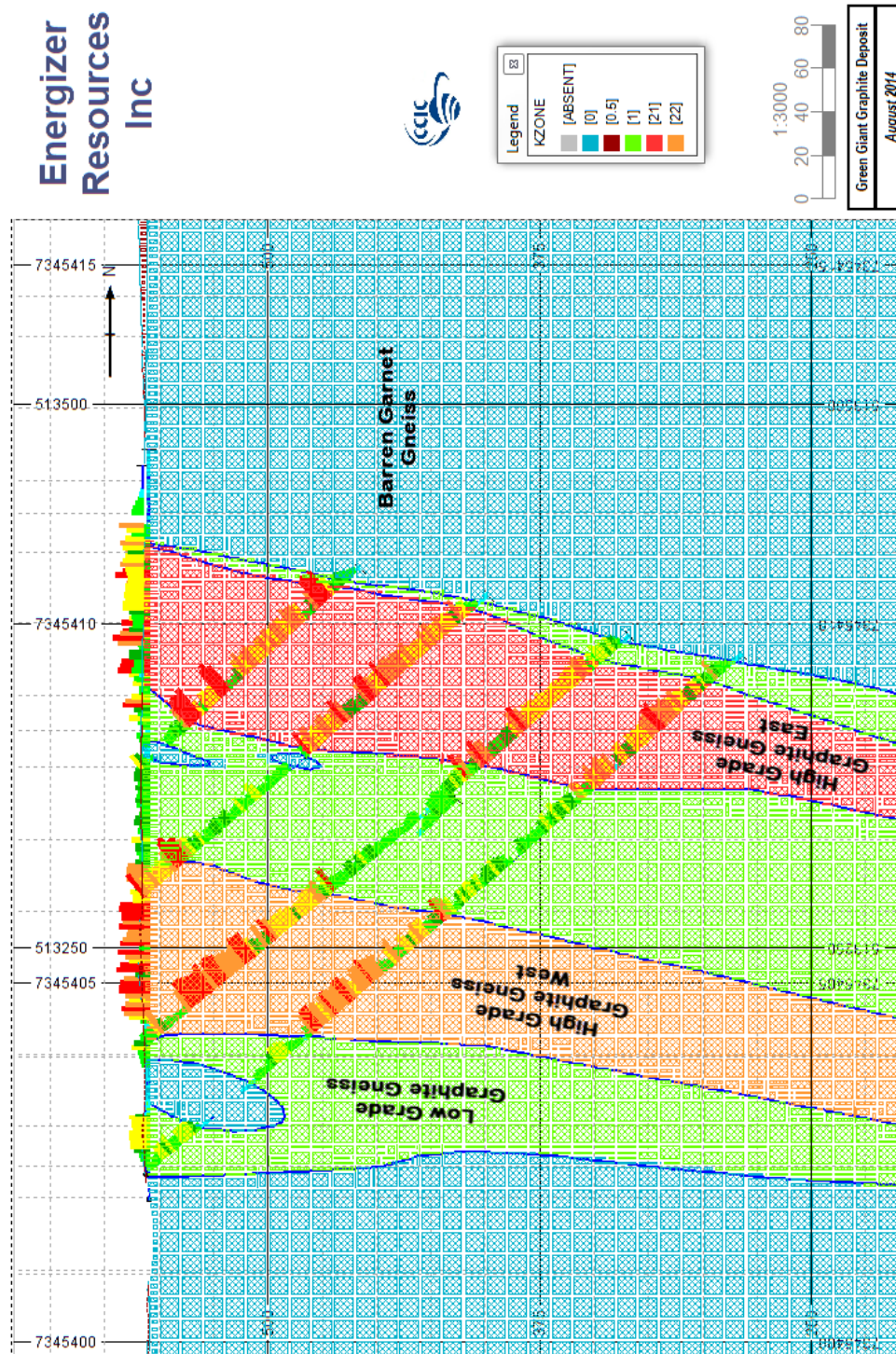


FIGURE 56: CROSS-SECTION SHOWING THE KZONE FLAGGING IN THE BLOCK MODEL.

A summary of the KZONE flagging undertaken in Datamine is provided below in Table 29.

TABLE 29: 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 0.5 m, 1.0 m, 1.5 m or 2.0 m, and hence a composite length of 2.0 m was used to include all samples (Figure 57). Compositing used the KZONE to ensure that samples were composited within the different domains. Any samples less than 0.5 m after compositing were excluded from the geostatistical analysis and estimations.

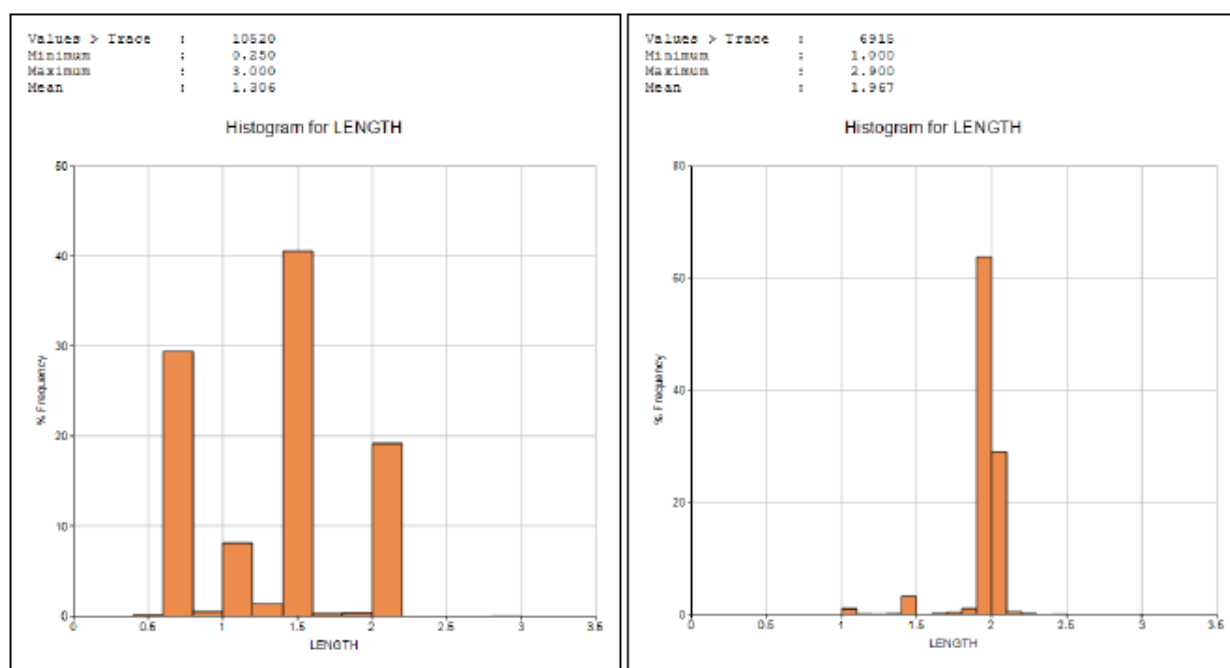


FIGURE 57: HISTOGRAM OF SAMPLE LENGTH PRIOR TO AND AFTER COMPOSITING.

14.3.1 Compositing Statistics

The sampling protocol for core intervals deemed to be barren with respect to graphite mineralisation is that they were not to be 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 30 below.

TABLE 30: STATISTICAL SUMMARY OF C% PER KZONE.

KZONE	0	1	21	22
FIELD	C	C	C	C
NUMTRACE	1730	3274	1282	2359
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	0.85	0.44	0.32	0.32

* STANDDEV = Standard Deviation

* CoV = Coefficient 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 Figure 58, Figure 59 and Figure 60 below. The sample distributions for both the “high grade” domains exhibit very similar statistical characteristics.

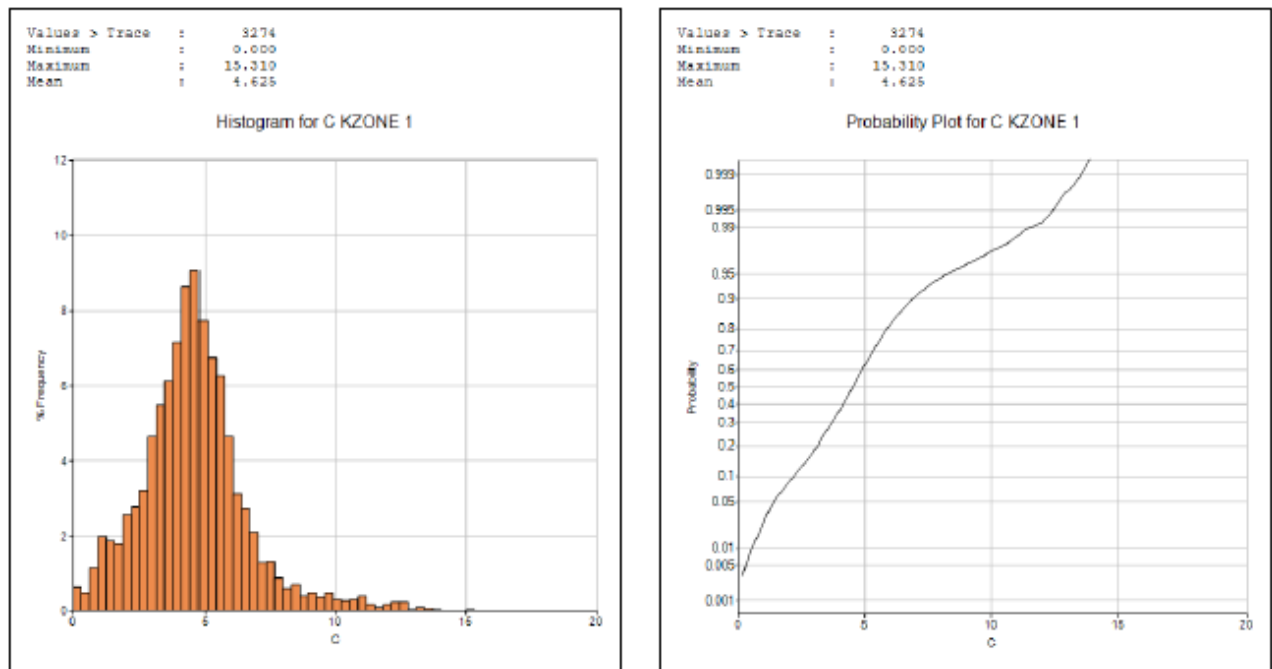


FIGURE 58: SAMPLE DISTRIBUTIONS FOR THE “LOW GRADE” DOMAIN.

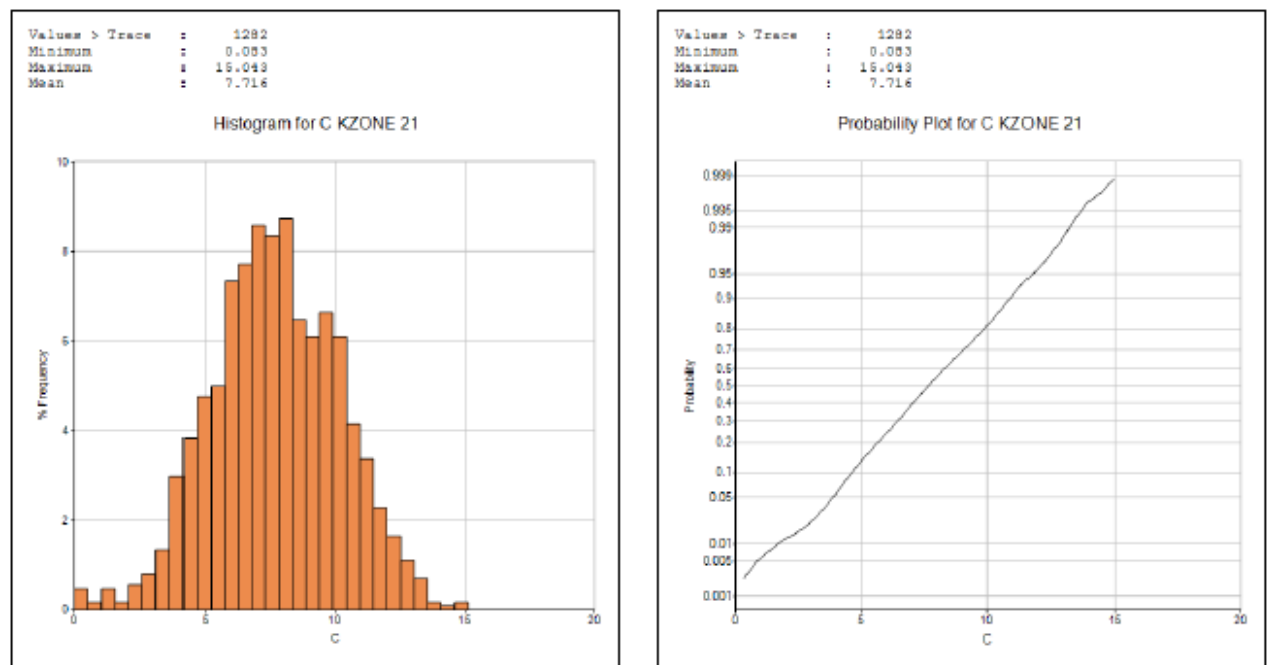


FIGURE 59: SAMPLE DISTRIBUTIONS FOR THE “HIGH GRADE EAST” DOMAIN.

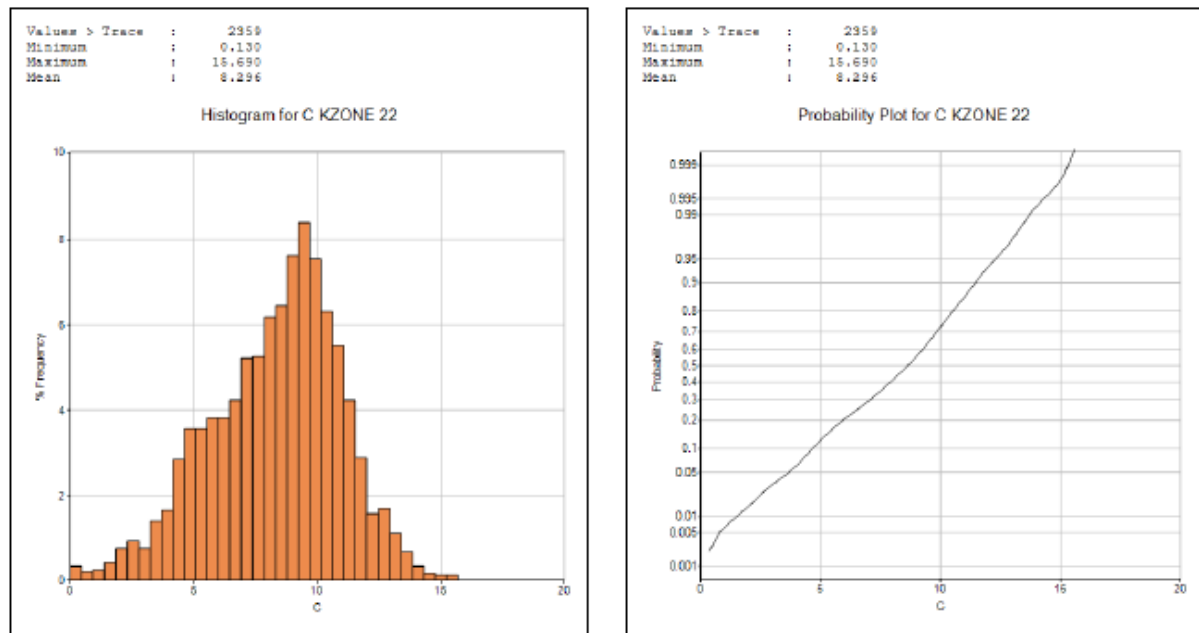


FIGURE 60: SAMPLE DISTRIBUTIONS FOR THE “HIGH GRADE WEST” DOMAIN.

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 Figure 61, Figure 62 and Figure 63. The nugget for the ‘low grade’ domain is ~10%, with more than 90% of the spatial variance occurring within the first 120 m. The “high grade” east domain has a nugget of ~10% and a range that extends to 80.0 m. The “High grade” west domain has a nugget of ~10% and a range of 55.0 m. 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.

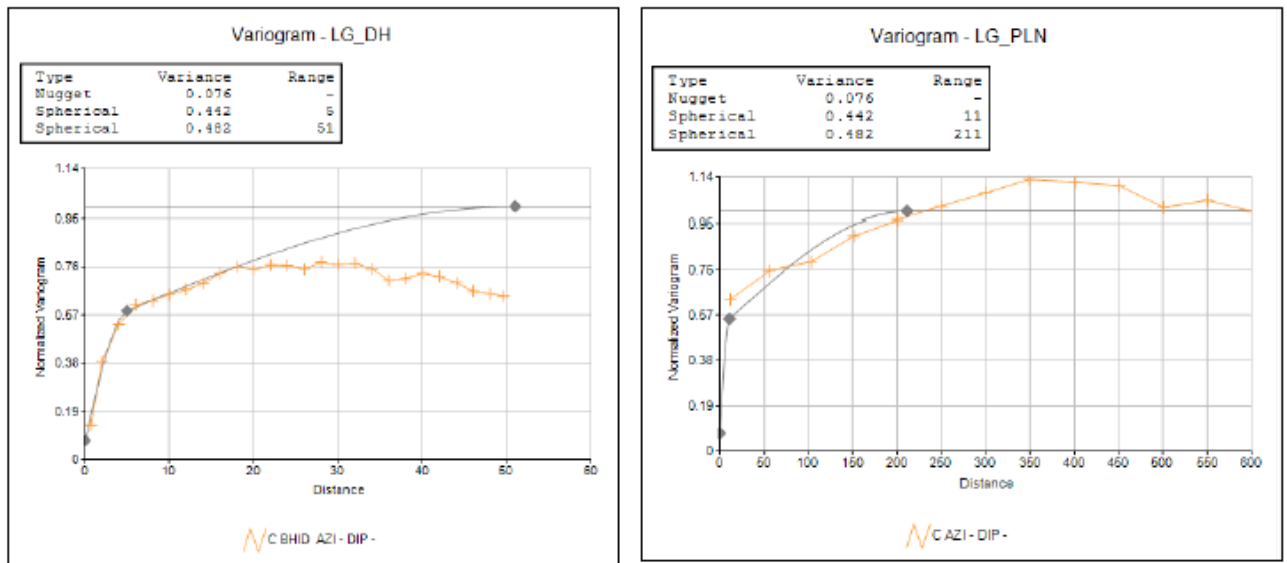


FIGURE 61: VARIOGRAM MODEL FOR “LOW GRADE” DOMAIN – DOWN HOLE AND ISOTROPIC.

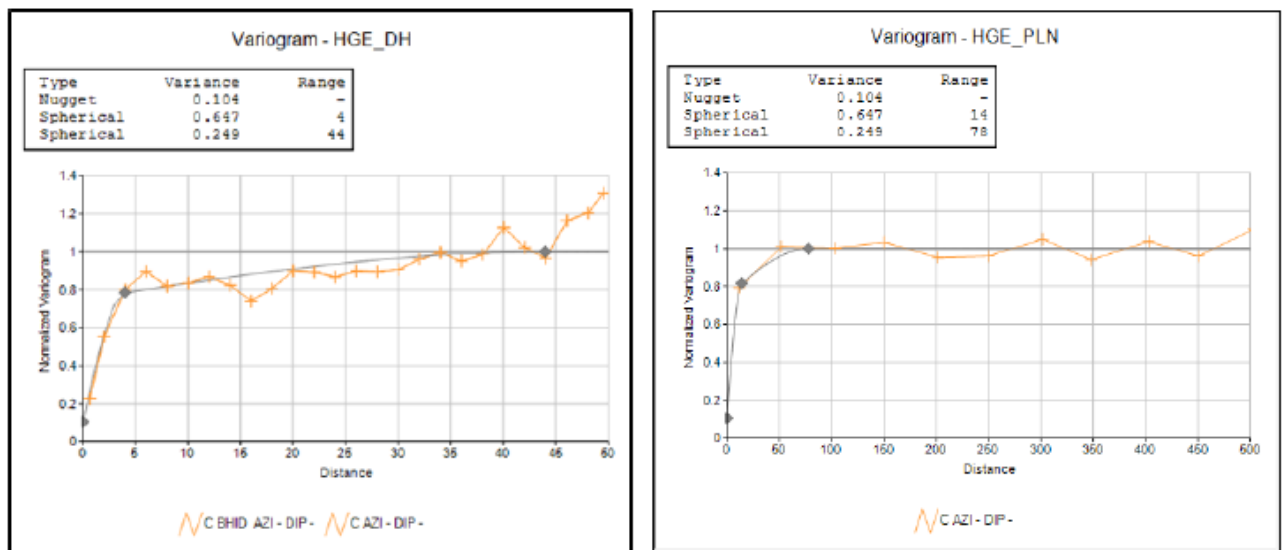


FIGURE 62: VARIOGRAM MODEL FOR “HIGH GRADE EAST” DOMAIN – DOWN HOLE AND ISOTROPIC.

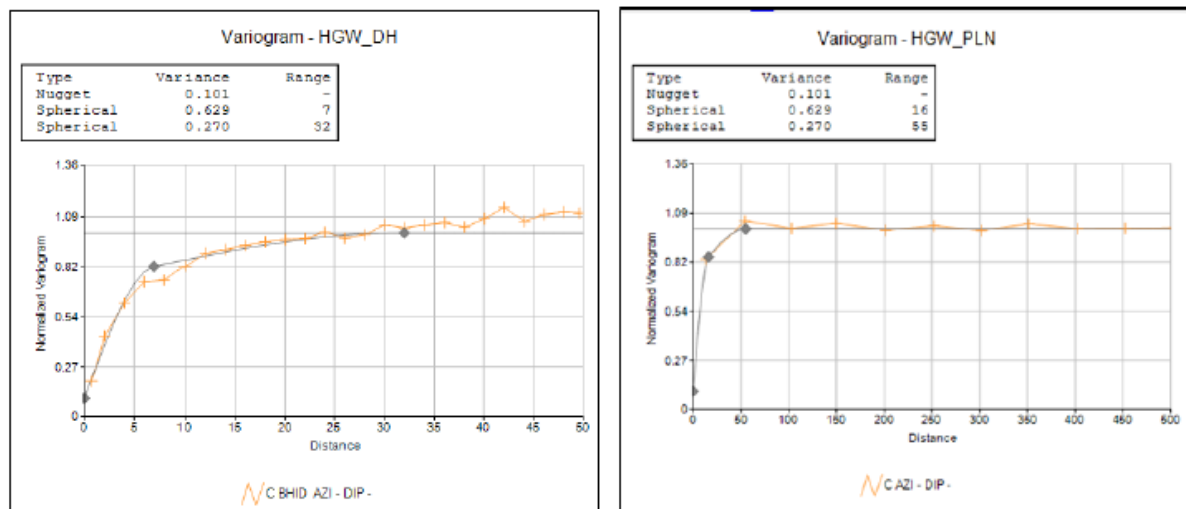


FIGURE 63: VARIOGRAM MODEL FOR “HIGH GRADE WEST” DOMAIN – DOWN HOLE AND ISOTROPIC.

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; and
- Validation of model estimates against samples.

A summary of the top capping values that were applied are presented in Table 31 below. All samples that were greater than the top capping value were re-set to the top capping value.

TABLE 31: 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) were 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 64 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.

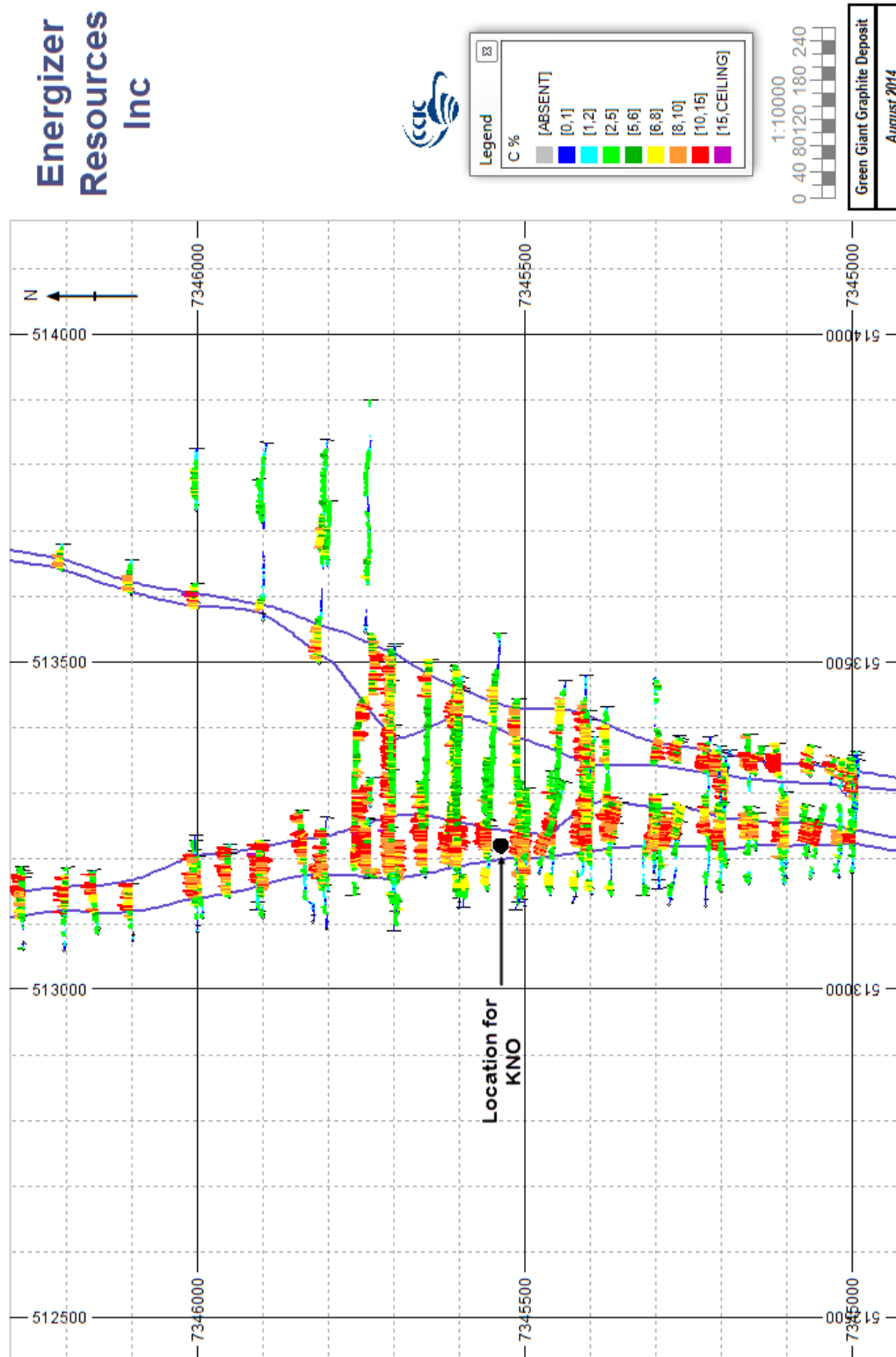


FIGURE 64: PLAN SHOWING THE LOCATION FOR THE KRIGE OPTIMIZATIONS.

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 10 m*10 m*10 m 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 5 m and then incrementally increased for each and 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 ("PCNEGWT") 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. Figure 65 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 40 m*10 m*10 m in the X, Y and Z directions respectively.

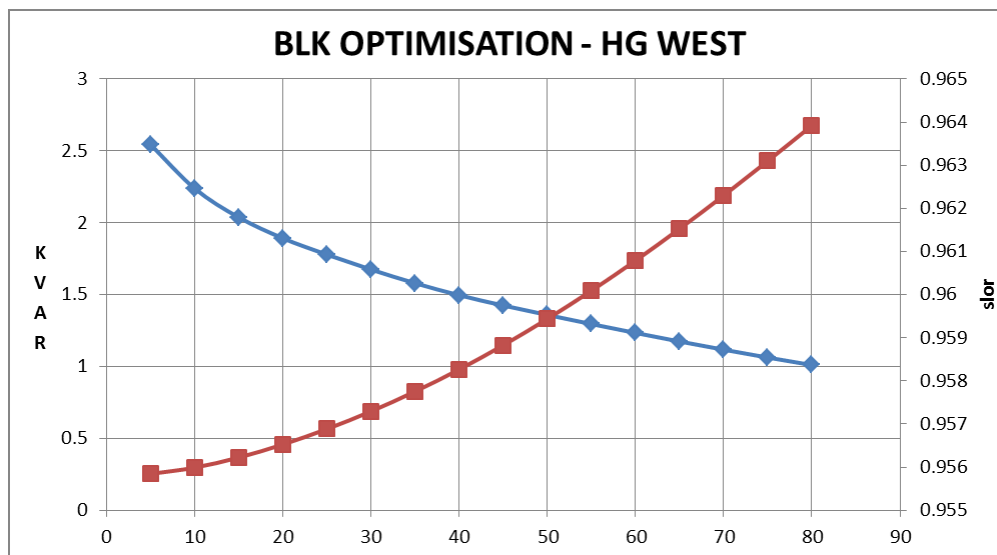


FIGURE 65: PARENT CELL SIZE OPTIMIZATIONS.

During search optimisations, the parent cell size was set to 10 m*40 m*10 m and, as described above, search distance in the X direction was incrementally increased. The results of the search optimisation are plotted in Figure 66, 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 70 m*40 m*10 m 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.

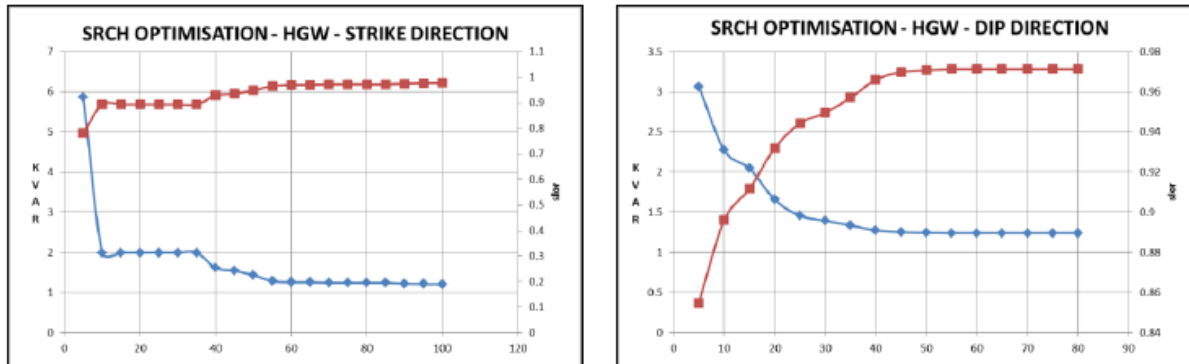


FIGURE 66: SEARCH DISTANCE OPTIMIZATIONS

14.6.3 Model Construction and Parameters

A block model was constructed using a parent cell size of 10 m*40 m*10 m 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 67. "Waste" domains have been assigned and estimated for indicative purposes only and have not been reported in the mineral resource statement.

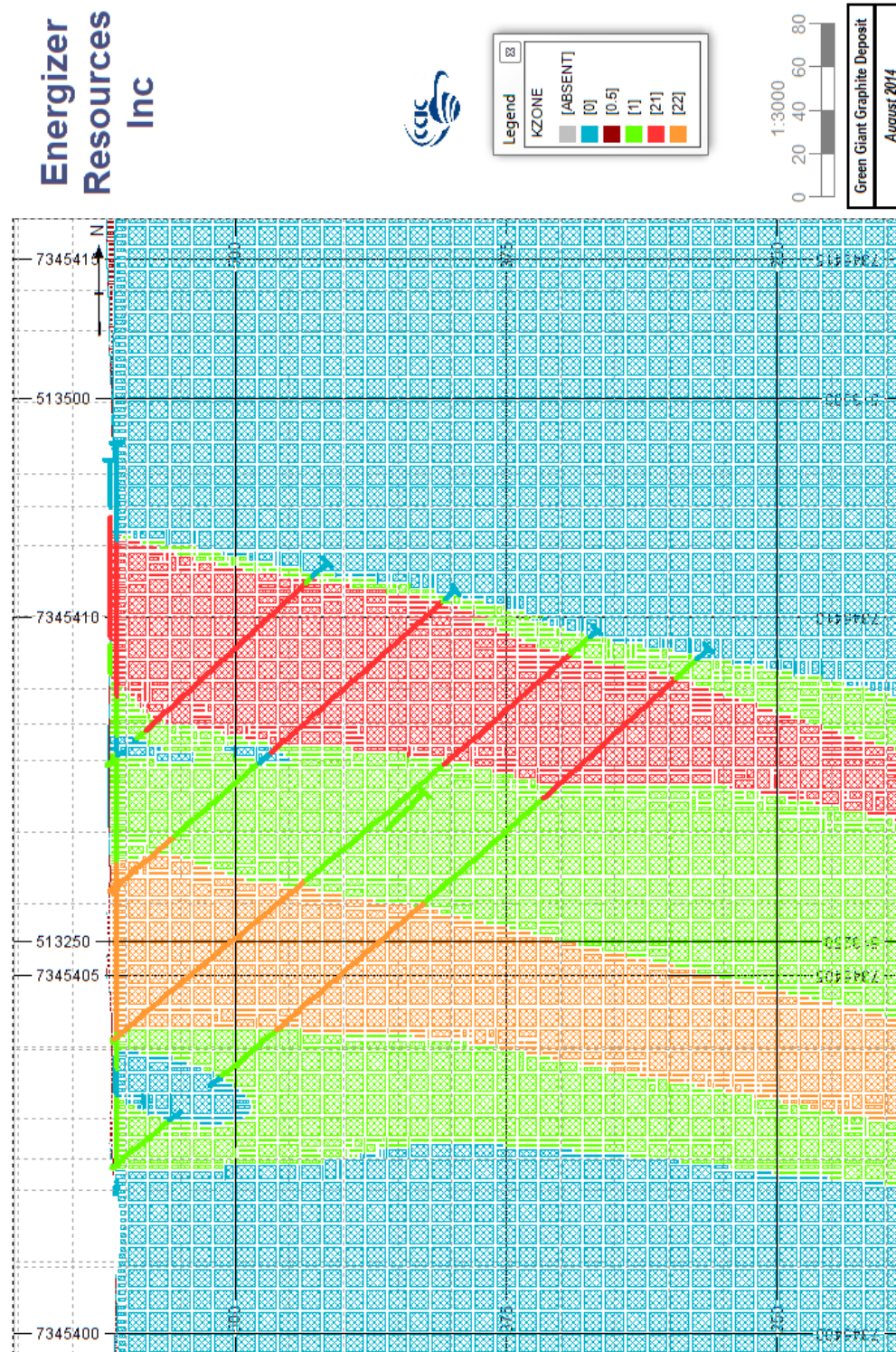


FIGURE 67: SECTION SHOWING BLOCK MODEL CODING

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; and
- Trends (or swath analysis checking) to ensure that the regional grade trends from the drill holes are present in the model. In order 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 32 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 32: SUMMARY OF TOP CAPPING VALUES.

Sample Composites				Model Estimates		
KZONE	LG	HG-E	HG-W	LG	HG-E	HG-W
FIELD	C	C	C	C	C	C
NSAMPLES	3312	1269	2359	56940	21348	16534
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*40m*10m blocks in the X, Y and Z direction respectively, and to select samples within each block, and compare 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 Figure 68, Figure 69 and Figure 70. The mean of the estimates are smoother and less variable than that of the samples, while preserving the grade trends of the samples.

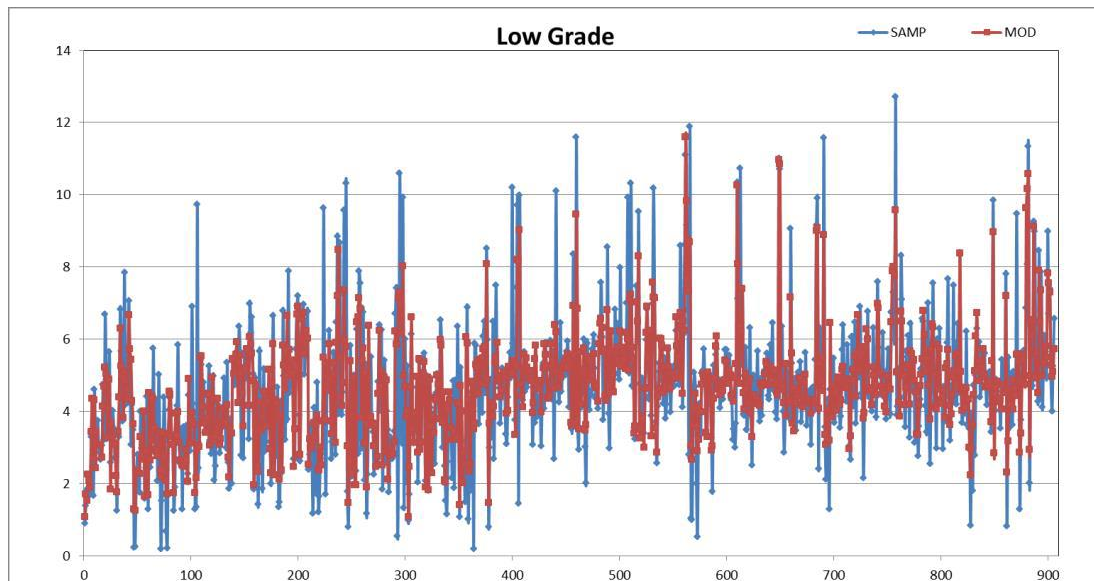


FIGURE 68: TREND ANALYSIS PLOT FOR %C, "LOW" GRADE DOMAIN.

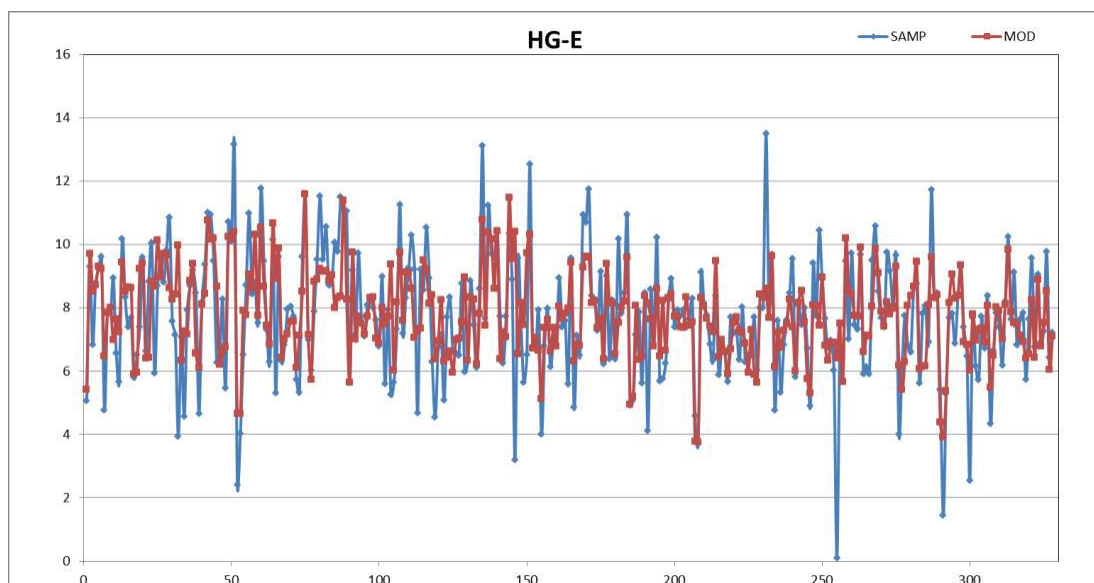


FIGURE 69: TREND ANALYSIS PLOT FOR %C, "HIGH GRADE EAST" DOMAIN.

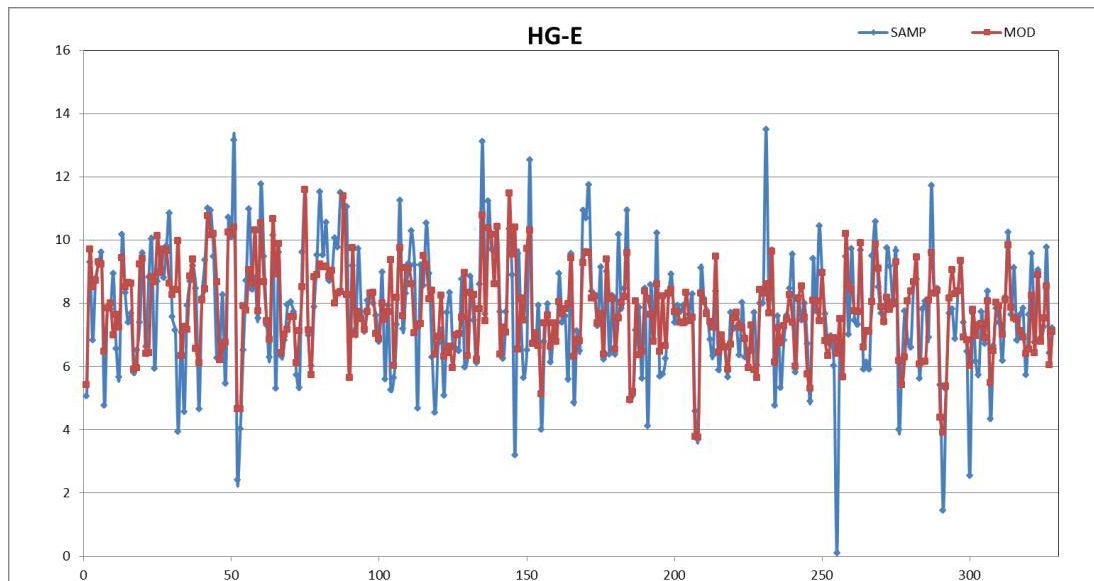


FIGURE 70: TREND ANALYSIS PLOT FOR %C, "HIGH GRADE WEST" DOMAIN.

14.8 GRADE DISTRIBUTION PLOTS

Figure 71 is a section illustrating the distribution of the block model superimposed on the drill holes. There is a good correlation between the grades in the drill holes and that of the block model.

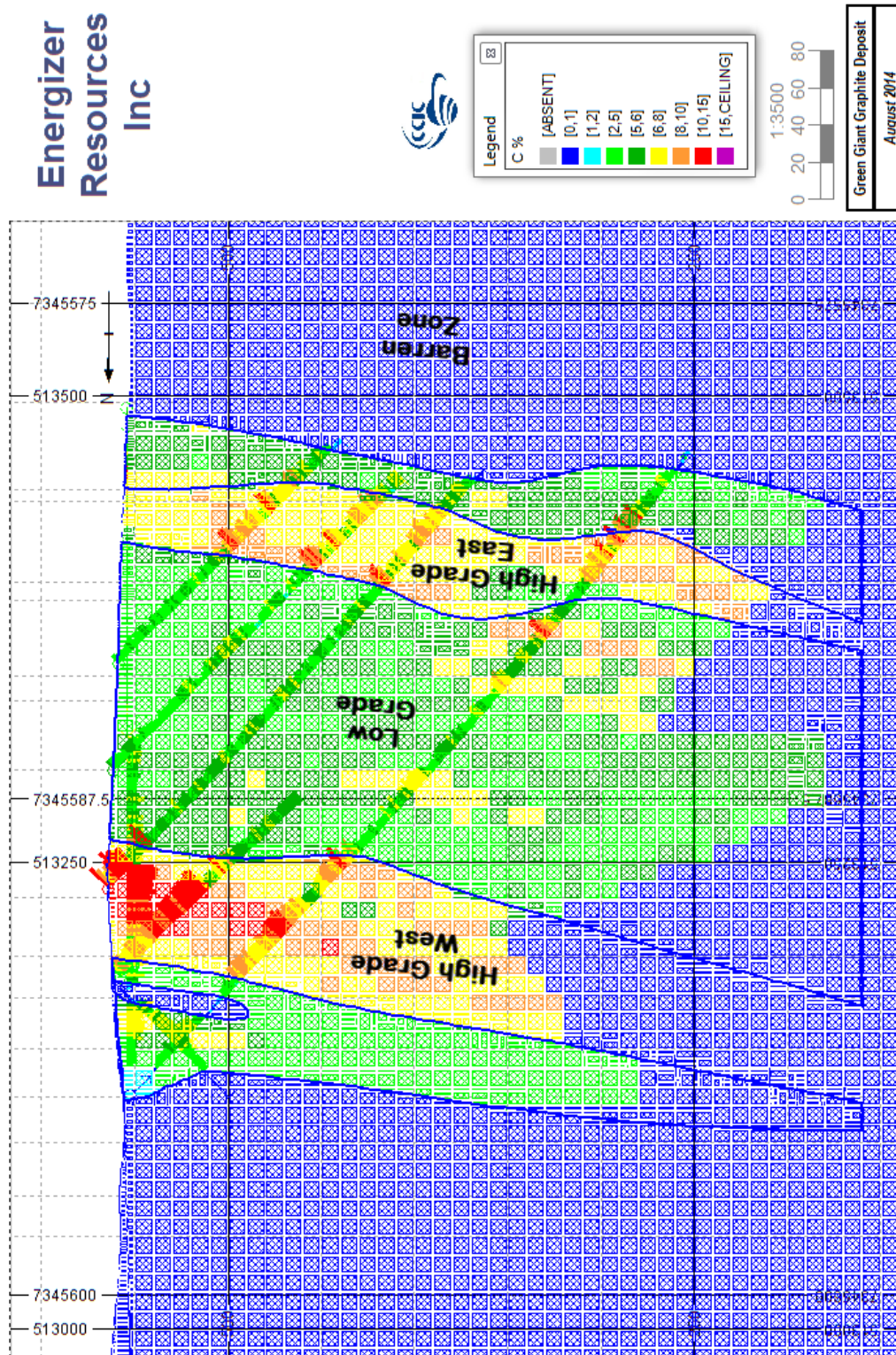


FIGURE 71: SECTION SHOWING C (%) GRADE DISTRIBUTION IN THE MODEL

The grade tonnage curves for the “low” and “high” grade domains are presented in Figure 72 and Figure 73. 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.

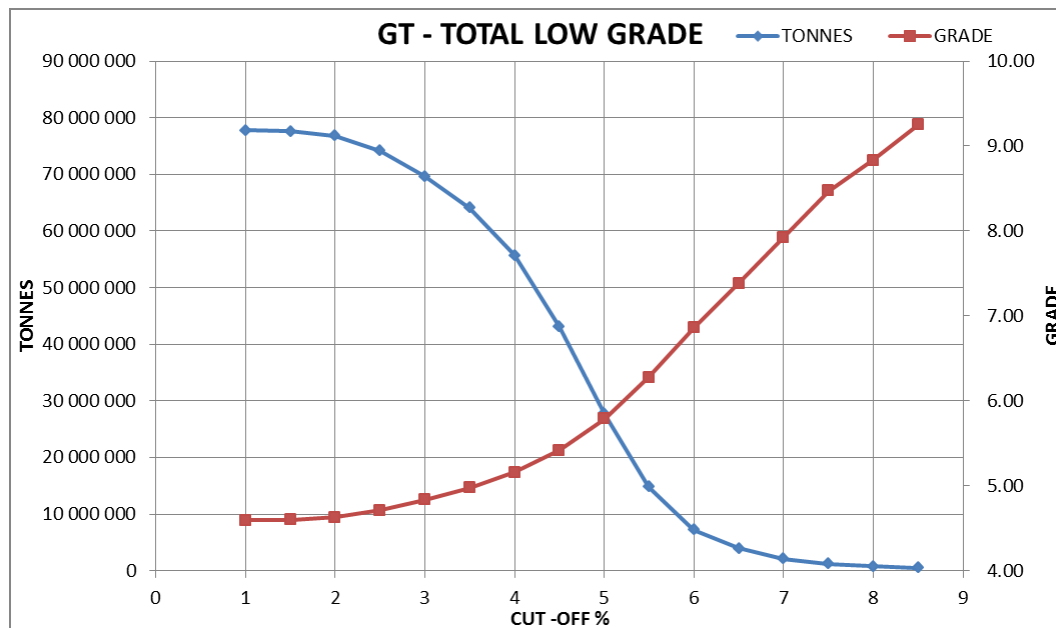


FIGURE 72: GRADE TONNAGE CURVE FOR THE “LOW” GRADE DOMAIN.

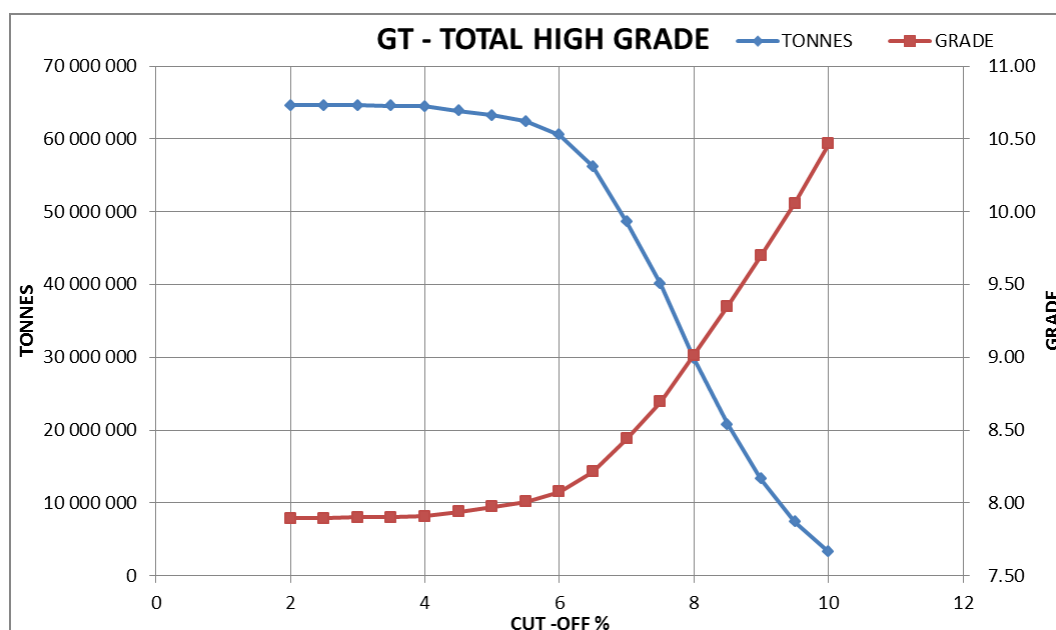


FIGURE 73: GRADE TONNAGE CURVE FOR THE “HIGH” GRADE DOMAINS.

14.9 RESOURCE CLASSIFICATION

The definition of a mineral resource according to the Canadian Institute of Mining ("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, estimated or interpreted from specific geological evidence and knowledge.*
- *Mineral Resources are subdivided, and must be so reported, in order of increasing 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 100 m by 100 m in some areas, to 50 m by 50 m 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 74 and Figure 75) was based on the following criteria:

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

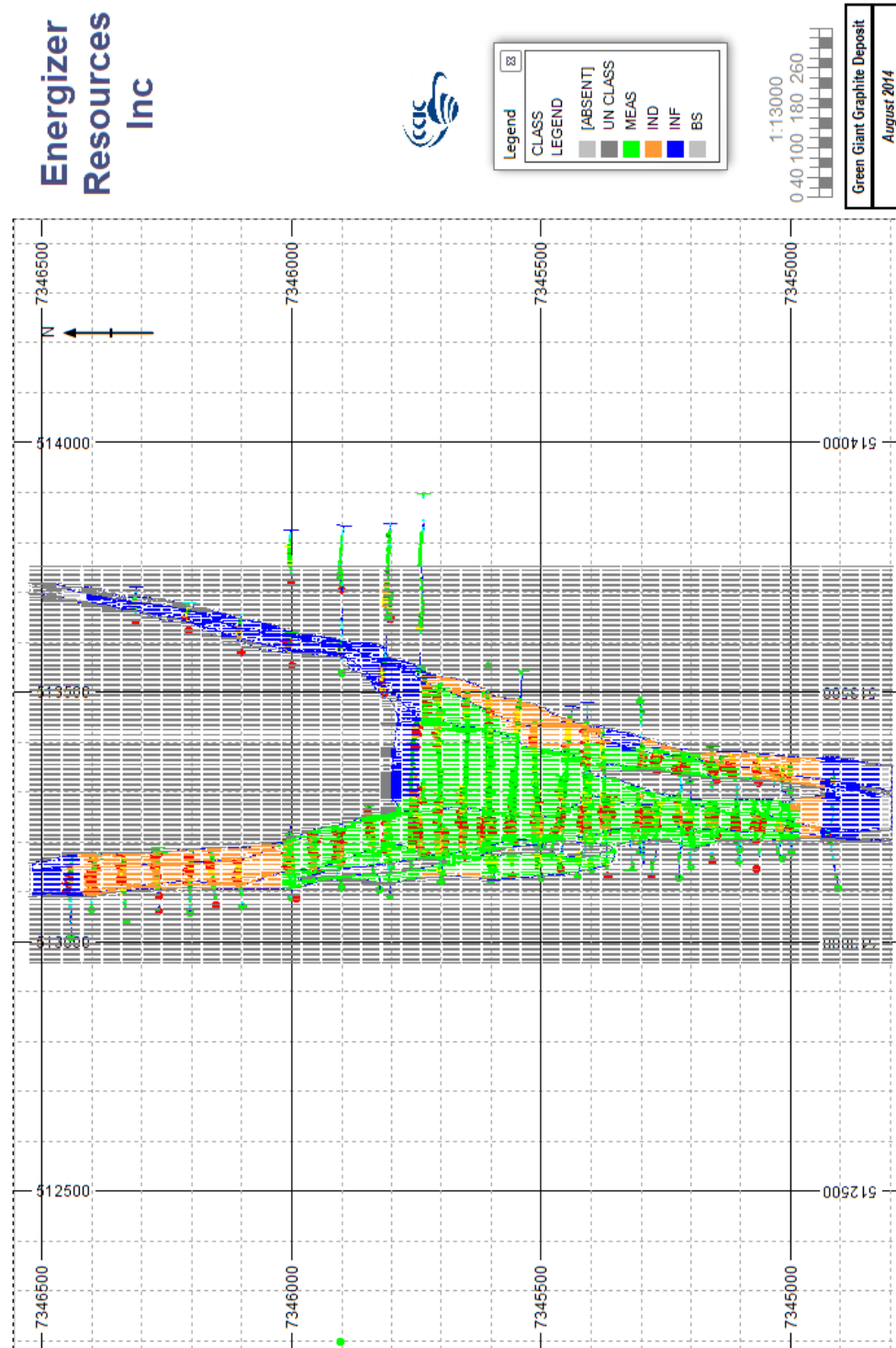


FIGURE 74: PLAN SHOWING MINERAL RESOURCE CLASSIFICATION.

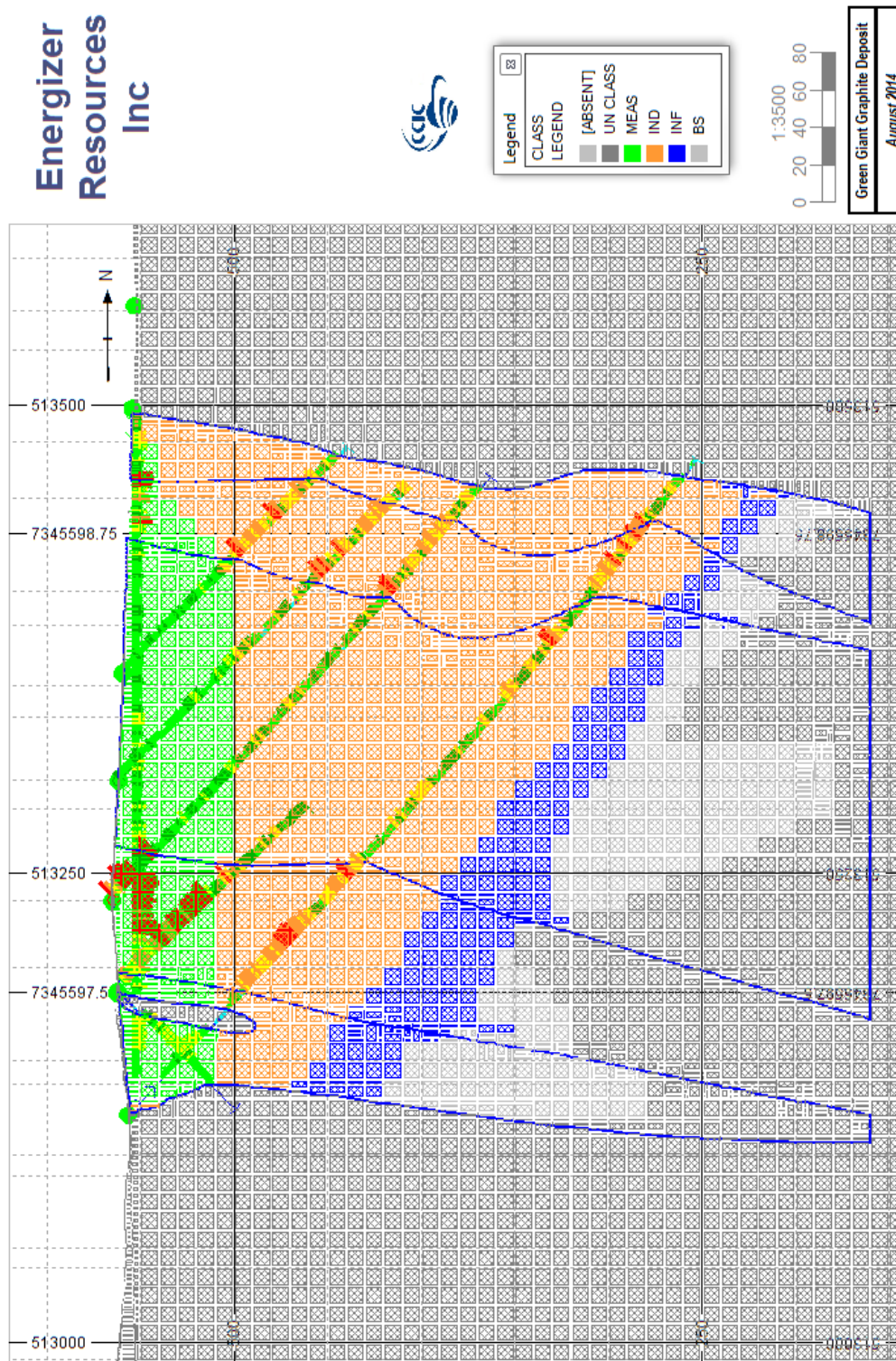


FIGURE 75: SECTION SHOWING MINERAL RESOURCE CLASSIFICATION.

14.10 RESOURCE TABULATION

The current mineral resource estimate for Molo is summarised in Table 33 below. A RD of 2.36 t/m³ was assigned to the mineralised domains. The mineral resources are classified as Measured, Indicated and Inferred categories according to the CIM Definition Standards. Resources within the low-grade domain are stated at a 2 % C cut-off, and resources within the high-grade domains are 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. 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. 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 have been upgraded by infill drilling into the Measure Resource category.

TABLE 33: MINERAL RESOURCE STATEMENT FOR THE MOLO GRAPHITE DEPOSIT – SEPTEMBER, 2014

Classification	Material Type	Tonnes	Grade - C%	Graphite - T
Measured	"Low Grade"	13 048 373	4.64	605 082
Measured	"High Grade"	10 573 137	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"	16 681 453	7.70	1 285 039
Total Inferred		40 914 721	5.78	2 365 716

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

Notes:

- Mineral Resources are classified according to the Canadian Institute of Mining definitions.
- Mineral Resources are reported Inclusive of Mineral Reserves.
- “Low Grade” Resources are stated at a cut-off grade of 2% C.
- “High grade” Resources are stated at a cut-off grade of 4% C.
- Eastern and Western high-grade assays are capped at 15% C.
- A relative density of 2.36 tonnes per cubic metre (t/m³) was assigned to the mineralised zones for the resource tonnage estimation.

15 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVES

The mineral reserves for the Project were estimated by applying a detailed mine design to the Whittle optimisation economic pit shell output which has been drafted on the CCIC prepared resource model and the Whittle input parameters. The CCIC resource model contains Mineral resources that have been reported in the stipulated Measured, Indicated and Inferred categories. Economic, technical factors and a grade cut-off of 4.5% have been applied to the above cut-off Measured and Indicated resources only in order to report the Molo reserves. The Molo mineral reserves are thus estimated and constrained to the following:

15.1.1 Whittle Optimisation Shell

15.1.2 Final Pit 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 was drafted including the key design parameters.

- Pit Ramps

Three pit ramps namely: south, central and north ramps have been designed to cater for the full length of the pit and are designed at a width of 15m. The ramp width is designed to allow dual way traffic and also satisfies the industry norm of the ramp width at least 3.5 times the width of the largest vehicle. The tipper trucks (20t) at a width of 2.5m are more than suited for these ramps. Allowance in the design has also been made for the use of bigger haul trucks if stage 2 expansion require it. The ramps are designed at a 10% gradient and ramp switchbacks or turns are designed in on a 180 degree turn with an inner radius of 15m.

- Bench Design

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

A push back in year 5 designed at limiting the amount of waste mined in the early years and maximising feed grade in order to optimise the project's economics.

The final pit design resulted in a pit that is 1.52 km long, 547m wide and 105m deep.

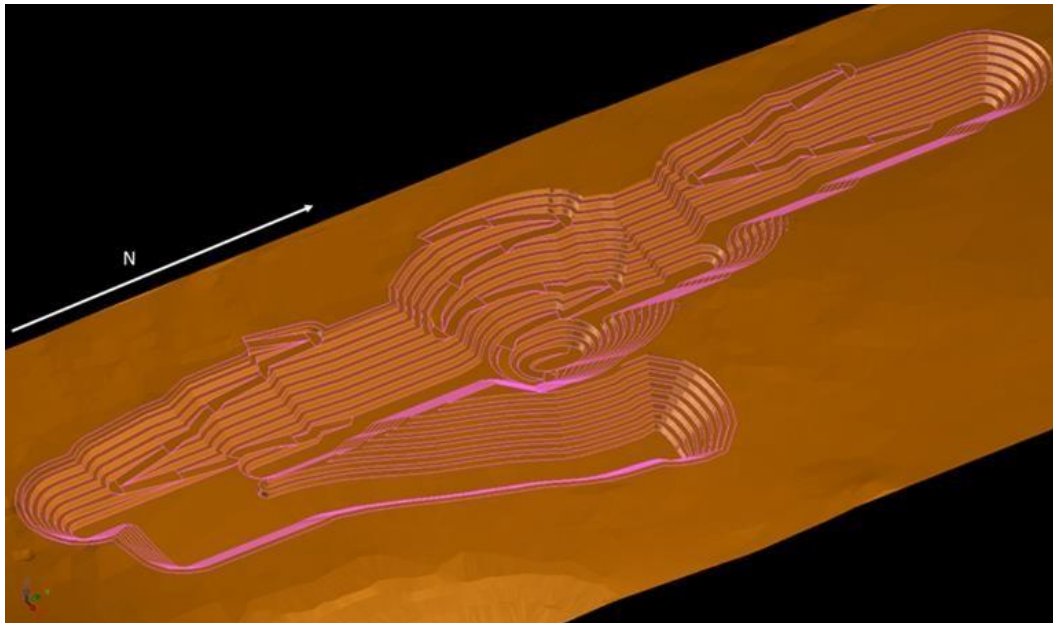


FIGURE 76: MOLO FINAL PIT DESIGN

15.1.3 Mining Modifying Factors

- Mining Dilution

Dilution is waste material that is mined together with ore and thus reduces the overall grade of the mined ore and also increases the tonnages of the ROM.

Despite Molo being a massive deposit regarding its strike, dip and width, dilution as a result of mining at the Molo deposit is still expected to be significant due to:

- The high-grade zones occur near the outer limits of the deposit and therefore more waste must be mined with depth
- The Molo deposit has an internal waste feature near the central southern section of the deposit and for geotechnical reasons, this waste material must be mined albeit selectively, and
- The transitional nature of the contact between ore and waste

For typical massive deposit types such as Molo, it is generally accepted that the dilution is 3%. Detailed grade control and dilution test work will be initiated at various bench levels once mining has commenced.

- Mining Recovery

Of the ore that is planned, or scheduled to be mined, a portion of that mineralised material that is in practice left unmined in the pit due to mineralised material / waste boundaries, geotechnical considerations, water in the pit, or other practical issues depending on the nature of the deposit. The mining recovery of the deposits depends on the process methodology and this can range from 82% to 90%. The massive nature of the Molo deposit, the absence of major geotechnical anomalies and the flexibility

benefits from using small equipment indicates that the mining recovery can comfortably be set at 95%.

- **Measured and Indicated Resources**

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 respectively. For technical and economic reasons, only Measured and Indicated resources that are inside the designed final pit and that are also above the grade cut-off of 4.5% are reported as proven and probable reserves. Inferred resources are not included in the reserve's declaration and also all other Measured and Indicated resources that are below the grade cut-off are also not included in the reserves statement as they are planned to be stockpiled separately as reject material.

- **Cut-Off Grades**

The cut-off grade is enforced simply to improve the economics of the entire project, there is thus potential for the reject material to be treated in the mill at a later date if economic conditions are more favourable. The rejected Measured and Indicated resources inside the pit are 5.1 Mt at an average grade of 3.26% C and these are not included in the reported Proven and Probable reserves.

The Inferred resources inside the final designed pit are 0.96 Mt at an average grade of 5.88% C and these are also not included in the reported reserves.

TABLE 34: MOLO RESERVES STATEMENT

Reserve Statement		As at January 2015		
Classification	Material type	Ore Tonnes	Grade - C%	Graphite - Tonnes
Proven Reserves	High Grade	9 889 536	7.76%	767 795
	Low Grade	4 280 205	5.24%	224 374
	Total	14 169 741	7.00%	992 169
Probable Reserves	High Grade	6 171 268	7.63%	471 069
	Low Grade	2 095 676	5.31%	111 218
	Total	8 266 944	7.04%	582 287
Grand Total		22 436 685	7.02%	1 574 456
1. All reserves estimated at an economic grade cut-off of 4.5%				

15.2 FACTORS AFFECTING THE MINERAL RESERVE ESTIMATE

In the mine design and production scheduling all the factors listed below and described previously have been considered in the final reserve declaration:

- Final Pit Design
- Pit Ramps
- Bench Design
- Mining Dilution
- Mining Recovery
- Cut-off Grades

16 MINING METHODS

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. In this mining method, the following activities are executed:

- The land is cleared, topsoil is removed and stockpiled at designated sites for use in future land rehabilitation. Depending on the extent of the base of weathering, any further waste, or ore that can be removed by free-digging is removed and stockpiled accordingly. The topsoil is planned to be used as a berm around the pit to prevent water flow into the pit and to minimise transportation costs
- In a number of cyclic processes, the waste and/or mineralised material is drilled, charged with explosives and blasted, excavated, hauled and dumped in designated sites
- At strategically planned periods the waste around the boundary of the pit is removed in order to mine out deeper ore

The conventional open pit mining activities are carried out with small to medium sized mining equipment including 20t dump trucks, a 2 m³ excavator and an 8 m³ Front End Loader.



FIGURE 77: MOLO OPEN PIT MINING BENCH LAYOUT

Detailed geotechnical and hydro-geological studies have been conducted and the reports indicate that there are no fatal flaws regarding the adoption of the open pit mining as the preferred mining method.

16.1 OPEN PIT OPTIMISATION

By using the prepared resource model together with the required economic and technical mine design input parameters, the Whittle Optimization software applies pessimistic and optimistic factors to the product price whilst generating corresponding nested pit shells, or possible pits at their specific total economic values. Whittle executes several possible iterations on the product price or revenue in order to obtain the best and optimal pit size and shape in three dimensions. This optimisation process in the Whittle software is based on the Lerchs-Grossman algorithm. The pit shell that ends up with the highest pit value is then selected as the optimal pit and this pit is established as the basis for any further detailed mine design work.

The Molo resource model has been delineated into two grade zones, high grade and low-grade zones. The tonnes and grades of these high grade and low-grade resources are reported in the resource statement at 4% and 2% cut-off grades, respectively. Since the graphite product is priced according to the flake size distribution and carbon content, test work studies have been conducted in order to determine if there are relationships between the grade of the ore fed to the mill and the flake size and carbon content of the graphite product. These studies initially indicated that the low-grade ore yielded larger flake sized product although this was not conclusive and therefore the Whittle optimisation has been run on grade value only. As a result of this the model naturally targets high grade ore early in the LOM.

TABLE 35: WHITTLE INPUT PARAMETERS

Mining	Value	Unit
Max bench height	8	M
Minimum bench width	6.14	m
Batter angle	75	deg
Overall Slope angle	46	deg
Minimum pit bottom	30	M
Mining recovery	95	%
Dilution	3	%
Mining operating cost	6.78	USD
Mining re-handle cost	0.56	USD
Processing		
Plant recovery	88.3	%
Mining re-handle cost	0.56	USD
Processing unit cost	19.11	USD
G & A cost	12.64	USD
Mill Throughput	240	Ktpa

Mining	Value	Unit
Product/Selling		
Product price	1 000	USD
Royalty	0%	%
Selling cost	190.55	USD

16.1.1 Pit Slopes

A detailed geotechnical study at the Molo deposit has been completed and the analysis thereof has yielded recommendations for the optimal pit slope angles aimed at maximising extraction and maintaining pit wall stability. Indications are that an overall pit slope angle of 46° could be adopted although the slope angles in the fresh and hard rock material could be made steeper, up to 48°.

16.1.2 Mining Dilution

Dilution is waste material that is mined together with ore and thus reduces the overall grade of the mined ore and also increases the tonnages of the ROM.

Despite Molo being a massive deposit regarding its strike, dip and width, dilution as a result of mining at the Molo deposit is still expected to be significant due to:

- The high-grade zones occur near the outer limits of the deposit and therefore more waste must be mined with depth;
- The Molo deposit has an internal waste feature near the central southern section of the deposit and for geotechnical reasons, this waste material must be mined albeit selectively, and
- The transitional nature of the contact between ore and waste.

For typical massive deposit types such as Molo, it is generally accepted that the dilution is 3%. Further dilution studies will be investigated at various bench levels once mining has commenced.

16.1.3 Mining Recovery

Of the ore that is planned, or scheduled to be mined, a portion of that mineralised material that is in practice left unmined in the pit due to mineralised material / waste boundaries, geotechnical considerations, water in the pit, or other practical issues depending on the nature of the deposit. The mining recovery of the deposits depends on the process methodology and this can range from 87% to 95%. The massive nature of the Molo deposit, the absence of major geotechnical anomalies and the flexibility benefits from using small equipment indicates that the mining recovery can reasonably be set at 95%.

16.1.4 Plant Recovery & Costs

The plant recovery of 88.3% is based on metallurgical test work results for the Molo samples.

The mining cost is based on owner mining equipment scenario where detailed and up to date quotations by potential suppliers were sourced.

The product price is quoted as USD\$1 000. The lower plant recovery attempts to maximise the low-grade ore which is expected to produce larger flake sized product.

16.2 OPEN PIT FINAL DESIGN

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

16.2.1 Ramps

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 also satisfies the industry norm of the ramp width at least 3.5 times the width of the largest vehicle. The (20t) tipper trucks at a width of 2.5 m are more than suited for these ramps. Allowance has also been made for the use of bigger haul trucks if needed.

The ramps are designed at a 10% gradient and ramp switchbacks or turns are designed in a 180° turn with a radius of 15 m.

16.2.2 Bench Design

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

16.3 MINE PLANNING

The Phase 1, mine plan is aimed at producing 240 000 t per annum whilst maximising the feed grade whilst minimising the strip ratio. The Phase 2, mine plan is aimed at producing 240 000 t per annum for the first 2 years after which an expansion to 720 000 t per annum is planned. Mining operations are planned to mine out the narrower and shallower southern and northern portions of the pit, before mining the deeper central section through the pit access ramp in the central section of the pit. This is done to ensure that access to this section of the pit is not cut off when the deeper central section of the pit is mined.

The mine plan has been drafted such that a fairly consistent ratio of high grade to low grade ore, as well as a consistent head grade in the run of mine is supplied to the mill. The average high grade to low grade ore ratio in the run of mine is 60% to 100% high grade and the balance low grade, at the average head grade is 8.05%. Ore is drilled, blasted, loaded, hauled and tipped onto ROM stockpiles or in the ROM tip in accordance with the required blending ratio. The material is crushed blended and conveyed to a processing plant which is planned to recover between 88.3% of the graphite. Depending on the ROM feed rate, feed grade and process recovery efficiency in the processing plant, graphite products from 15.3 ktpa to 18.3 ktpa can be produced.

16.3.1 Production Schedule

The final pit production schedule is based on a run of mine production target of 720 000 t per annum. To sustain this production rate an average waste to ROM stripping ratio of 0.52 waste tonnes (inclusive of reject ore tonnes) per 1 ROM tonne will be required over the LOM. The material movement in the production schedule comprises the high-grade ore stockpile, low grade ore stockpile, reject ore stockpile and waste dumps.

The Phase 1 and 2, 33-year production schedule is depicted in the following Table 36:

TABLE 36: MOLO PRODUCTION SCHEDULE FOR THE LOM

Molo Production Schedule			Grade cut off of 4.5%											
Year	Topsoil	Waste	High Grade Ore	HG Ore Grade %	Low Grade Ore	LG Ore Grade %	Reject Ore	Reject Ore Grade %	Total Waste	Plant Feed Ore excluding Reject Ore	Plant Feed Ore Grade %	Total Tonnes Mined	Total Plant Feed Tonnes	Total Product Tonnes
1	29 339	117 513	172 594	8.1%	27 406	7.4%	53 867	4.7%	200 718	200 000	8.0%	400 718	200 000	14 214
2	21 695	113 420	192 804	7.9%	47 196	5.9%	25 604	4.7%	160 718	240 000	7.5%	400 718	240 000	15 852
3	29 665	784 212	539 968	7.7%	184 903	5.7%	359 819	3.5%	1 173 696	724 871	7.2%	1 898 566	724 871	46 273
4	16 507	864 224	671 805	8.0%	58 318	5.9%	467 284	3.5%	1 348 015	730 123	7.9%	2 078 138	730 123	50 618
5	9 756	928 613	650 554	8.2%	56 904	6.3%	653 632	3.5%	1 592 000	707 458	8.1%	2 299 458	707 458	50 403
6	79 626	858 413	599 756	8.1%	127 800	5.5%	369 600	3.2%	1 307 638	727 555	7.7%	2 035 194	727 555	49 257
7	30 845	651 567	520 795	7.8%	188 274	5.5%	486 351	3.2%	1 168 763	709 069	7.2%	1 877 832	709 069	44 968
8	29 103	365 033	504 920	7.7%	219 909	5.4%	397 390	3.1%	791 526	724 829	7.0%	1 516 355	724 829	44 639
9	10 121	517 459	502 831	7.7%	217 085	5.4%	445 983	3.2%	973 564	719 915	7.0%	1 693 479	719 915	44 524
10	7 532	253 455	503 022	7.7%	217 108	5.4%	444 061	3.2%	705 047	720 130	7.0%	1 425 177	720 130	44 478
11	18 369	206 272	503 970	7.7%	216 030	5.3%	355 296	3.3%	579 936	720 000	7.0%	1 299 936	720 000	44 479
12	27 130	324 069	503 970	7.8%	216 030	5.2%	337 273	3.5%	688 472	720 000	7.0%	1 408 472	720 000	44 624
13	7 175	345 860	503 970	7.8%	216 030	5.3%	332 141	3.4%	685 177	720 000	7.0%	1 405 177	720 000	44 605
14	3 070	290 262	503 970	7.7%	216 030	5.3%	298 182	3.5%	591 514	720 000	7.0%	1 311 514	720 000	44 229
15	15 663	460 753	503 970	7.6%	216 030	5.2%	262 783	3.8%	739 199	720 000	6.9%	1 459 199	720 000	43 632
16	24 732	524 700	503 970	7.5%	216 030	5.2%	265 010	3.8%	814 442	720 000	6.8%	1 534 442	720 000	43 396
17	917	432 064	503 970	7.6%	216 030	5.2%	193 280	3.8%	626 261	720 000	6.9%	1 346 261	720 000	43 637
18	27 432	427 852	503 970	7.7%	216 030	5.1%	349 323	3.3%	804 607	720 000	6.9%	1 524 607	720 000	43 986
19	4 516	394 609	503 970	7.7%	216 030	5.1%	413 859	3.3%	812 984	720 000	7.0%	1 532 984	720 000	44 272
20	415	361 686	503 970	7.7%	216 030	5.2%	359 826	3.4%	721 927	720 000	7.0%	1 441 927	720 000	44 219
21	367	488 165	503 970	7.8%	216 030	5.2%	315 732	3.3%	804 265	720 000	7.0%	1 524 265	720 000	44 366
22	-	229 577	503 970	7.8%	216 030	5.2%	418 374	3.1%	647 951	720 000	7.0%	1 367 951	720 000	44 372
23	299	474 648	503 970	7.7%	216 030	5.2%	371 965	3.2%	846 912	720 000	6.9%	1 566 912	720 000	44 155
24	7 400	624 230	503 970	7.6%	216 030	5.2%	339 279	3.3%	970 908	720 000	6.9%	1 690 908	720 000	43 904
25	5 629	657 542	503 970	7.5%	216 030	5.2%	265 441	3.6%	928 613	720 000	6.8%	1 648 613	720 000	43 359
26	1 900	488 554	503 970	7.5%	216 030	5.2%	245 016	0.0%	735 470	720 000	6.8%	1 455 470	720 000	43 350
27	-	-	503 970	7.6%	216 030	5.2%	244 047	0.0%	244 047	720 000	6.8%	964 047	720 000	43 518
28	-	-	503 970	7.6%	216 030	5.2%	210 598	0.0%	210 598	720 000	6.9%	930 598	720 000	43 858
29	-	-	503 970	7.7%	216 030	5.2%	279 176	0.0%	279 176	720 000	7.0%	999 176	720 000	44 196
30	-	-	503 970	7.7%	216 030	5.2%	33 236	0.0%	33 236	720 000	7.0%	753 236	720 000	44 196
31	-	-	503 970	7.7%	216 030	5.2%	-	0.0%	-	720 000	7.0%	720 000	720 000	44 196
32	-	-	485 636	7.7%	232 490	5.2%	-	0.0%	-	718 126	6.9%	718 126	718 126	43 704
33	-	-	129 891	7.7%	84 224	5.2%	-	0.0%	-	214 115	6.7%	214 115	214 115	12 700
Totals	409 203	12 184 750	16 057 934	8.4%	6 198 256	6.3%	9 593 427	4.54%	22 187 381	22 256 191	8.05%	44 443 572	22 256 191	1 386 180

The scheduled ROM tonnes in this LOM schedule add up to a total of 22.4 Mt and this is equivalent to the reported Proven and Probable reserves statement including Measured and Indicated resources at 22.4 Mt.

16.3.2 Production Schedule Results

The following graphs and figures summarise the production schedule results:

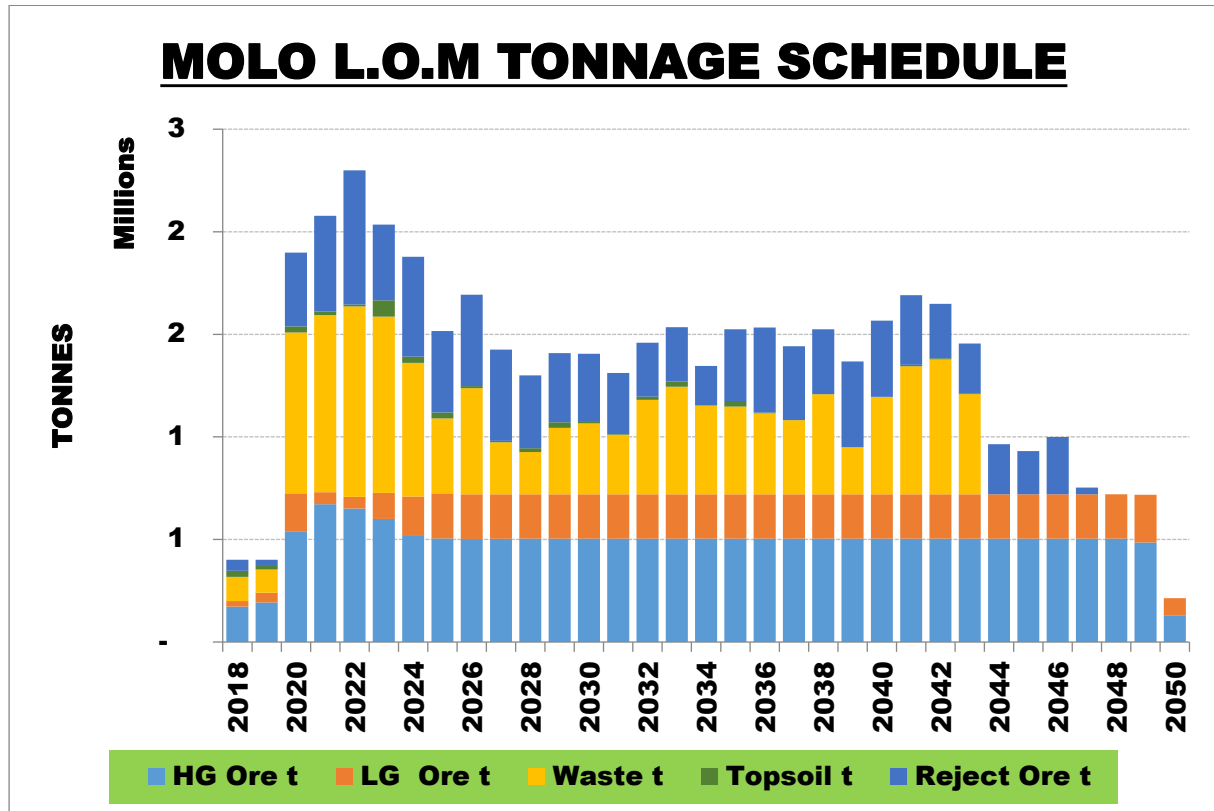


FIGURE 78: MOLO ANNUAL PRODUCTION SCHEDULE FOR THE LOM

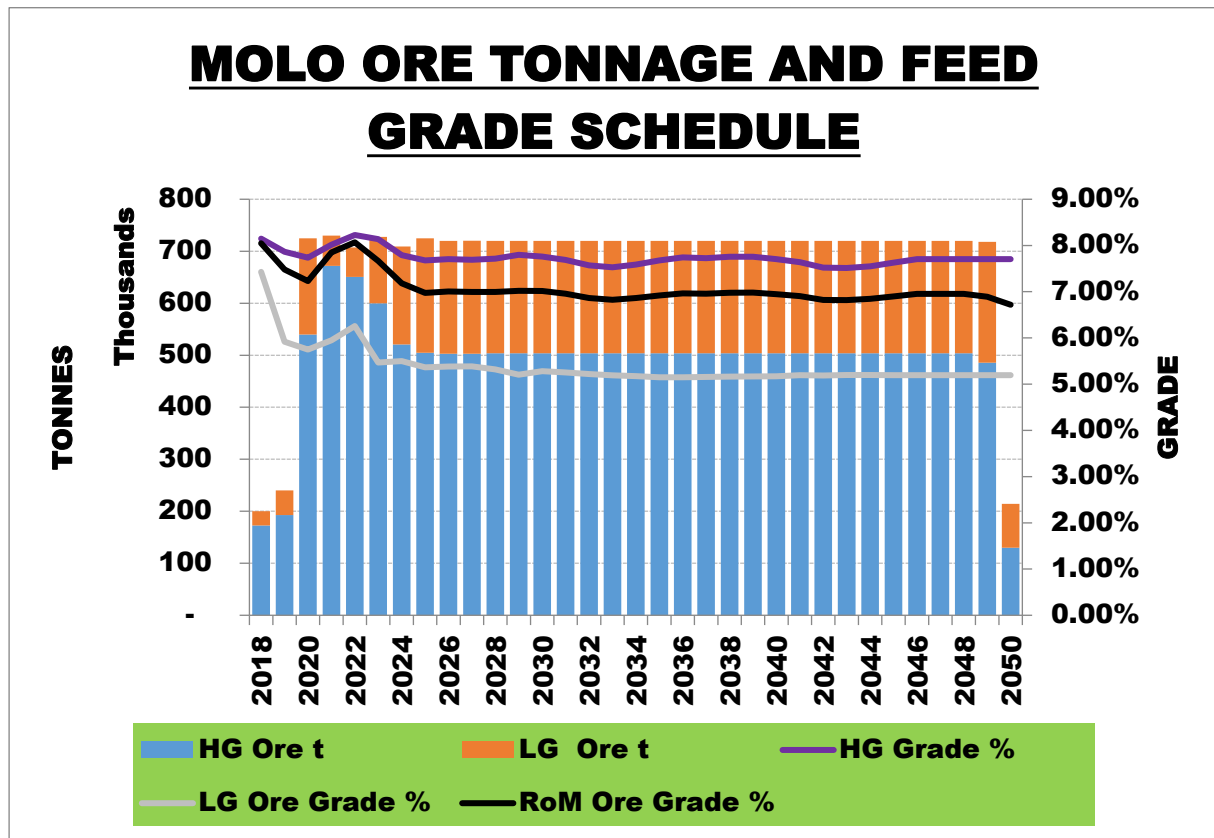


FIGURE 79: MOLO MILL FEED SCHEDULE FOR THE LOM

16.4 MINING OPERATIONS

16.4.1 Mining Production and Support Equipment

To facilitate the mining and material handling required in the mining cycle the following production and support equipment will be required.

TABLE 37: MINING PRODUCTION AND SUPPORT EQUIPMENT

MACHINE / ITEM	REPLC. LIFE (hours)	Phase 1 Equipment Numbers	Phase 2 Yr3 to Yr33 Equipment Numbers
EXCAVATORS & LOADERS			
SANY SY465H Excavator - 2m ³	15,000	1	4
LIUGONG Front End Loader 3m ³	15,000	2	2
HAUL TRUCKS			
SINOTRUK HOWO Tipper Truck 16m ³ (Howa)	15,000	2	6
SINOTRUK HOWO Water Truck 20000L (Howa)	15,000	1	1
MOBILE EQUIPMENT			
HOWA Truck Mounted Drill Rig	10,000	1	2
SANY Motor Grader 200HP	15,000	1	1
SANY Compactor 10t	15,000	1	1
SHANTUI SD22W HD Tracked Dozer	15,000	1	2
SERVICE & SUPPORT EQUIP.			
Buses 60-seater	6 years	2	2
Maintenance / Diesel Truck	6 years	1	1
Pump Diesel 50m ³ /hr	6 years		2
Supervision Vehicle	6 years	6	6
Light Plant			
Spares (2 Years) -			
TOTAL		19	30

To maintain the targeted mining production, a mining equipment selection exercise was done on the basis of the estimated annual operating hours, cycle times and the effective equipment capacities. This exercise encompassed the primary mining equipment including all the loaders, haul trucks, drill rigs and support equipment. With the mine operating on a one 10-hour shift per 5-day week, 8 effective hours per shift and 260 days per annum, the production mining equipment selected is capable of moving 450 000 t per annum during the initial two years and then 2 500 000 t per annum thereafter.

One truck mounted drill rig was selected to drill the designed 120 mm blast holes and will be capable of drilling, (5 hours per day 5 day a week) approximately 19,500 metres per annum at a penetration rate of 15 metres per hour.

The initial haulage distances for the tipper trucks are expected to be approximately 1.2 kilometres, from the open pit-pit to the ROM tip, or 2.0 kilometres the pit to stockpile areas.

16.4.2 Labour

The number of Expats to be used in the Project has been limited to as few as possible. This is due to the mining being relatively simple, and the productivity levels required being achievable considering the conservative scheduling approach.

This will allow adequate time for operator training and a reasonable production build-up, to be executed.

TABLE 38: MINING LABOUR COMPLIMENT FOR PHASE 1

THE COMPANY'S STAFF LIST: Mining				M	S	U
Personnel Complement Breakdown	Responsibility	Grade	No.			
Management						
Safety & Training						
Safety Officer	Safety	C4	1		1	
Training Officer	Training	C4	1		1	
Safety & Training			2			
Technical Services						
Tech Manager	Technical Services	D3	0			
Surveyor	Technical Services	C4	1		1	
Geologist	Technical Services	D2	1	1		
Mine Planner	Technical Services	C4				
Technical Services			2			
Mining						
Production Foreman	Mining	C4	1		1	
Miner Blaster	Mining	A2	1		1	
Blasting / Grade Assistants	Mining	A2	3			3
Pumping / Cleaning	Mining	A2	2			2
Excavator Operator	Mining	B3	1			1
FEL Operator	Mining	B3	2			2
ADT Operator	Mining	B3	2			2
Drill Operator	Mining	B3	1			1
Dozer Operator	Mining	B3	1			1
Truck W/E Operator	Mining	B3	1			1
Backhoe Operator	Mining	B3	0			0
Comp / Grader Operator	Mining	B3	1			1
Relief Operator	Mining	B3	3			3
Mining			19			
Engineering						
Engineering GES	Engineering	D1	1	1		
Diesel Mechanic	Engineering	C2	2		2	
Auto Electrician	Engineering	C2	1		1	
Boilermaker	Engineering	C1	1		1	
Diesel Assistant	Engineering	A3	3			3
Boiler Assistant	Engineering	A3	1			1
Engineering			9			
TOTAL MINING STAFF			32	2	9	21

TABLE 39: MINING LABOUR COMPLIMENT FOR PHASE 2 YEAR 3 TO 33

THE COMPANY'S STAFF LIST: Mining				M	S	U
Personnel Complement Breakdown	Responsibility	Grade	No.			
Management						
Safety & Training						
Safety Officer	Safety	C4	1		1	
Training Officer	Training	C4	1		1	
Safety & Training			2			
Technical Services						
Tech Manager	Technical Services	D3	0			
Surveyor	Technical Services	C4	1		1	
Geologist	Technical Services	D2	1	1		
Mine Planner	Technical Services	C4				
Technical Services			2			
Mining						
Production Foreman	Mining	C4	2		2	
Miner Blaster	Mining	A2	2		2	
Blasting / Grade Assistants	Mining	A2	6			6
Pumping / Cleaning	Mining	A2	2			2
Excavator Operator	Mining	B3	5			5
FEL Operator	Mining	B3	2			2
Truck Operator	Mining	B3	5			5
Drill Operator	Mining	B3	2			2
Dozer Operator	Mining	B3	2			2
Truck W/E Operator	Mining	B3	1			1
Backhoe Operator	Mining	B3	0			0
Comp / Grader Operator	Mining	B3	2			2
Relief Operator	Mining	B3	3			3
Mining			34			
Engineering						
Engineering GES	Engineering	D1	1	1		
Diesel Mechanic	Engineering	C2	4		4	
Auto Electrician	Engineering	C2	2		2	
Boilermaker	Engineering	C1	1		1	
Diesel Assistant	Engineering	A3	3			3
Boiler Assistant	Engineering	A3	1			1
Engineering			12			
TOTAL MINING STAFF			50	2	14	34

16.4.3 Drill and Blast Design

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
- Emulsion
- Boosters
- Electric Detonators
- Detonating cord
- Basting wire

The drill and blast parameters:

- Hole diameter = 120 mm
- Bench height = 4 m
- Stemming length = 3.0 m
- Burden = 3.6m
- Spacing = 4.0 m
- Sub drill = 0.4 m
- Explosive density = 1.15 g/cm³
- Powder factor = 0.45
- Charge mass = 26 kg per hole

16.4.4 Mine Infrastructure

Provision has been made at the plant site for a mining office which will accommodate 6 people and parking facilities outside the office. The mine terrace adjacent to the plant site has provision for 2 equipment service workshops, tyre storage and general storage areas, a vehicle wash bay, brake test ramp and parking for the mine fleet. Additional ablution facilities have been allowed for at the mining infrastructure area.

The change house will be a shared facility which will be utilised by all personnel on the mine, including the plant operators, owners and security personnel.

Unless otherwise specified, all buildings will be prefabricated steel framed buildings installed on a concrete foundation slab. The proposed buildings to be installed on site include:

- 2 x EMV Workshop, wash bay, tyre bay, offices, fuel farm and storage yard
- Mine administration offices
- Change House including dressing room, laundry and boiler room (shared)
- Ablution facilities

16.4.5 Explosives Storage

The contracted explosives supplier shall use an approved system, (Code of Practice) of delivering raw materials and chemicals which should conform to the legal and safety standards on the mine. The raw materials required to obtain the required quality of explosive will be stored separately and blended once required. A demarcated fenced off storage facility shall be used to store the raw materials and chemicals.

An approved mixing machine designed and supplied by the contracted blasting contractor shall be set up on site to produce the daily ANFO requirements. The raw chemicals shall be stored in appropriated shed / silos set up on the mine. All blasting accessories will be stored in approved explosives and accessories magazines. Only daily requirements shall be withdrawn in accordance with the approved procedure.

The mine shall utilise the magazines to be constructed at the mine. The magazines will be capable of storing boosters, detonator cord, safety fuse, packaged explosives and accessories in accordance with Madagascar laws and regulations.

The entire explosives infrastructure shall be located at a safe site that will meet the regulatory and standard requirements.

A maximum consumption of 77 tonnes of explosives per annum is expected during the first two years and then it should increase to 236 tonnes per annum.

17 RECOVERY METHODS

This section describes the process design basis adopted in order 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 graphite ore deposit in Madagascar in accordance with the mining design basis set out in Section 16 above.

The ore processing circuit consists of three-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, for an expected overall graphite recovery of 88.3% to final concentrate, at the required grade, from an average plant feed head grade of 7.0%. This will produce an estimated average of 15-17 kilotonnes per annum (ktpa) of final concentrate over the LOM, at the design throughput tonnage of 240 ktpa.

Negative variations in graphite head grade could be expected to lead to reduced recoveries and/or less graphite product for the same feed tonnage into the plant.

17.1 PROCESS DESCRIPTION

17.1.1 Primary, Secondary and Tertiary Crushing

The crusher circuit is fed with ROM material using a front-end loader (FEL) at a rate of 50 t/h. The ROM material is screened using a ROM static grizzly screen (100-SR-004), located at the top of the primary crusher bin (100-BN-006) to reject rocks larger than 300 mm (oversize material). The oversize material shall be stockpiled near the feed bin and may be manually crushed using rock breakers. The static grizzly undersize material (<300 mm) is further screened using the grizzly feeder (100-FD-008), which is fitted with 70 mm openings; the oversize material (>70 mm) is discharged into the jaw crusher (100-CR-010). The crusher is set to a Closed Side Setting CSS value of 75 mm and is expected to produce 100% <120 mm material. The crusher product and the grizzly undersize are discharged onto the primary crusher discharge conveyor (100-CV-011).

The primary crusher product is fed to the classification screen (100-SR-040) via the classification screen feed conveyor (100-CV-012). The classification screen is fitted with two decks; the top deck is fitted with 35 mm screen panels while the bottom deck is fitted with 16 mm panels. Oversize from the top deck (>35 mm) discharges into the secondary crusher feed bin (100-BN-022) and is fed to the secondary crusher (100-CR-026) via a feed conveyor (100-CV-034) and the secondary crusher feeder (100-FD-024).

Oversize material from the bottom deck (<35+16 mm size fraction) flows by gravity into the tertiary crusher feed bin (100-BN-028) and is fed to the tertiary crusher (100-CR-032) via the tertiary crusher feed conveyor (100-CV-036) and the tertiary crusher feeder (100-FD-030). Products from the secondary and tertiary crushers are combined and recycled back to the classification screen. The bottom deck screen undersize (<16 mm size fraction) from the classification screen is fed to the mill feed bin (200-BN-058) via the mill feed conveyor (100-CV-044).

17.1.2 Primary Milling and Flash Flotation

The crushed material from primary mill feed bin is fed to the primary mill scalping screen (200-SR-070) at the flowrate of 30 t/h via the mill feed bin feeder (200-FD-060) and the primary mill feed conveyor (200-CV-066). The scalping screen is fitted with 2 mm aperture panels. Oversize from the screen (>2 mm) flows into the primary ball mill (200-ML-078) by gravity. Spillage material in the primary milling circuit is pumped to the scalping screen through the mill area spillage pump (200-PP-094).

The mill is expected to produce a product with a grind of 80% passing 500 microns; this product is discharged into the primary ball mill discharge sump (200-TK-082). Slurry from the mill discharge sump is pumped (via the primary mill discharge pump 200-PP-084/086) to the primary mill screen (200-SR-072) which is fitted with a 600 µm aperture panels and discharges the oversize (>600 µm) back into the primary mill.

The -2 mm material from the scalping screen (200-SR-070) flows by gravity into the flash flotation cell; the flash flotation cell is installed to recover graphite flakes liberated at coarse particle sizes. Flash flotation produces a concentrate at a mass yield of 3%. Flash flotation tailings are pumped into the primary mill discharge sump via the flash flotation tails pump (200-PP-090/092).

17.1.3 Rougher Flotation

The rougher flotation feed slurry (-600 µm), which consists of the combination of flash flotation tailings and the primary mill product, flows into the rougher flotation bank (300-FC-110/112/114/116/118) by gravity at the solids flowrate of 29 t/h. Frother and collector reagents are added into rougher cell no.1 (300-FC-110) and rougher cell no.3 (300-FC-114). Rougher concentrate mass pull is expected to be 8%. The flash flotation concentrate, and primary rougher concentrate are combined in the total concentrate sump (300-TK-134) and pumped to the primary concentrate cleaning circuit using the pumps (300-PP-136/137).

The tailings from the rougher flotation circuit are discharged into the combined tailings sump (300-TK-138); the combined tailings stream is then pumped to the final tailings thickener (400-TH-310) using the combined tailings pump (300-PP-140/142). Any spillage from the rougher flotation section is pumped to the primary ball mill scalping screen using the rougher flotation spillage pump (300-PP-144).

17.1.4 Primary Concentrate Cleaning Circuit

The combined concentrate is screened on the primary cleaner dewatering screen (300-SR-146); the screen is fitted with 74 µm panels. The oversize from the dewatering screen is discharged into the primary polishing mill (300-ML-148). The mill discharge and the dewatering screen undersize are discharged into the primary polishing mill discharge sump (300-TK-150). The primary mill discharge sump slurry is pumped to the primary cleaner column cell (300-FC-154) using the primary polishing mill discharge sump pump (300-PP-152). The column flotation cell tailings are pumped to the primary cleaner flotation bank (300-FC-164/166/168); the tailings stream from this bank is discharged into the primary cleaner scavenging flotation cell bank, which consists of three flotation cells (300-FC-170/172/174). The primary cleaner flotation concentrate is recirculated back to the primary cleaner column cell. The cleaner scavenger concentrate is recycled back to the polishing mill feed classification screen, while the tails are pumped to the combined tails sump in the rougher flotation area (Area no. 300) via the final cleaner tails sump (300-TK-184) and the final cleaner tails pump (300-PP-186).

The concentrate from the primary cleaner column cell is screened on the primary column flotation cell concentrate screen (300-SR-190), which is fitted with 212 µm screen panels. The oversize from this screen flows by gravity into the fine flake cleaning circuit while the undersize is pumped to this circuit using the primary column cell concentrate undersize pump (300-PP-196).

17.1.5 Fine-Flake Concentrate Cleaning Circuit

The undersize from the primary column flotation cell concentrate (fine flakes stream) is dewatered at 75 µm on the fine flake dewatering screen (300-SR-267). Oversize from the

dewatering screen flows into the fine flake polishing mill (300-ML-268); the discharge from the mill and the dewatering screen undersize are discharged into the fine flake polishing mill discharge sump (300-TK-270). The sump slurry is pumped to the fine flake column cell (300-FC-274) using the fine flake polishing mill discharge pump (300-PP-272) to produce a concentrate at a mass pull of about 7%. The fine flake column concentrate is combined with the primary flotation column screen oversize stream from the primary cleaning circuit; this combined stream is pumped to the attrition cleaning circuit (Area no. 360) using the combined cleaner concentrate pump (300-PP-280).

The fine flake column flotation tailings are subjected to cleaning (300-FC-286/288/290) and scavenging (300-FC-292/294) flotation. Concentrate from the cleaner cells is recycled back to the fine flake column cell by gravity while the scavenger concentrate is pumped (300-PP-300) back to the fine flake dewatering screen. The tailings from the scavenger cells flow by gravity back into the final cleaner tails sump in the primary concentrate cleaning circuit (Area no. 310).

17.1.6 Final Concentrate Attritioning and Cleaning Circuit

The cleaner concentrate stream is subjected to thickening in the attrition concentrate thickener (300-TH-001); overflow from the thickener is discharged into the process and storm water tank in the water services area (Area no. 600). The underflow, which has a density of 1.19 RD, is pumped to the attrition mill (300-ML-005). Product from the attrition mill flows into the attrition mill discharge sump (300-TK-010) and this is pumped to the attrition cleaner column flotation cell (300-FC-020) using the attrition mill discharge pump (300-PP-015). Tailings from the attrition column flotation cell are pumped (300-PP-035) to the attrition cleaner (300-FC-040/045) and scavenging (300-FC-050) flotation cells. The concentrate from the cleaners is recirculated by gravity back to the attrition column flotation cell while the scavenger concentrate is pumped (300-PP-060) back to the attrition concentrate thickener. The tailings from the scavenger cells are recycled by gravity back to the final cleaner tails sump in the primary concentrate cleaning circuit (Area no. 310).

17.1.7 Concentrate Handling

The final concentrate is thickened in the final concentrate thickener (500-TH-342). The thickener overflow is discharged into the process water dam in Area no. 600, whereas the underflow is filtered in the final concentrate belt filter (500-FL-358), which is expected to produce a filter cake with less than 20% moisture. The filtrate is pumped back to the final concentrate while the filter cake is discharged onto the final concentrate filter cake conveyor and fed to the final concentrate drying and bagging circuit (Area no. 570).

17.1.8 Concentrate Screening and Bagging

The filter cake concentrate from the belt filter is fed into a diesel-fired dryer (500-DR-376) via a screw conveyor (500-CV-374). The filter cake is dried to a moisture content of lower than 1% (w/w). The dry concentrate is then screened into four size fractions: +48, -48+80, -80+100 and -100 mesh. Each of the size fractions is bagged separately as final product.

17.1.9 Final Tailings Handling and Disposal

Tailings stream from the combined tailings sump (300-TK-138) in the rougher flotation circuit (Area no. 300) is thickened in the final tailings thickener (400-TH-310). The thickener underflow from the thickener, which has a slurry density of 1.51 RD, is pumped to the final tailings filtration circuit and disposal circuit through the final tails thickener underflow pump (400-PP-312/314) and the final tails pump (400-PP-320/322). The thickener overflow stream flows into the process and storm water tank in the water services area.

This part of the circuit is operated in a semi-batch manner and is expected to produce a filter cake with 15% moisture. The final tailings thickener underflow material from the final filtration feed tank (400-TK-040) is pumped to the final tails filter; the filtrate is discharged into the final tails filtrate sump (400-TK-056) and recycled to the final tails thickener while the filter cake is stockpiled and discarded by an FEL.

17.1.10 Water Services

Thickener overflow streams from the final tails thickener (400-TH-310), final concentrate thickener (500-TH-342) and attritioning concentrate thickener (300-TH-001) are discharged into the plant process tank no.1 (400-TK-470). Process water tank no.1 is interconnected with process water tank no.2 (600-TK-472). Make-up raw water is pumped into process water tank no.1 from a well field and from the storm/pollution control dams (600-TK-478/480/482). Process water is pumped from process water tank no.2 (600-TK-472) using the plant raw water pump (600-PP-425/427) and distributed to the following circuits via process water pump no.1 (600-PP-474) or no.2 (600-PP-476):

- Attritioning
- Fine flake cleaning
- Primary milling
- Rougher flotation
- Primary concentrate cleaning
- Secondary cleaning

A portion of the raw water from the well field is screened using a mesh screen (600-SR-428) and distributed to the following areas:

- Flocculent make-up plant
- Potable water plant
- Gland water service
- Fire water

17.1.11 Air Services

A blower package (600-HB-484) provides low pressure air into the various flotation cells in the plant with the exception of the column cells. Column cells are supplied with air from the instrument air compressor (600-HA-488) via the receiver (600-TK-498).

Instrument air is supplied from the instrument air compressor (600-HA-488), dried using the desiccant air drier (600-DR-492) and fed to the various actuated valves via the receiver (600-TK-496).

Filtration air is supplied by the filtration air compressor (600-HA-491) and the air receiver (600-TK-499).

17.1.12 Reagents

Collector

The collector (diesel) is stored in an isotainer (500-TK-500). This is dosed via an outlet manifold and 13 peristaltic reagent dosing pumps.

Frother

The frother is stored in an isotainer (500-TK-524). This is dosed via an outlet manifold and 13 peristaltic reagent dosing pumps.

Flocculent

Flocculent is delivered in 25 kg bags and off-loaded into the plant reagent receiving area. It is then transferred to the flocculent make-up area using a forklift. The flocculent make-up plant is a vendor package.

The make-up plant is divided into two: one half of the plant feeds the final concentrate and the attrition thickener while the other feeds the final tailings thickener.

Flocculent powder is fed into the hopper (800-TK-574); this is fed to the mixing tank (800-TK-580) via a screw feeder (800-FD-576). Clean water is introduced into the make-up tank, where the flocculent is prepared to the correct concentration. Flocculent mixed to the correct concentration overflows into the dosing tank (800-TK-584), from which it is dosed to the final concentrate thickener and attrition thickener via the dosing pumps (800-PP-586/587).

Flocculent feeding the final tails thickener is prepared in a manner similar to the flocculent fed to the final concentrate and attrition thickener. It is mixed in the mixing tank (800-TK-558), fed to the mixing tank (800-TK-564) via the screw feeder (800-FD-560) and dosed to the final tailings thickener via the dosing tank (800-TK-568) and dosing pump (800-PP-570).

Coagulant powder is fed into the hopper (800-TK-594); this is fed to the mixing tank (800-TK-600) via a screw feeder (800-FD-596). Clean water is introduced into the make-up tank, where the coagulant is prepared to the correct concentration. Coagulant mixed to the correct concentration overflows into the dosing tank (800-TK-604), from which it is dosed to the three thickeners via the dosing pumps (800-PP-606/612/615).

17.2 TAILINGS STORAGE FACILITY (TSF)

Due to the substantially reduced tonnages for the project as envisaged, tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining. Despite this co-disposal approach, a detailed design has been completed, complete with environmental and social impact assessment and closure to allow for the upgrade to a more conventional, cyclone facility, should the throughput be increased during the life of the mine. This approach has been pursued to ensure that sufficient flexibility is built into the project development strategy to accommodate the anticipated increase in market demand.

- Section 17.2 of the NI43-101 dated 13 July 2017 indicates that due to the reduced tonnages the tailings shall be dried and co-disposed with the waste rock generated as part of the open cast mining;
- The work undertaken by Epoch predominantly involved looking at possible alternative sites for the disposal of the thickened slurry tailings and determining CAPEX for a smaller capacity tailings disposal facility to that assessed by SRK during the Molo 2015 FS (see various attachments);
- With the decision to go dry tailings disposal and co-disposal with the waste rock, the need for a tailings disposal facility isn't warranted as it forms part of the waste rock dump, with the majority of the waste mass being waste rock;
- The above approach results in little to minor changes in the infrastructure outside of what would be required for a waste rock dump itself. (typically storm water management infrastructure & settlement ponds);
- Additional work in terms the characterisation of the co-disposed waste stream and how it behaves when disposed may be required prior to moving to further phases of the project.

17.3 WATER

17.3.1 Baseline Environment

The climate of Madagascar is highly varied, largely due to its geographical position in the Indian Ocean, variable topography and associated microclimates. The climate at the Project site, based on the Köppen-Geiger climate classification system, comprises elements of hot semi-arid and tropical savannah climates.

Rainfall in areas around the site varies between 750 and 850 mm per annum, while site rainfall is simulated at 799 mm per annum. Seasons are nominally divided into a wet and dry season, with relatively small variations in temperature, but a strongly defined rainy season between November and March. The proposed port side at Ford Dauphin has a more temperate climate rainfall of 1500 mm per annum is distributed more evenly across the year. Both sites should expect average annual Symons-Pan evaporation of between 1380 and 1700 mm per annum.

While extreme weather events across Madagascar are dominated by the impacts of Cyclones and Tropical Storms, both sites seem to be fairly isolated and protected from these weather

systems. Extreme 1-day rainfall that could be expected, on average, once in 50 years is estimated at 171 mm for the Molo site and 283 mm for Fort Dauphin.

The Molo site is located on the divide of three major river basins: Onilahy River Basin (32 361 km² catchment), Linta River Basin (6 177 km² catchment) and Menarandra River Basin (8 681 km² catchment). The calculated mean annual runoff is about 150 mm per annum or roughly 18.8% of the mean annual precipitation (MAP).

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. Watercourses are dry during the dry season and only flow during the rainy season.

The surface water is generally sodium, calcium, manganese – bicarbonate (chloride) dominant. Stream water quality in general complies with Madagascar and WHO drinking water criteria, except for elevated turbidity.

Local surface water sources are largely underdeveloped, and no significant local water demands were identified during water use surveys. There are no known major industrial developments, irrigation schemes, impoundments and/or major abstraction works in the river catchments surrounding the proposed Project site. The major surface water users are local communities, who use water for drinking, livestock, washing and small-scale subsistence agricultural with crops such as rice, cassava, tomatoes and bananas.

The Molo site falls within the Ampanihy shear zone with a regional north-northeast – south-southwestern trend. The local geology consists mainly of crystalline metamorphic rock such as quartzofeldspathic gneiss. No significant faulting or dyke intrusions have been identified with the site. The main aquifer zone is hosted within the weathered (0 to 30 m deep) and lower fractures rock, with most groundwater intersections occurring less than 50 m below surface. The gneiss at the proposed mine site has low to medium aquifer potential, with the marble layers to the west having a higher aquifer potential. Hydrogeological boreholes drilled in the proposed mining site have yields mostly less than 1 l/s.

A wet and dry season hydro census investigation was completed in a 5 km radius of the proposed site, and also around the nearby town of Fotadrevo. Eighteen (18) groundwater features were identified consisting of shallow boreholes, springs and hand dug wells within the study area. Groundwater use in the area is relatively low (15 to 20 m³/d), mainly for domestic and stock watering purposes. The groundwater depth measurement across the area ranged between 2 and 17.6 metres below ground level.

Groundwater is sodium-bicarbonate and calcium (magnesium)-bicarbonate dominant waters were present at the site, with subordinate sodium-sulphate (chloride) dominant waters. The total dissolved solids contents (TDS) of the groundwater is typically 150 to 550 mg/l and pH levels typically range from 7 to 8.5. Groundwater quality is mostly suitable for human consumption.

17.3.2 Hydro-geochemical assessment of waste material

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 seepage associated with the mine waste facilities. However higher sulphate loads could be expected in the long-term (up to 3 200 mg/l for tailings and 2 700 mg/l for waste rock). The pH of water emanating from waste facilities will be 6.5 to 8 over the short term, decreasing up to 5.5 in the long-term. Metal concentrations are low.

The result from the geochemical test work and modelling was incorporated into the salt balance and groundwater numerical contaminant transport modelling in order to quantify and mitigate potential impacts. Short-term runoff quality is in general acceptable for discharge based on Madagascar and international criteria.

17.3.3 Raw Water Resources

The preferred raw water make-up supply for proposed 240 ktpa process plant, potable water and dust suppression will be the wellfield proposed in a 5 km radius around the Molo site. The safe yield was calculated for the existing hydrogeological boreholes at 15.3 l/w (55 m³/hr or 1 322 m³/d) which is more than the maximum raw water requirements of -23 m³/hr or 551 m³/d. The sustainable yield for the 5 km radius wellfield, based on a 20 mm/a recharge rate and allowing for the ecological water requirements, is 101 m³/hr or 2 429 m³/d. The sustainable yield for long-term drought conditions is 60 m³/hr or 1 429 m³/d and the wellfield will be operated based on this conservative rate.

Two existing boreholes MMB12 and MMB13 have been identified for supplying the raw water to the mine. The boreholes have a combined safe yield of 42.8 m³/hr. The calculated daily borehole abstraction is impacted on by the dust suppression requirement (seasonal variation and over life of mine) and will vary between 222 and 551 m³/d. It was concluded that the groundwater resources of the area can with a high degree of confidence supplying the raw water make-up for the proposed mine.

17.3.4 Conceptual Storm Water Management Plan

Mine storm water run-off areas were defined, and storm water management measures were 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 (3) 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 at all PCDs.

17.3.5 Water and Salt Balances

The water balance serves to investigate the water availability for the Project based on the given water demands and available surface or groundwater resources, optimise a responsible water management plan, verify the PCD sizes (and sizing of other infrastructure) and to simulate water quality parameters in a mass transport balance model (Salt Balance).

The raking of water sources to be used in the ore beneficiation process, in order of priority, are 1) Sewage effluent return, 2) PCD water from storm water/seepage from plant, co-disposal, reject dump, ore stockpiles and open pit water, 3) raw water sources (Wellfiled – Module 1).

The operational philosophy of mining operations will be to operate the process 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. Results from the salt balance, based on information from geochemical tests and modelling, suggests that PCD water under discharge conditions will comply with Madagascar and international criteria.

Both static and dynamic water balance calculations were undertaken from the Base Case plant (dry tails).

17.3.5.1 *Process water*

The total process water requirements for the Base Case is 7.58 m³/hr (clean water) and 0.67 m³/hr for process top-up.

17.3.5.2 *Open pit water*

Groundwater numerical modelling suggested a groundwater inflow rate of 8 to 590 m³/d for different mining stages over the operational life. The average groundwater and storm water volume likely to be reporting to the open pit sump over the life-of-mine is about 900 m³/d. Allowance was made for a sump pump capacity up to 3 500 m³/d.

17.3.5.3 *Water Impact Assessment*

The potential impact that mining activities are likely to have on surface and groundwater resources in the surrounding environment was analysed. These potential impacts were quantified, and a significance rating was undertaken.

Potential impacts relate to mine dewatering, wellfiled abstraction, dirty water runoff, runoff 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.

18 PROJECT INFRASTRUCTURE

18.1 MINING INFRASTRUCTURE

18.1.1 Mining Office

The mining office is located on the shared plant and mine terrace and is a pre-fabricated modular structure. The building will provide office space for the mining personnel including the mine manager, technical services manager, geology personnel, survey personnel, maintenance engineers and mining support staff.

The mine office will be constructed using modular pre-fabricated materials which will be shipped to site in a flat-pack configuration and erected and commissioned on site.

18.1.2 Earth Moving Vehicle (EMV) Workshop

The EMV Workshop is the main workshop which will serve the mining fleet (mainly the ADTs) and the LDV fleet on site. It will cater for all maintenance and re-builds of the plant and LDV vehicles and will also be able to serve the haul truck fleet if required.

Figure 80 below is an illustrative layout view of the proposed EMV workshop to be installed as part of the permanent works on site.



FIGURE 80: TYPICAL ILLUSTRATIVE LAYOUT OF EMV WORKSHOP

The workshop is sized to accommodate two (2) Mine haul trucks at any time. No provision is included for fire hose reels or hydrants. 9 kg hand held dry chemical fire extinguishers will be provided as part of the fire control system.

18.1.3 EMV Wash Bay

The EMV wash bay will be provided for the maintenance and cleaning of vehicles and plant and will service the mining fleet and the LDV fleet. One wash bay has been provided for and has been sized to accommodate the largest vehicle in the mining fleet which would be the mine haul trucks and bulldozers.

18.1.4 ANFO Storage

The ammonium nitrate / fuel oil (ANFO) storage building is located to the north of the mine and plant site, a minimum of 500 m away from all proposed and existing infrastructure, villages, product transport route, access road and the mining operation (pit). Access to the ANFO storage building is via an access road off the existing road which runs in an east-west direction past the proposed mine site and is located on an engineered terrace.

Access to the ANFO storage building 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.

18.1.5 Explosives Magazine

The explosives magazine consists of a totally enclosed reinforced concrete structure on an engineered terrace.

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.

18.1.6 Explosives Accessories Magazine

The explosives accessories magazine consists of a totally enclosed reinforced concrete structure on an engineered terrace.

Access to the explosives accessories 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.

18.1.7 Access Road to Explosives Magazine and ANFO Terrace

The access road to the ANFO terrace, explosives accessories magazine and the explosives magazine will be installed as part of the construction works on site. The road will be 6m wide and will comprise an engineered fill layer of no less than 300 mm, installed in 150 mm thick layers.

18.1.8 Main Haul Roads

The main haul roads between the plant and mine terrace and between the pit and the ROM tip ramp will be installed as part of the construction works on site. The main haul roads have been designed to accommodate 40T mine haul trucks.

The haul roads are a minimum of 15m wide and accommodate 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, min 1.5m high.

18.1.9 ROM Tip Ramp

The ROM tip ramp has been included near the pit, south east of the plant and mine terrace, to reduce the haul distance for the mining fleet.

The crushing circuit proposal incorporates a skid-mounted primary crushing plant, secondary and tertiary crushing plant and a screening plant. The ROM tip ramp has a height of 3.5 m.

The mining fleet will dump material on a stockpile adjacent to the primary crushing plant and the front-end loader will feed the crusher.

The front-end loader will be included in the plant resource as it will need to be operated as part of the plant and not as part of the mining fleet. (It will operate as per the plant operations, 24/7, not as per the mining fleet which operates one 10-hour shift, 5 days a week).

18.2 PLANT INFRASTRUCTURE

The infrastructure to support the processing plant operation was developed by DRA and positioned to optimise the operation of the plant.

The Plant infrastructure includes for:

- Plant Office
- General Engineering Services (GES) Workshop
- Laboratory
- Process Water Dam
- Raw Water Dam
- Ablution Facilities
- Reagent Stores

18.2.1 Plant Office

The plant office has been designed to accommodate 6 single offices, a reception area with one desk for one receptionist, an open plan area for 20 people (grouped in pods of 4 people each), a boardroom which will accommodate 20 people with audio / visual equipment and bathroom facilities to accommodate up to 30 people (10 female and 20 male). The plant office will also include for a small kitchenette to serve the members of staff and visitors to the mine).

18.3 GENERAL ENGINEERING SERVICES (GES) WORKSHOP

The GES will serve the plant area on site. It will be located on the plant terrace on a concrete slab which will be installed by the civils and earthworks contractor as part of the overall civil and earthworks package. The GES workshop floor area is approximately 270 m².

The GES workshop area will comprise a concrete slab surrounded on 3 sides by converted shipping container units which will be used as offices, ablution facilities and storage areas. There will be four (4) 6 m container offices and eight (8) 12 m containers which will be used for storage.

18.4 LABORATORY

The laboratory is a bought-out containerized solution which will be located on the plant terrace on concrete slabs which will be installed by the civils and earthworks contractor as part of the overall civil and earthworks package.

The laboratory setup will consist of four 12m containers that have been fitted out to accommodate the laboratory equipment and personnel. The laboratory will be serviced by 13 personnel in total (2 expat and 11 local personnel).

18.5 RAW WATER SUPPLY

A dynamic water balance model was used to quantify water demands throughout the proposed life of mine. The re-use of dirty water in mining processes was maximised to reduce raw water demands. Raw water requirements and potential variations in water demands were calculated. Maximum demands for raw water make-up are on average 561 cubic metres per day (m³/day) in the beginning of the life of mine for the processing plant. As the life of mine progresses maximum raw water make-up will decrease to 222 cubic meters per day on average (m³/day). The raw water source for the plant and potable water will be groundwater, abstracted from a well field within a 5 km radius of the proposed processing plant. Existing boreholes MMB12 and MMB13 have been verified as feasible boreholes to supply the raw water make-up.

The area covered by the 5 km radius well field is shown Figure 19 indicating existing production boreholes MMB12 and MMB13. The existing test production boreholes drilled in the area identified the marble layers to the west of the site as the most productive aquifer. The existing production boreholes have a combined safe yield of 1,322 m³ per day. The potential sustainable yield of the 5 km radius well field, based on groundwater rainfall replenishment rates and ecological water requirements, is about 2,430 m³ per day and 1,430 m³ per day taking extreme long-term drought conditions into account. The groundwater sustainable yield is substantially higher than the foreseen raw water make-up requirement.

An environmental impact assessment was undertaken for the well field with no major impacts foreseen based on the formulated Water Management Plan.

18.6 PROCESS PLANT, STORM WATER AND RAW WATER SUPPLY DAMS

A Storm Water Management Plan (SWMP) has been developed in accordance to 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 include the implementation of three Pollution Control Dams at the processing plant, Co-Disposal Facility and Material Reject Dump and a set of drains and berms. Additional measures proposed include sediment traps where drains flow into the Pollution Control Dams to reduce the amount of silt that deposits in these dams.

The operational philosophy of mining operations will be to operate the process 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. Results from the salt balance, based on information from geochemical tests and modelling, suggests that PCD water under discharge conditions will comply with Madagascar and international criteria

The ranking of water sources to be used in the ore beneficiation process, in order of priority, are 1) Sewage effluent return, 2) PCD water from storm water/seepage from plant, co-disposal, reject dump, ore stockpiles and open pit water), 3) raw water sources (Wellfield – Module 1). A raw water tank will receive water from the well field 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 and the liner will be installed on an A4 bidim geotextile protection layer.

18.6.1 Ablution Facilities

An ablution facility is provided for at the plant to accommodate the work force at the plant.

18.6.2 Reagent Stores

The reagents building is located adjacent to the plant on the plant and mine complex terrace. The building will serve as storage for the reagents required for the process plant. The building is open on all sides and will store barrels of flocculent and 220 litre drums of reagent.

18.7 SHARED INFRASTRUCTURE AND SERVICES

The shared infrastructure and services to support the processing plant operation and the mining operation was developed by Erustrat and positioned to optimise the overall mining and processing operation.

The shared infrastructure and services includes:

- Access Roads
- Terraces and Bulk Earthworks
- Gate House and Turnstile Access Control
- Training Centre
- Refuelling Station
- Waste Water Treatment plant
- Potable Water Treatment Plant
- Storm Water Drainage
- Storm Water Dams
- Consumer Sub-station
- MCC's
- Fencing
- Fire Water System, Reticulation and Storage
- Potable Water Reticulation
- Potable Water Storage Tank
- Waste Water Reticulation
- Raw Water Supply
- Welding Workshop
- Strategic Spares Storage
- Tyre Storage Yard
- General Storage Yard
- Power Generation Facility
- Fuel Storage Facility

18.7.1 Access Roads

The access roads to be installed include the road to the plant and mine site, the road to the explosives magazine, the road to the ANPHO terrace and the accessories magazine.

The layer works are to be installed in 150 mm thick layers and the road will be 6 m wide with 0.5 m wide shoulders (7 m width in total). The access roads are designed with a 1:50 fall across the road from the centreline to the edge of the shoulder.

No provision is included for access roads on the terraces. Vehicle will drive on the finished terrace surface, there is no provision for surfacing of areas demarcated for roads on the terraces.

Culvert crossings have been allowed for as required along the new access roads to allow for overland water flow to migrate under the roads and continue on with the natural drainage courses.

18.7.2 Terraces and Bulk Earthworks

The main terraces for the works include:

- Plant and Mine Construction Camp
- Permanent Camp
- Surge Bin
- Crushing Circuit

Terraces will be installed as part of the works and will be installed at a minimum of 300 mm above natural ground level in order to reduce the risk of flooding in the rainy season.

The total volume of cut and fill for the project is detailed in Table 40 below.

TABLE 40: EXCAVATION, FILL MATERIAL AND CRUSHING / PROCESSING SUMMARY

Description	Fill Volume (m ³)	Excavation Volume (m ³)
Excavation for Pipes etc		55 011
Hard Rock Excavation for Pipes etc		33 070
Excavation for Civils		8 257
Bulk Excavation		80 208
Hard Rock Excavation for Civils		4 494
Hard Rock Excavation		545 153
Backfill for Pipes etc	12 001	
Backfill (G7)	416 641	
Bulk Excavation at TSF	450 623	
Crushing/Processing	383 030	
Backfill at TSF (Red borrow material)	151 250	
Trench backfill with Aggregate at TSF	13 600	

18.7.3 Gate House and Turnstile Access Control

The gate house and turnstile access to the site will be provided by two 12 m containers that have been fitted out as an office and search facilities and a turnstile access unit.

18.7.4 Training Centre

The training centre has been designed to accommodate 6 single offices, a boardroom to seat 50 people and bathroom facilities to accommodate 50 people.

18.7.5 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 refuelling 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.

18.7.6 Waste Water Treatment Plant

Provision has been included for two waste water treatment plants on site, one at the plant and mine complex and one at the permanent accommodation camp.

The discharge requirements for the waste water treatment plants to be installed on site are in line with the requirements as set out by the Department of Water Affairs (DWA) General Limits, South Africa.

The unit for installation at the permanent camp will initially be utilised for the construction camp operation during construction and will be relocated to the permanent accommodation camp location prior to commissioning of the plant.

The unit has been designed to accommodate the 800 people (120 m³/day) in the construction camp and has been configured in a modular arrangement where one of the two modules can be decommissioned following the construction phase and the plant will be downsized to accommodate the 340 permanent personnel (54 m³/day) at the mine and plant. For the camp, it has been assumed that the water consumption will be 150 litres / person / day for the construction and the operational phase of the project.

The unit for installation at the plant and mine complex will be installed and commissioned as part of the construction works phase of the project. The unit will serve all the temporary ablution facilities on site during the construction phase of the works as there is no waste water treatment facility in the surrounding region of the site where waste water can be discharged.

The unit has been designed to accommodate the 800 people (56 m³/day) in the construction phase and 340 permanent personnel (54 m³/day) during the operation phase of the project. It has been assumed that the water consumption will be 70 litres / person / day for the construction and 150 litres / person / day during the operational phase of the project.

Drying beds have been included for the discharge and drying of the sludge from the units on an annual basis. These concrete drying beds are designed and installed as part of the main earthworks and civil package.

18.7.7 Potable Water Treatment Plant

Provision has been included for two potable water treatment plants on site, one at the plant and mine complex and one at the permanent accommodation camp.

The potable water treatment plants have been designed to accommodate a varying capacity throughput with the use of VSDs on the pumps during the operational phase of the project.

The two plants have been identically sized for the plant and the mine complex and the accommodation camp. The plant to be installed at the permanent camp will initially be installed at the construction camp and will later be relocated to the permanent camp location.

The plants are designed to provide 7.5 m³/h to support the 800 people in the construction camp and at the plant and mine complex. It has been assumed that each individual on site will require 225 litres / day.

Raw water will be pumped to the two treatment plants from the raw water dam located adjacent to the plant terrace. Treated water will be stored in the potable water storage tanks which will be located on the plant and mine complex and the permanent accommodation camp terraces. The water storage tanks each have a capacity of 516 m³.

18.7.8 Storm Water Drainage

Storm water run-off will be diverted around the plant - 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 and mine terrace, open pit, stockpile areas, material reject dump and the co-disposal facility is considered dirty water and as such, is collected in the storm water dams or Pollution Control Dams (PCDs) located on site and discharge to the environment is limited. All the PCDs on site have been sized to accommodate a minimum of a 24 hour, 1:10 year return period storm event. Discharge from the PCDs will, however, occur and will need to be carefully controlled and monitored. Results from a daily salt balance model, based on information from geochemical tests and modelling, suggests that PCD water under discharge conditions will comply with Madagascar and international quality criteria.

18.7.9 Storm Water Dams

The storm water dam or PCD located adjacent to the processing plant has been sized to accommodate all the storm water run-off from the plant and ore stockpile area. The dam has been sized to accommodate the 24 hour, 1:20 year return period storm event and will accommodate 8,000 m³ of storm water and return water from other storm water dams (if required).

The storm water dam or PCD located adjacent to the material reject dump and open pit, has been sized to accommodate all the storm water run-off from the material reject dump area and water from the pit dewatering operation. This dam will accommodate 5,000 m³ of storm water and pit water. This dam was designed to discharge on a regular basis taking the predictive

water quality analysis results into account, complying with Madagascar and international quality criteria.

The storm water dam located adjacent to the co-disposal facility, has been sized to accommodate all the storm water run-off from the co-disposal facility. The dam has been sized to accommodate the 24 hour, 1:10 year return period storm event and will accommodate 30,000 m³ of storm water.

18.7.10 Consumer/Incomer Substation

The consumer sub-station will be a weatherproof pre-fabricated modular unit and installed on concrete bases and plinths with steel access stairways and landings. Access will be via the ends of the substation 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.

The 2 x 1.4 MW, 3-ph, 400Vac diesel generators shall be controlled and connected to this substation, rated at 4,500A, 35kA either being stand-alone or part of the generator panels.

From this Consumer substation, the power shall be distributed at 400Vac, via copper cables, to the 3-x respective plant MCCs (Crusher, Mill & Floatation MCCs) located strategically in the plant, from where the plant loads and some infrastructure shall be supplied.

18.7.11 Plant Substations

The plant substations shall will be a pre-fabricated and installed on concrete bases and plinths with steel access stairways and landings. Access will be via the ends of the substation 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 3 plant substations shall be powered form the Consumer Substation, each being of Form 1 separation, rated at minimum of 800A, 35kA, 3-ph, 400Vac, and be of a Form – 1 isolation & construction format.

It is envisaged that the Crusher (MCC11), Mill & Floatation (MCC21 & 22) MCCs placed strategically in order to be closest to the loading, reducing in cable runs and installation costs.

18.7.12 Transformers

As the generators excite at 400Vac + N + E, no distribution transformers are required on this plant, other than that required by control and instrumentation.

18.7.13 Fencing

The project site is located near the village of Fotadrevo (approx. 10 km by road) and smaller local villages (within 2 km of the site) and as such, the tailings storage facility, plant and mine complex, permanent 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 2.4 m high diamond mesh security fencing with 3 strands of galvanised barbed wire to the top of the fence. 6 m wide access gates together with pedestrian access gates 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.

18.7.14 Fire Water System, Reticulation and Storage

The fire water system for the plant 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 such that the fire water requirements are not reduced / utilized except for firefighting 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.

18.7.15 Potable Water Reticulation

Potable water for the project will be supplied via the potable water treatment plants which are fed from the raw water dam. A well field will be installed which will provide all the raw water requirements for the project, including all process, raw and potable water requirements.

The potable water reticulation system will include a network of buried uPVC pipes installed at the permanent camp and the plant and mine complex. This network of pipes will be fed from the potable water supply storage tanks at the plant and mine terrace and at the permanent camp terrace.

Each storage tank will have a booster pump included in order to ensure adequate pressure is maintained throughout the water supply network on site.

Applications for water extraction permits and permitted water usage will be undertaken at execution stage by the Company with the assistance of GCS and Environment.

18.7.16 Potable Water Storage Tank

Provision has been included for two potable water storage tanks, one located at the permanent accommodation camp and one at the plant and mine complex. Both tanks have been sized to accommodate 516 m³ of water which will provide for 3 days' supply in the event of a failure of the raw water supply system or the potable water treatment plant. It has been assumed that each person on site will require 225 litres / day. For a peak of 800 people on site, the daily requirement for the construction camp will be 180 m³.

18.7.17 Waste Water Reticulation

The waste water reticulation will comprise a buried network of uPVC pipes, manholes and 2 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 plants.

18.7.18 Welding Workshop

A welding workshop has been provided for which will support both the mining operation and the process plant operations. The workshop includes for two (2) 12 m shipping containers which will be refurbished and used as offices and storage. A concrete slab is included at the welding workshop and it is proposed that works will be undertaken in the open air. There is no provision of a roof structure at the workshop. Heavy lifting and crange will be supplied by the mobile plant purchased for the Project by the Company, no separate provision has been included for crange at the welding workshop.

18.7.19 Strategic Spares Storage

A strategic spares storage area has been provided for which will support both the mining operation and the process plant operations. The storage area includes for two (2) 12 m shipping containers which will be refurbished and used as offices and storage.

18.7.20 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. Provision is included for security fencing at this location. An area of 320 m² has been allocated for the tyre storage area.

18.7.21 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 spare's storage area. Provision is included for security fencing at this location. An area of 350 m² has been allocated for the general storage area.

18.7.22 Bulk Fuel Storage Facility

During the operational phase of the project, the bulk fuel storage system will consist of five (5) containerised, double-walled fuel storage tanks, with sufficient storage capacity for two 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 in order 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 litres) in two (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.



FIGURE 81: TYPICAL LAYOUT OF CONTAINERIZED FUEL STORAGE TANK

18.8 POWER GENERATION FACILITY

The objective of the Electrical, Control & Instrumentation scope is to provide independent power to the stand-alone graphite plant, for Phase 1 of the Molo Mine.

Independent power shall be supplied by diesel generators, whereas the control & automation shall deliver an integrated state of the art control and data acquisition solution.

The total installed generation capacity equals to 2.8 MW. Power to the plant and infrastructure shall be supplied by 2 x 1.4 MW, 3-ph, 400Vac diesel generators, interlocked and operating either in parallel or independently. Power to remote areas, such as the water wells shall be supplied via independent stand-alone 15kVA, 3-ph, 400Vac diesel generators.

Subject to the sequential starting of the larger loads for e.g. the Mill, it might be required during an ore loaded start-up, that both generators run in parallel. Once the plant has reached equilibrium the standby generator shall be stepped back. During continuous normal operation of the plant, one generator will suffice. The second unit shall primarily be used during maintenance and operational rotations.

18.8.1 Generator Loading

A bottom up estimating methodology was used to arrive at a predicted Notified Maximum Demand (NMD) and electrical consumption (excluding standby loads) for the proposed installation, taking into consideration, motor efficiencies, circuit types (for e.g. VSD or DOL) actual load duties and utilisation factors.

Typically, the plant energy requirement is calculated in combination with the plant process philosophy, start-up procedure as well as that presented in the mechanical equipment list. The infrastructure loading includes power to the Water Purification plant, Sewerage plant, Administration & Plant Office, Workshop, Wash bay, Stores, Laboratory, 100-Man Change House, Clinic/Dressing Station, Gate House and plant general lighting.

The static electrical loading on one 1.4 MW, 3-ph, 400Vac diesel generator is calculated to be equal to 88% of which the remainder of ± 0.15 MW is allocated to infrastructure.

The running electrical loading including the infrastructure on one 1.4MW, 3-ph, 400Vac diesel generator is calculated to be equal to 76%.

a) Power Demand Summary

Table 41 below indicates the breakdown of the major values pertaining to the electrical power supply as well as the loading of the generators.

TABLE 41: POWER DEMAND SUMMARY

Load Centre	Generator			Connected % Loading [Excluding Standby]	Dynamic Variables [No Power Factor Correction]			
	kW Rating	Full Load Current [A]	LV Voltage (kV)		Estimated Run Current	% Volt drop with Largest Motor starting	% Volt drop steady state	% Running Load
SINGLE GEN	1,400	2,127	0.380	88%	1,219	3.68%	2.00%	65%

During normal operation, there will only be one unit running, with a second allowed to assist with mill starting, if required.

18.8.2 Control & Instrumentation

18.8.2.1 Plant Control System

The plant control system will consist of a PLC, Remote I/O and SCADA based system, typically as indicated in Figure 82 below.

Due to the conductivity nature of the graphite it is envisaged that the Remote I/O panels be installed in the control room, with the minimum Junction boxes in the field.

Communication in the field and Remote I/O panels shall be Profibus-DP whereas communication amongst the PLC, SCADA & Engineering systems shall be done on Ethernet. All instruments and field I/O shall be hard wired to the respective Remote I/O panels.

EXECUTION PLAN

CONTROL ARCHITECTURE

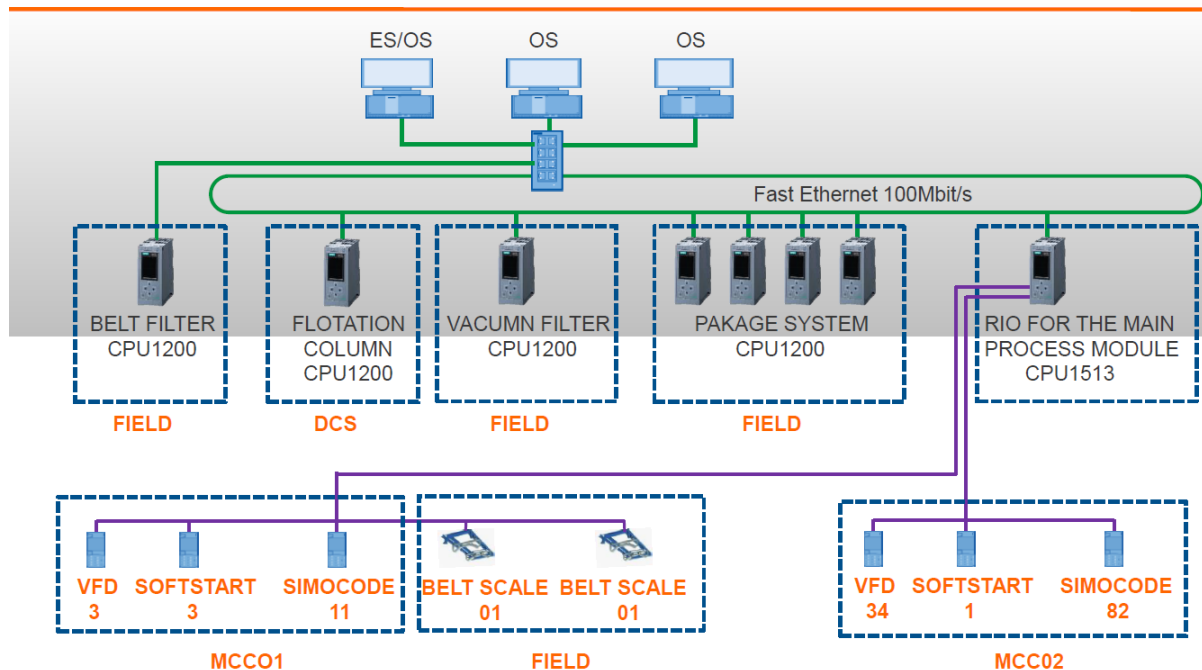


FIGURE 82: PLANT CONTROL ARCHITECTURE

All motor control, valve control, interlocking, PID control and sequence start/stops will be controlled via the PLC control system. The PLCs and SCADAs will communicate over an Ethernet network. Managed network switches will be used to minimize network collisions and to help guarantee the data throughput.

18.8.2.2 UPS Power

All PLC's, SCADA's and network equipment will be powered from UPS power.

Provision is be made for a minimum of 30 min backup time at 100% load. The UPS (If Required) in the main Server Room should allow a minimum of 60 min backup time at 100% load.

18.8.2.3 Level of Automation

All equipment must be controlled from the control room unless specified otherwise as a 3-level control mode will be implemented, namely Maintenance, Interlocked and Sequence modes.

Maintenance

This mode is for maintenance only and all process interlocks will be bypassed. Individual drives can be started and stopped in the field.

Interlock

The plant is controlled from the SCADA using the individual Stop/Start buttons on the SCADA. All process interlocks will be active.

Sequence

In sequence mode, the plant will be started and stopped using the PLC sequence logic.

The field emergency stop buttons are hardwired to the MCC and can be used in any mode.

All the modes will be PLC controlled.

18.8.2.4 Local Isolator and Start/Stop Stations

All motors will have a local isolator and stop/start station installed as close as possible to the motor. The stop/start station will consist of emergency-stop and maintenance-start buttons.

18.8.2.5 Flow Measurement

There will be magnetic flow measurement of the potable water and industrial water. Flow rates and totalised flows will be monitored and recorded on the SCADA system in accordance with the requirements.

18.8.2.6 Level Measurement

Ultrasonic type level probes will be used for liquid levels. Still-wells should be installed where required. Additional low-low level-switches (if required.) will also be installed on bins where liner protection is required.

18.8.2.7 Scale Weight-Meter

The scales will be installed on a horizontal section. The required accuracy should dictate the type of scale required. A Profibus interface is required on all scales.

18.8.2.8 Densitometers

The densitometers to be installed shall be of the nuclear type.

18.8.2.9 Pressure Transmitter

The pressure transmitters to be installed shall be of the conventional appropriate for the typical application.

18.8.2.10 Control Valves

Control valves will be of the motorized type with position feedback. No pneumatic control valves are considered.

18.8.2.11 Solenoid Valves

All solenoid valves shall be 110VAC powered.

18.9 TAILINGS DISPOSAL AND STORAGE FACILITY

Due to the substantially reduced tonnages for the project as envisaged, tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining. Despite this co-disposal approach, a detailed design has been completed, complete with environmental and social impact assessment and closure to allow for the upgrade to a more conventional, cyclone facility, should the throughput be increased during the life of the mine. This approach has been pursued to ensure that sufficient flexibility is built into the project development strategy to accommodate the anticipated increase in market demand.

18.10 GEOTECHNICAL REPORT ASSESSMENT AND FOUNDING STRATEGY

A geotechnical investigation for the revised plant site and the camp site for the Molo Graphite Mine project was conducted. The works included the excavation of 23 test pits, six DCP tests and the drilling of 11 rotary core boreholes. The fieldwork component for the camp site involved the mechanical excavation of 10 test pits and the advancing of five DCP tests.

The bedrock encountered in the test pits and boreholes was variable and ranged from extremely soft rock to hard rock strengths with variable weathering and jointing. The expected allowable bearing capacities for the varying hardness of Gneiss bedrock encountered across the site are summarised below:

TABLE 42: EXPECTED ALLOWABLE BEARING CAPACITY FOR ENCOUNTERED GNEISS BEDROCK

Gneiss Bedrock	Allowable Bearing Capacity (kPa)
Completely weathered, foliated, Extremely Soft Rock	300
Very closely to moderately closely jointed, highly to slightly weathered, Very Soft Rock	500
Very closely to moderately closely jointed, highly to slightly weathered, Soft Rock	700
Very closely to closely jointed, medium to slightly weathered, Medium hard to Hard Rock	1000

As rock is located at a shallow depth at most locations it is recommended that structures generally be founded on rock rather than the overlying thin soils. However, light structures with loads of less than 100 kPa could be founded on the soils if necessary.

All temporary excavations in excess of 1.5 m must be fully supported or battered back to at least 1v:1.5h in soil and 2v:1h in rock. For excavations over 3.0 m in depth, the soil and rock conditions should be assessed by a geotechnical engineer or engineering geologist prior to and during excavation and the safe slope angles determined by analysis.

The transported and residual soils classify as G8 and G7 materials respectively and the very soft rock classifies as G6 to G7 class materials. These materials are however thin and of insufficient volumes to be considered as significant sources of construction materials for terrace construction.

The sandy lean CLAY (CL) material present approximately 6 km away from this site at the TSF site has an estimated available volume of 525 000 m³ of this material. The strength of these materials may be increased significantly with the addition of cement and/or lime and may then be suitable as terrace construction materials. Further laboratory testing will be required to determine this.

Due to the frequent rock outcrop and boulders present across the site, surfaces will need to be prepared by a bulldozer prior to trafficking.

Construction fill material for all terrace construction is sourced from the excavation for the return water dam. Approximately 450,000 m³ of material is to be excavated from the return water dam and approximately 383,000 m³ of this excavated material is to be crushed by means of mechanical jaw crushing. The total volume of material required for terraces is approximately 416,000 m³.

Concrete aggregate material for all concrete works on site will be sourced from borrow pit (s) on site, the material will be crushed using a cone crusher, supplied by the Company and operated by the earthworks and civil contractor, to produce 19 mm concrete aggregate. The total volume of concrete aggregate is approximately 4,500 m³ (5,800 m³ of concrete).

19 MARKET STUDIES AND CONTRACTS

19.1 GRAPHITE OVERVIEW

Graphite is highly refractory with a melting point of 3927°C, is chemically inert, is a superior lubricant, and is one of the most electrically and thermally conductive nonmetals (Taylor, 2006; Moores et al., 2012; Olson, 2014). The carbon atoms in graphite are densely arranged in parallel-stacked, hexagonal honeycomb-lattice sheets, so graphite shows perfect basal cleavage (Olson, 2014). Graphite can be manufactured artificially in electric furnaces from petroleum coke but is more cost-effectively extracted from naturally occurring deposits (Taylor, 2006). Natural graphite is mined from three types of deposit: 1) vein (also known as lump or chip), 2) amorphous, and 3) disseminated flake (Benchmark, 2017a; Robinson et al, 2017). Olson (2017) estimated the overall natural graphite market to be 1.2 million tons per annum (“tpa”), with the flake market accounting for 55%, amorphous accounting for 44%, and vein accounting for less than 1% (Industrial Minerals, 2017).

19.2 FLAKE GRAPHITE MARKET

Natural flake graphite is used in industries requiring high temperature refractory materials such as high temperature bricks and linings utilized in metal production, ceramics, petrochemical and cement industries (Moores et al., 2012; Industrial Minerals, 2017). It is also used to dissipate heat in consumer electronics, as a gasket material for highly corrosive chemical environments, as a lubricant, as a liner in high performance brakes, as a carburizer in steel

production, and as a precursor for the production of graphene (Moores et al., 2012; Benchmark, 2017a; Industrial Minerals, 2017). Natural flake graphite is mined predominantly in China (Robinson et al., 2017), with over half of global supply coming from the Heilongjiang, Shandong and Inner Mongolia provinces (Benchmark, 2017a). Outside of China, Brazil is the dominant flake graphite producer, followed by India, North Korea, Canada and Madagascar (Moores et al., 2012; Industrial Minerals, 2017; Figure 83). Minor production also comes from Sri Lanka, Norway, Sweden, Ukraine and Russia (Olson, 2017).

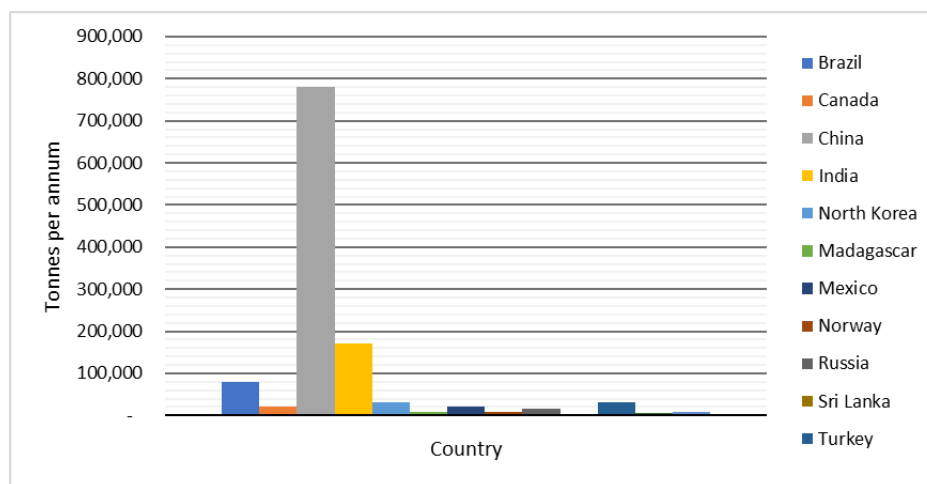


FIGURE 83: NATURAL GRAPHITE PRODUCTION VOLUMES BY COUNTRY (DATA FROM OLSON, 2017)

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 due to its role as the primary anode component in lithium-ion batteries (Benchmark, 2017a). According to Industrial Minerals (2017), the natural and synthetic graphite industries have little crossover in market share and end-uses with the exception of lithium-ion battery production where they are both used as anode material.

With the advent of the electrification of vehicles, it is forecast in the next 5 years that flake graphite's number one use will be in battery applications, overtaking its traditional industrial uses. As illustrated in Figure 84, the current natural flake graphite market is roughly 645,000 tons per annum with less than 1/5 of that volume being utilized for battery anode material (Clarke, 2017).

**Current natural flake graphite market
(645,000 tons)**

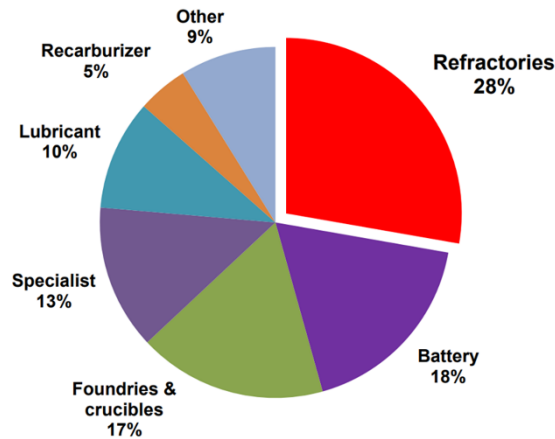


FIGURE 84: 2016 PRODUCTION OF 645,000 TONS OF NATURAL FLAKE GRAPHITE (CLARKE, 2017)

Analysis by the International Energy Agency (2017) quantified that globally, the total number of electric vehicles (“EVs”) reached two million in 2016, with China accounting for 40% of sales. Currently, lithium-ion anode material represents 18% of the flake graphite market but this number is expected to rise dramatically, with the Organization of Petroleum Exporting Countries (2017) projecting the total number of electric vehicles to reach 80 million by 2040. Oil producer BP Global (2017) has an even more bullish prediction of 100 million EVs by 2035, while the Chief Commercial Officer of mining giant BHP (Reuters, 2017) has predicted this total to reach 140 million by 2035. Benchmark (2017b) has predicted that the amount of flake graphite feedstock required to supply the lithium-ion anode market will grow from a current total of 120,000 tons per annum to 1,250,000 tons per annum by 2025 (Figure 85).

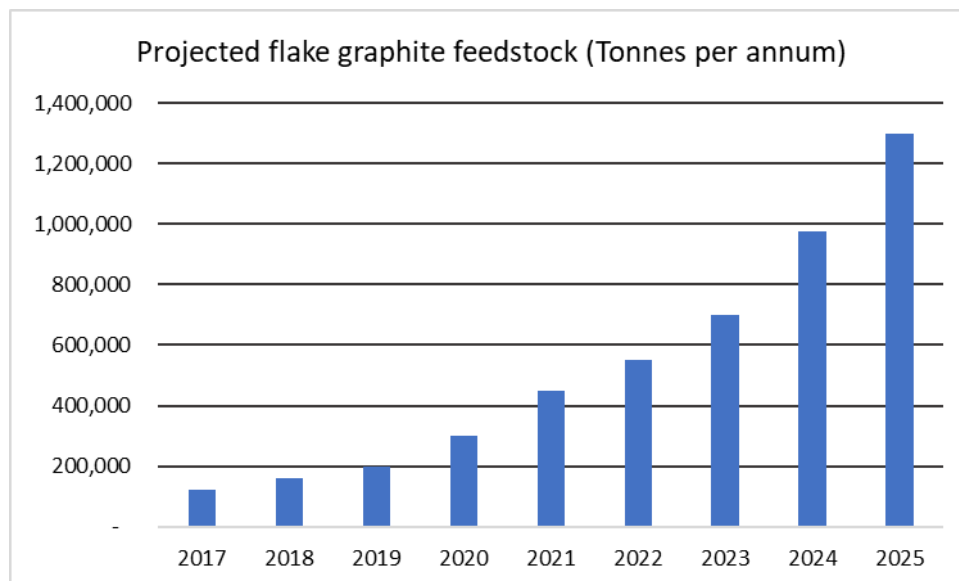


FIGURE 85: FLAKE GRAPHITE FEEDSTOCK REQUIRED TO SUPPLY THE LITHIUM-ION ANODE MARKET IS PROJECTED TO GROW FROM 120,000 TONNES PER ANNUM IN 2017 TO 1,250,000 TONNES PER ANNUM BY 2025 (BENCHMARK, 2017B).

19.3 FLAKE GRAPHITE PRICING

As an industrial mineral, flake graphite pricing is determined by three factors: 1) flake size, 2) carbon purity and 3) industry-specific technical attributes of the flakes (Benchmark, 2017a; Roskill, 2017). Flake sizing is broadly classified into four ranges: small (-100 mesh, or <75µm) medium (-80 to 100 mesh, or 75µm to 180µm), large (-50 to 80 mesh, or 180µm to 300µm), and extra-large or jumbo (+50 mesh, or >300µm). These flake sizes are in turn classified by carbon content ("C"), and are typically sold in ranges of 88-93% 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. There are however subscription pricing services that provide monthly graphite pricing for various flake sizes and carbon purities based upon input from graphite purchasers. Figure 86 identifies the average monthly flake graphite pricing for the past 12 months as provided by Benchmark Mineral Intelligence (Benchmark, 2019).

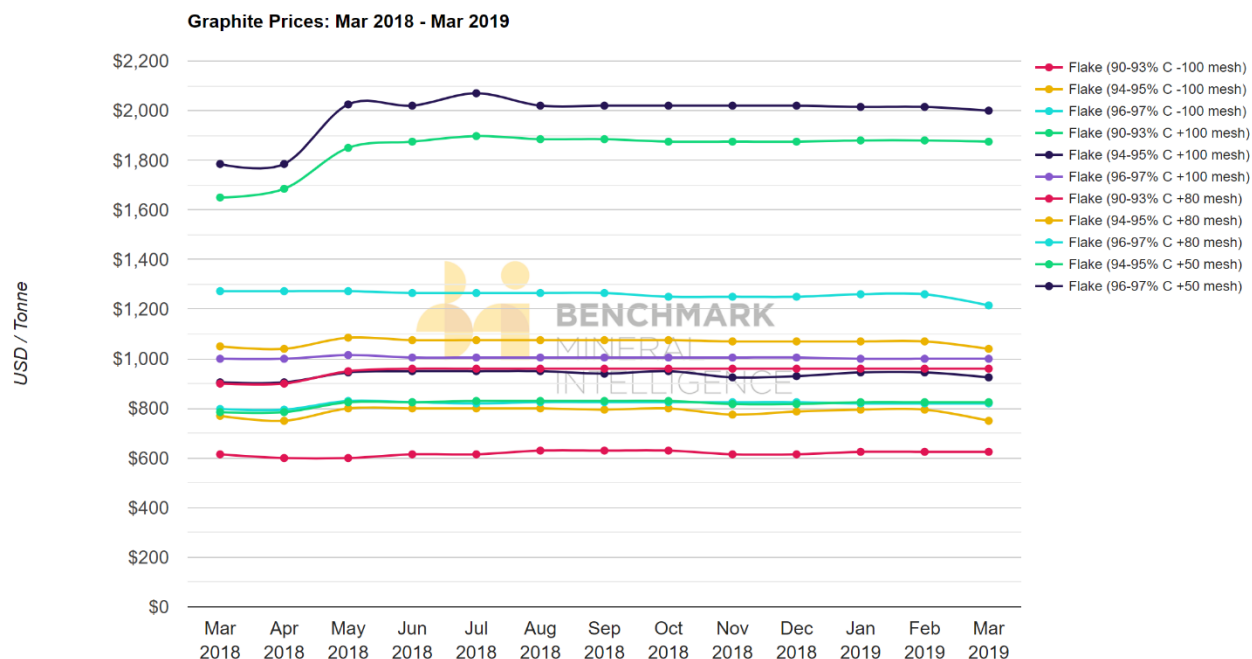


FIGURE 86: MONTHLY FLAKE GRAPHITE PRICING FOR VARIOUS FLAKE SIZES AND CARBON CONTENTS (BENCHMARK, 2019).

As Table 43 illustrates, the final flake graphite concentrate from the Molo deposit metallurgical work yielded material ranging from 96.9% C to 98.1% C.

Table 43 summarizes FOB China flake graphite pricing from Benchmark (2019) over the past 12 months for material with a carbon content ranging between 96-97% C.

TABLE 43: 12 MONTH FLAKE GRAPHITE PRICING WITH CARBON CONTENTS BETWEEN 96-97% C AND ON A FOB CHINA BASIS.

Date	+50 mesh			+80 mesh			+100 mesh			-100 mesh		
	PriceHigh	PriceLow	PriceMid	PriceHigh	PriceLow	PriceMid	PriceHigh	PriceLow	PriceMid	PriceHigh	PriceLow	PriceMid
18-Mar	\$ 1,850	\$ 1,720	\$ 1,785	\$ 1,300	\$ 1,245	\$ 1,273	\$ 1,080	\$ 920	\$ 1,000	\$ 825	\$ 770	\$ 798
18-Apr	\$ 1,850	\$ 1,720	\$ 1,785	\$ 1,300	\$ 1,245	\$ 1,273	\$ 1,080	\$ 920	\$ 1,000	\$ 820	\$ 770	\$ 795
18-May	\$ 2,150	\$ 1,900	\$ 2,025	\$ 1,300	\$ 1,245	\$ 1,273	\$ 1,080	\$ 950	\$ 1,015	\$ 890	\$ 770	\$ 830
18-Jun	\$ 2,140	\$ 1,900	\$ 2,020	\$ 1,290	\$ 1,240	\$ 1,265	\$ 1,065	\$ 945	\$ 1,005	\$ 880	\$ 770	\$ 825
18-Jul	\$ 2,140	\$ 2,000	\$ 2,070	\$ 1,290	\$ 1,240	\$ 1,265	\$ 1,065	\$ 945	\$ 1,005	\$ 870	\$ 770	\$ 820
18-Aug	\$ 2,140	\$ 1,900	\$ 2,020	\$ 1,290	\$ 1,240	\$ 1,265	\$ 1,065	\$ 945	\$ 1,005	\$ 880	\$ 770	\$ 825
18-Sep	\$ 2,140	\$ 1,900	\$ 2,020	\$ 1,290	\$ 1,240	\$ 1,265	\$ 1,065	\$ 945	\$ 1,005	\$ 880	\$ 770	\$ 825
18-Oct	\$ 2,140	\$ 1,900	\$ 2,020	\$ 1,300	\$ 1,200	\$ 1,250	\$ 1,065	\$ 945	\$ 1,005	\$ 880	\$ 770	\$ 825
18-Nov	\$ 2,140	\$ 1,900	\$ 2,020	\$ 1,300	\$ 1,200	\$ 1,250	\$ 1,065	\$ 945	\$ 1,005	\$ 880	\$ 770	\$ 825
18-Dec	\$ 2,140	\$ 1,900	\$ 2,020	\$ 1,300	\$ 1,200	\$ 1,250	\$ 1,065	\$ 945	\$ 1,005	\$ 880	\$ 770	\$ 825
19-Jan	\$ 2,130	\$ 1,900	\$ 2,015	\$ 1,280	\$ 1,240	\$ 1,260	\$ 1,055	\$ 945	\$ 1,000	\$ 870	\$ 770	\$ 820
19-Feb	\$ 2,130	\$ 1,900	\$ 2,015	\$ 1,280	\$ 1,240	\$ 1,260	\$ 1,055	\$ 945	\$ 1,000	\$ 870	\$ 770	\$ 820
19-Mar	\$ 2,100	\$ 1,900	\$ 2,000	\$ 1,280	\$ 1,150	\$ 1,215	\$ 1,055	\$ 945	\$ 1,000	\$ 870	\$ 770	\$ 820

Using the flake size distribution arrived at from metallurgical testing (Table 26) with the average pricing as identified in Table 43, yields a 12 month average “basket price” of US \$1207.55 for Molo graphite as per Table 44 below.

TABLE 44: AVERAGE 12 MONTH (MARCH 2018 THROUGH MARCH 2019) FLAKE GRAPHITE PRICING FOR MOLO DISTRIBUTION.

Yield (%)	Microns	Mesh Size	C%	Sale \$	\$/T
23.60%	>300 µm	+50 mesh	96.90%	\$1,985.77	\$468.64
22.80%	180-300 µm	+80 mesh	97.00%	\$1,258.65	\$286.97
6.90%	150-180 µm	+100 mesh	97.20%	\$1,003.85	\$69.27
46.70%	<75 µm	-100 mesh	97.60%	\$819.42	\$382.67
100%			97.20%		\$1,207.55

19.4 CONTRACTS AND AGREEMENTS

The Company announced the signing of an offtake agreement with a prominent Japanese trading company (“Japanese Partner”) to purchase 20,000 tonnes per annum of Molo graphite for use in battery anode applications for electrical vehicles (“EVs”) on October 16, 2018. The Japanese Partner is a major supplier of flake graphite to Japan’s largest battery processor and manufacturer of graphite anode material in lithium ion batteries (“LiB”) for electric vehicle applications and has verified that Molo graphite meets or exceeds their current technical specifications. This battery anode manufacturer currently has over 50 percent market share of premium-grade graphite anode production in Japan for electric vehicle battery applications, and its EV automotive customers are global. To protect certain confidential aspects of the Offtake Agreement, the Japanese Trading Company and the Japanese electric vehicle anode producer have requested not to be identified at this time.

The Offtake Agreement is for a period of ten (10) years and activates on the commencement of commercial production at the Project, with an automatic renewal for an additional five (5) years. The Japanese Partner will have the exclusive right to import and sell graphite concentrate produced from the Project in Japan. Provided that commercial production commences within 3 years, following the ramp up period, the Japanese Partner will purchase 20,000 tonnes of SuperFlake® graphite per annum. Product prices will be negotiated on a per order basis between the parties and will be based on FOB market pricing.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

20.1 DESCRIPTION OF AVAILABLE INFORMATION

The location of this Project, specifically within a French speaking country, and the common intricacies and nuances associated with projects of this nature resulted in the establishment of a partnership between a South African based Environmental consultancy firm, GCS Water and Environment (Pty) Ltd (GCS) and a Malagasy consultancy firm, L'Agence d'Exécution des Travaux d'Intérêt Public et d'Aménagement communément appelée (Agetipa).

The availability of current and accurate environmental, social and permitting information on the Project is substantial. Table 45 provides a summary of the information which complements this Report.

TABLE 45: INFORMATION SUMMARY

Title	Author	Year
Environmental and Social Baseline (initial)	Agetipa	2011 - 2012
Environmental Legal Review	GCS	2013
Environmental and Social Sensitivity Study (spatially integrated)	GCS	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	2017
Environmental and Social Impact Assessment and Management Plan (ESIA & ESMP) including residual baseline	GCS and Agetipa	2017
Relocation Action Plan (RAP)	Agetipa	2017
Conceptual Closure Plan	Globesight	2017
Permitting and Stakeholder Register	Environmental Law Group	2017

20.2 APPLICABLE LAWS AND STANDARDS

This Molo 2019 720ktpa (Phase 2) FS was undertaken in accordance with Malagasy national legislation, Equator Principles, World Bank and IFC requirements on environmental, cultural, health and safety protection.

20.3 ENVIRONMENTAL AND SOCIAL SENSITIVITIES

The studies preceding the compilation of the ESIA revealed no fatal flaws, or land use restrictions in terms of environmental and/or social baseline conditions and all significant sensitivities were incorporated into the planning and design of the Project.

Various options were considered at commencement of the FS regarding the geographical placement of infrastructure, type and volume of water supply, type of electricity supply, tailings storage facility, (location and deposition strategy) and product transportation routes and shipping locations, specifically related to environmental and social benefits. Refer to Figure 87 for a depiction of the environmental sensitivities and final placement of infrastructure. The following final base case and alternatives have been selected and are relevant to this Molo 2019 720ktpa (Ph 2) FS:

20.3.1 Geographic Placement of Infrastructure

Sensitive areas were superimposed onto the Project design and the placement of infrastructure such as roads, processing plant, dams, TSF, camp and other applicable ancillary

infrastructure. Placement was undertaken in such a manner as to not have any significant impacts on the environment or on the livelihoods of local inhabitants.

20.3.2 Water Supply

A detailed water study was completed for this report by GCS Water and Environmental Consultants.

20.3.3 Electricity Supply

Base case: Diesel generators.

Alternative: HFO (Heavy Fuel Oil) generators.

A trade-off study was undertaken to assess hybrids such as Diesel / HFO with Solar Photovoltaic (PV) and Diesel / HFO with Concentrated Solar Power (CSP), but none of these proved commercially feasible.

20.3.4 Tailings Storage Facility (TSF)

Due to the substantially reduced tonnages for the Project as envisaged, tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining. Despite this co-disposal approach, a detailed design has been completed, complete with environmental and social impact assessment and closure to allow for the upgrade to a more conventional, cyclone facility, should the throughput be increased during the life of the mine. This approach has been pursued to ensure that sufficient flexibility is built into the project development strategy to accommodate the anticipated increase in market demand.

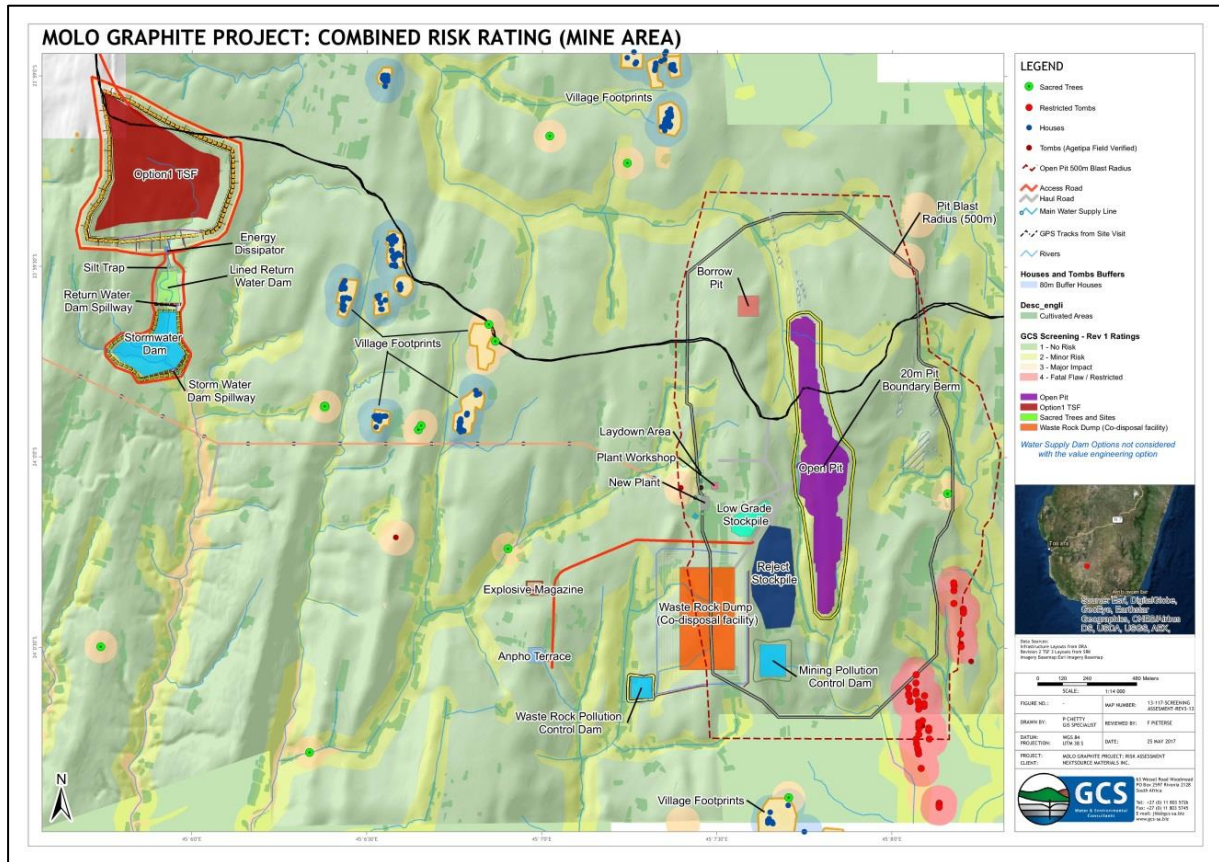


FIGURE 87: INFRASTRUCTURE OVERLAID ON ENVIRONMENTAL AND SOCIAL SENSITIVITIES

20.3.5 Transport Route and Shipping Port

The base case shipping port is Fort Dauphin via the RN10 and/or RN13. Refer to Figure 88.

Three shipping port sites were originally considered. These were Tulear, Soalara (Antseraky) and Fort Dauphin. Information obtained via the detailed Transport and Logistics Study indicated that weight limits on the Tongobory Bridge over the Unilahy River would render the transport of product to Tulear and Soalara (Antseraky) unviable. This route will in all likelihood be used to bring fuel to the proposed project site. The only remaining feasible option is Fort Dauphin via the RN10 and/or RN13.



Integrating knowledge of environmental and social sensitivities and risks into project planning and design ensured that the final impact assessment component revealed that there are no fatal flaws from an environmental and social perspective. The significance levels of impacts range from minor to major before any mitigation measures are applied and from minor to average with mitigation measures included. Notably all major risks require significant reduction in risk via stringent controls. These controls have been incorporated into the project design and planning with additional operational controls specified within the various environmental and social management plans.

In addition, each specific environmental and social component has a prescribed monitoring plan which should be followed during each project development phase. This is aimed at monitoring compliance against various specifications such as the baseline environment and predicted impact removal and reduction measures.

Next Source Materials Inc
Molo 240ktpa Feasibility Study - 2019
Reference: CMGDM1770-TRPT-001 Rev 0

20.4.1 Preparation Phase

All potential impacts are rated from minor to average post mitigation. The preparation phase will give way to the construction phase, during which certain potential impacts will continue due to the nature of large-scale construction, noting that the potential impacts would be of short duration mostly. Soil compaction at and around the open pit area is seen as a potential average impact and the proposed mitigation measures should be adhered to.

20.5 CONSTRUCTION PHASE

All potential impacts are rated from minor to average post mitigation. The most notable potential impact during this phase will be the construction of the Waste Rock / Dry Tailings Co-Disposal Facility and potentially the Tailings Storage Facility (TSF), which will have a visual change impact. The proposed mitigation measures are aimed at concurrent rehabilitation of the Co-Disposal and TSF, yet this can only commence once the construction phase gives way to the operational phase. Thus, the impact will be noticeable, yet of limited duration until such time as the side slope re-vegetation commences.

20.6 OPERATIONAL PHASE

The operational phase's potential impacts are rated from minor to average post mitigation. The most notable potential impact during this phase will again be the development of the Waste Rock / Dry Tailings Co-Disposal and the TSF, which will have a visual change impact. The proposed mitigation measures are to continue with the concurrent rehabilitation of the Waste Rock / Dry Tailings Co-Disposal and the TSF throughout the remaining life of the facilities. Final capping will occur during the closure phase. Thus, the impact will be noticeable yet of limited duration until such time as the new terraces are re-vegetated.

20.7 DECOMMISSIONING AND CLOSURE PHASE

All potential impacts are rated from minor negative to minor positive post mitigation. A conceptual closure plan has been developed which indicates a positive, achievable and sustainable post closure land use. The post closure land use includes grazing land, agricultural development and managed domestic water supply. Active agriculture and managed domestic water supply are potential improvements on the base case, as these options do not exist at present.

From a conceptual closure perspective, the open pit will be left as a void upon closure. Difficulties could be experienced by concurrent in-filling in those cases where the ore body is limited to a single open pit, as is the case with the Project, and various grades of ore needs to be sourced from the pit. This requires access to the full pit and in-filling could sterilize ore reserves. In these cases, rehabilitation will be facilitated as follows:

The open pit perimeter walls will be rendered safe for humans and domestic animals. This is achieved by means of the following:

- Sloping the perimeter walls of the opencast pit to the pit floor, or to stable groundwater level that could establish within a reasonable period within the opencast pit

- Provide enviro-berms and ditches along the opencast pit perimeter when perimeter wall flattening is not possible
- Owing to removal of the mined product off site, and the creation of permanent mine residue facilities, insufficient material remains and a final void with respect to this project is unavoidable
- Conceptual land use options include a natural water body. Geochemical and hydrological studies indicate that the pit water quality will be suitable for stock watering and controlled irrigation (within limits).

These measures are in line with international practice and the opportunities available to utilise the flooded open pit for livestock watering and controlled irrigation provides a unique opportunity for the project to contribute to a sustainable post mining land use.

Regarding the co-disposal facility, the minimum objectives for the closure and rehabilitation are to stabilise the facility both chemically and physically and thereby prevent air, ground and water pollution, in accordance with the requirements of the relevant regulations and in line with good international practice. The intended end use considers the prior land-use and the location with respect to current and potential future socio-economic development. The walls of the co-disposal facility should be top soiled and vegetated progressively during construction and operation of the dam, whilst the upper surface may only be top soiled and vegetated once the final fill level has been reached.

20.8 WASTE AND WATER MANAGEMENT

20.8.1 Mine Waste Assessment

A hydro-geochemical assessment was undertaken, including laboratory test work and modelling, on tailings material and representative waste rock. The results 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, decreasing to slightly acidic in the long-term. Metal concentrations will be low. Appropriate capping systems were specified to manage the long-term post closure water quality emanating from residual waste product.

20.8.2 Surface Water Environment

The Project site is located on the divide of three major river basins: Onilahy River Basin (32,361 km² catchment), Linta River Basin (6,177 km² catchment) and Menarandra River Basin (8,681 km² catchment). The calculated mean annual runoff 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. Watercourses are dry during the dry season and only flow during the rainy season.

Local surface water sources are largely underdeveloped, and no significant local water demands were identified during water use surveys.

20.8.3 Groundwater Environment

The Project site falls within the Ampanihy shear zone with a regional north-northeast – south-southwestern trend. The local geology consists mainly of crystalline metamorphic rock such as quartzofeldspathic 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 50 metres 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. Groundwater use in the larger area is low, 15 to 20 m³ per day maximum.

20.8.4 Water Management

The potential impacts that mining activities are likely to have on surface and groundwater resources in the surrounding environment were analyzed. These potential impacts were quantified with different groundwater and surface water models and a significance rating was assigned.

Potential impacts relate to mine dewatering, wellfield abstraction, dirty water runoff, runoff reduction, and potential contaminant seepage from the Co-Disposal Facility and Material 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 re-use of water, storm water management measures, water monitoring and waste capping systems.

20.9 Socio-Economics

A comprehensive assessment pertaining to the demographics and economic status of the communities in proximity to the project was undertaken, specifically on those communities within or adjacent to the area of influence. Significant reductions in the overall area of influence was achieved through careful consideration of environmental and social sensitivities, in order to minimize the affected area.

Various avenues of engagement with stakeholders have been undertaken to date. These can be categorized as follows:

- Local and Regional engagement with government and other regulatory stakeholders
- Public consultation

- Land tenure and occupation
- Cultural heritage
- Sources of income

Engagement outcomes can be summarised as follows:

- Job creation
- Education and skills transfer
- Strengthening of Municipal Resources
- Improvement in local security
- Increasing business opportunities
- Relocation of houses and associated assets (minimum requirement in order to allow for public safety and operational efficiency)
- Loss of grazing land

In this case though, there is a need to relocate crops and grazing land utilised by local inhabitants within close proximity to the project site, for which a Relocation Action Plan (RAP) has been developed.

20.10 MINE REHABILITATION AND CLOSURE

The development of a MRCP creates a platform for a mine to always have the end of the life of the mine in focus and to plan ahead for successful closure. Closure planning is guided internationally by different principles, frameworks, guidelines and local legislation.

The framework for the MRCP specifically for the proposed Project was compiled from various recognised standards. These being:

- Equator Principals
- International Finance Corporation (IFC) Performance Standards
- IFC Environmental, Health and Safety Guidelines for Mining
- Madagascar Legislation
- The International Council of Mining and Metals (ICMM) guidelines

The proposed Project is currently in the Feasibility Phase and requires, at this stage, a conceptual MRCP. The conceptual MRCP was developed to reach the following target outcomes and goals:

- That future public health and safety are not compromised
- The after-use of the site is beneficial and sustainable to the affected communities in the long term

- Adverse socio-economic impacts are minimized, and socio-economic benefits are maximized
- Make the area of influence healthy and stable, and restore its ability to allow another activity compatible with any form of life and activity in the region where it is located, after the closure of the mining operation
- Ensure the safety of the location during and after the mining operation
- Reduce the harmful effects of the mining operation on the atmosphere and on the water regime at an acceptable level
- Integrate the mine and infrastructure landscape through appropriate negotiations and potential arrangements with the local municipality
- Prevent the introduction of pests and weeds in areas where they were not present, and
- Promote rapid re-generation and renewal of native plant species, or compatible with the ecosystem of the area

Rehabilitation and closure estimates were annualised over the Project's life of mine.

Refer Table 46 below for the closure estimates, based at end of LOM at present time value of money.

TABLE 46: REHABILITATION AND CLOSURE COST SUMMARY

			720 ktpa Project	
			Owner Operator Rehabilitation	Third Party Rehabilitation
Plant Area and Associated Infrastructure			\$544 668.57	\$730 945.22
Opencast			\$736 724.55	\$988 684.35
Waste Rock Dump / Co-Disposal Facility			\$842 804.82	\$1 131 044.07
Tailings Storage Facility			\$2 277 070.85	\$3 055 829.09
Explosive Magazine			\$15 755.87	\$21 144.37
Roads and Pipeline			\$44 407.61	\$59 595.01
Water Infrastructure / Water Supply			\$69 938.01	\$93 856.82
Sub-Total*			\$4 531 370.29	\$6 081 098.92

* Excludes value-added taxes

20.11 PERMITTING AND STAKEHOLDERS

As input into the FS, pertinent information on the required permits / licenses to mine, including additional applicable permits / licenses for associated activities, was gathered and compiled into a permit register, a stakeholder database was compiled in support of the permitting register to assist with the identification of and approach to the relevant parties concerned.

The Madagascar Ministry of Environment's Office National pour l'Environnement (the National Office for the Environment) or "ONE", granted the Company its Global Environmental Permit/License for the 240ktpa (Phase 1) Project on April 8, 2019 after reviewing the following:

- Receipt of Exploitation Permit PE #39807
- Environmental & Social Impact Assessment ("ESIA") and Relocation Action Plan ("RAP") to International Finance Corporation (IFC) Performance and World Bank Standards
- Completion of local and regional stakeholder and community engagement, with overwhelming support from both the local community and local government, as well as regional government
- Signed agreements with all potentially affected land occupants to accept compensation for any affected crops and grazing land and relocation if needed
- Approved capital investment certification from the BCMM
- Receipt of Cahier des Charges Miniér (mining specification) from the BCMM as pre-requisite to submitting the ESIA & RAP to ONE for review
- Successful completion of the ONE's technical evaluation process which consisted of a site visit and four separate community consultations
- Joint agreement and signature of the Cahier des Charges Environnementales (environmental specification) with the ONE

In addition to the above, a number of sectorial EMP's and other permits (listed below) will be undertaken during the early project execution phase.

The proposed Project has triggered the following list of permits / licenses (list is not exhaustive):

- Authorisation for extracting of borrow materials from a quarry
- Building permit (housing and ancillary infrastructure)
- Authorisation to construct a waste water treatment facility and to discharge treated water
- Water uptake authorisation

- Authorisation to rehabilitate and maintain roads
- Authorisation to generate and distribute electricity
- Permit to operate a domestic landfill site
- Authorisation to circulate haul trucks on public roads
- Authorisation for tree clearing
- Authorisation to beneficiate raw ore into concentrate
- Permits for recruitment of labour
- Lease of the land

The relevant stakeholder groups are (list is not exhaustive):

- Ministries
- Regional Government
- District Government
- Local Decentralized Collectivises, or Municipalities
- NGO's and Public Companies
- Attached Authorities to Ministries; and
- The local population

21 CAPITAL AND OPERATING COSTS

21.1 CAPEX

The Phase 1 initial CAPEX is estimated at \$21.0 M, including a 10% contingency, with an additional \$3.1 M required to cover the first 3 months of working capital. Phase 2 CAPEX (inclusive of Phase1) is estimated at \$67.4 M, including a 12.5% contingency, with an additional \$7.3 m required to cover the first 3 months of working capital. Over the life of the mine, sustaining capital of \$3.3 M will be required for equipment replacement and for rehabilitation at the end of the project.

Table 47 summarizes the capital requirements.

The base date for the capital costs is May,2019 and no provision has been made for escalation. The accuracy of capital costs is considered to be with +/- 10%.

TABLE 47: CAPITAL COSTS

Capital Cost Breakdown	Phase 1	Phase 2
Process Equipment	\$8,438,609.00	\$25,315,827.00
Civil & Infrastructure	\$2,103,672.21	\$6,661,016.63
Tailings	\$0.00	\$0.00
Mining	\$2,574,143.85	\$4,913,341.38
Buildings	\$1,154,609.43	\$2,886,523.59
Electrical Infrastructure	\$128,804.10	\$386,412.30
Project Services/EPCM	\$931,481.79	\$2,794,445.38
Construction Services	\$1,474,775.11	\$3,686,937.78
Indirect Costs	\$372,750.00	\$1,118,250.00
Environmental & Permitting costs	\$729,827.94	\$1,459,655.89
Owner's Costs	\$1,197,000.00	\$4,189,500.00
Sub-total	\$19,105,673.44	\$53,411,909.93
Contingency (10%; 12.5%)	\$1,910,567.34	\$6,676,488.74
3 Months Working Capital	\$3,100,000	\$7,300,000
CAPEX TOTAL	\$24,116,241	\$67,388,398.67
Sustaining CAPEX over Life of Mine		\$3,300,000

21.1.1 BASIS OF ESTIMATE

The capital costs for the Project are built up as follows:

21.1.1.1 Modular Plant

The revision of the previous Molo 2015 FS was motivated by a modular plant, designed at an appropriate size to accommodate current market realities.

The plant has been designed to a +90% accuracy with the only potential variances contributable to final vendor equipment details which are only available post procurement.

A modular engineering and manufacturing company has been part of the project team for multiple months and have provided a lump sum, design, supply, delivery and installation price, complete with allowances for commissioning and spares.

Detailed design reviews were completed and there is a high level of comfort with the accuracy of the design and costing.

Table 48 and 49 summarize the CAPEX breakdown of the turn-key price provided by the modular plant manufacturer.

TABLE 48: PHASE 1 MODULAR PLANT CAPEX

Description	Cost
Fixed price offer for Manufacture and Delivery of Process Plant Complete	\$ 7,500,000.00
Duties	\$ 938,609 .00
Sub-Total	\$ 8,438,609.00

TABLE 49: PHASE 2 MODULAR PLANT CAPEX

Description	Cost
Fixed price offer for Manufacture and Delivery of Process Plant Complete	\$ 22,500,000.00
Duties	\$ 2,815,827.00
Sub-Total	\$ 25,315,827.00

21.1.1.2 Earthworks, Civils and Infrastructure

DETAIL DESIGNS HAVE BEEN COMPLETED FOR ALL OF THE EARTHWORKS AND CIVILS, COMPLETE WITH ACCURATE SCHEDULES OF QUANTITIES WHICH HAVE BEEN USED TO CONFIRM CURRENT PRICING. TABLE 50 AND

Table 51 summarize the CAPEX requirement for earthworks and civils.

TABLE 50: PHASE 1 EARTHWORKS AND CIVILS CAPEX

Description	Supply	Transportation	Erection	Duties and taxes	Total
<u>Bulk Earthworks</u>					
					\$ 586 331
<u>Concrete Works</u>					
					\$ 385 481
Preliminary & General					\$ 485 475
<u>Infrastructure</u>					
Gabion retaining wall	\$ 75,000.00				\$ 80 625
Water purification Plant	R 2,266,434.00	R 106,000.00	R 250,000.00	R 232,560.00	R 2,854,994
Sewage treatment works	R 1,571,148.00	R -	R -	R 123,120.00	R 1,694,268
450m ³ storage tanks	\$ 32,338.00	\$ 1,820.00	\$ 4,850.00	\$ 3,519.84	\$ 42,528
Fencing	\$ 83,800.00				\$ 83,800
Water Reticulation					\$ 30,000
Sewer reticulation					\$ 15,000

Weighbridge	R 565,582.50	R 106,000.00	R 50,000.00	R 54,000.00	R 775,583
Sub-Total	\$ 2,103,672.21				

TABLE 51: PHASE 2 EARTHWORKS AND CIVILS CAPEX

Description	Supply	Transportation	Erection	Duties and taxes	Total
<u>Bulk Earthworks</u>	\$ 1,762,992				\$ 1,762,992
<u>Concrete Works</u>	\$ 1,075,760				\$ 1,075,760
Preliminary & General	\$ 1,914,813				\$ 1,914,813
<u>Infrastructure</u>					
Gabion retaining wall	\$ 225,000	\$ 56,000	\$ 38,000	\$ 120,000	\$ 439,000
Water purification Plant	\$ 430,667	\$ 23,556	\$ 55,556	\$ 51,680	\$ 561,458
Sewage treatment works	\$ 228,000			\$ 27,360	\$ 255,360
450m ³ storage tanks	\$ 87,996	\$ 5,460	\$ 14,550	\$ 10,560	\$ 118,566
Fencing	\$ 251,400				\$ 251,400
Water Reticulation	\$ 90,000				\$ 90,000
Sewer reticulation	\$ 45,000				\$ 45,000
Weighbridge	\$ 100,000	\$ 23,556	\$ 11,111	\$ 12,000	\$ 146,667
Sub-Total	\$ 6,211,629	\$ 108,572	\$ 119,217	\$ 221,600	\$ 6,661,016.63

21.1.1.3 Mining

Mining CAPEX have been developed from first principles using an owner mining philosophy where all equipment has been priced for supply and delivery to site. Operational costs are accurately captured based on first principles and expected organograms. Table 52 and

Table 53 summarize the mining CAPEX requirements.

TABLE 52: PHASE 1 MINING CAPEX

Machine	Replacement (hours)	COST		Capital
EXCAVATORS & LOADERS				
SANY SY465H Excavator - 2m ³	15000	\$ 303,417	1	\$ 303,417.40
LIUGONG Front End Loader 3m ³	15000	\$ 75,323	2	\$ 150,647.80
HAUL TRUCKS				\$ -
SINOTRUK HOWO Tipper Truck 16m ³ (Howa)	15000	\$ 64,714	5	\$ 323,574.50
SINOTRUK HOWO Water Truck 20000L (Howa)	15000	\$ 64,714	1	\$ 64,714.90
MOBILE EQUIPMENT				\$ -
HOWA Truck Mounted Drill Rig	10000	\$ 229,154	1	\$ 229,154.40
SANY Motor Grader 200HP	15000	\$ 194,144	1	\$ 194,144.70
SANY Compactor 10t	15000	\$ 85,932	1	\$ 85,932.90
SHANTUI SD22W HD Tracked Dozer	15000	\$ 154,891	1	\$ 154,891.40
SERVICE & SUPPORT EQUIPMENT				\$ -
Buses 60-seater	6 years	\$ 111,999	1	\$ 111,999.62
Maintenance / Diesel Truck	6 years	\$ 185,657	1	\$ 185,657.50
Pump Diesel 50m ³ /hr	6 years	\$ 115,734	1	\$ 115,734.55
Supervision Vehicle	6 years	\$ 26,271	7	\$ 183,897.47
Light Plant				
Spares (1 Years) -		10% of Total		\$ 210,376.71
Explosive Magazines				\$ 210,000.00
TOTAL ESTIMATED CAPITAL SPEND.				\$2,574,143.85

TABLE 53: PHASE 2 MINING CAPEX

Machine	Replacement (hours)	COST		Capital
EXCAVATORS & LOADERS				
SANY SY465H Excavator - 2m ³	15000	\$ 303,417	2	\$ 606,835
LIUGONG Front End Loader 3m ³	15000	\$ 75,323	3	\$ 225,972
HAUL TRUCKS				
SINOTRUK HOWO Tipper Truck 16m ³ (Howa)	15000	\$ 64,714	11	\$ 711,864
SINOTRUK HOWO Water Truck 20000L (Howa)	15000	\$ 64,714	2	\$ 129,430
MOBILE EQUIPMENT				
HOWA Truck Mounted Drill Rig	10000	\$ 229,154	2	\$ 458,309
SANY Motor Grader 200HP	15000	\$ 194,144	2	\$ 388,289
SANY Compactor 10t	15000	\$ 85,932	2	\$ 171,866
SHANTUI SD22W HD Tracked Dozer	15000	\$ 154,891	2	\$ 309,783
SERVICE & SUPPORT EQUIPMENT				
Buses 60-seater	6 years	\$ 111,999	1	\$ 111,999
Maintenance / Diesel Truck	6 years	\$ 185,657	2	\$ 371,315
Pump Diesel 50m ³ /hr	6 years	\$ 115,734	2	\$ 231,469
Supervision Vehicle	6 years	\$ 26,271	10	\$ 262,710
Light Plant				
Spares (1 Years) -		10% of Total		\$ 440,000
Transportation of mining equipment to site				\$ 400,000
Explosive Magazines				\$ 93,500
TOTAL ESTIMATED CAPITAL SPEND				\$4,913,341

21.1.1.4 EPCM costs

These costs were built up in detail based on the scope and time frame of the project with an owners engineering philosophy. Table 54 summarizes the project EPCM costs for Phase 1. Phase 2 EPCM costs were factored as 3 times the cost of Phase 1, at \$2,794,445.37.

TABLE 54: EPCM CAPEX

Description	Unit	Quantity	Rate	Unit Cost	Allowances	Cost
Engineering Team						
Project Manager	Month	9	\$27,048	\$243,440		\$ 243,440
Process	Month	6	\$11,000	\$ 66,000		\$ 66,000
Electrical	Month	6	\$11,000	\$ 66,000		\$ 66,000
Civil/structural/infrastructure	Month	6	\$11,000	\$ 66,000		\$ 66,000
Mining	Month	3	\$11,000	\$ 33,000		\$ 33,000
Procurement / Commercial	Month	6	\$ 8,000	\$ 48,000		\$ 48,000
Environmental consultant	Month	12	\$ 5,000	\$ 60,000		\$ 60,000
HR Consultant	Sum	1	\$30,000	\$ 30,000		\$ 30,000
Site Team						
Construction manager	Month	7	\$15,000	\$105,000	\$ 8,750	\$113,750
HSE Manager	Month	7	\$ 6,500	\$ 45,500	\$ 3,792	\$ 49,292
Electrical Sup	Month	6	\$ 9,000	\$ 54,000	\$ 4,500	\$ 58,500
Project Planner	Month	6	\$ 6,000	\$ 36,000	\$ 3,000	\$ 39,000
Mining Sup	Month	6	\$ 9,000	\$ 54,000	\$ 4,500	\$ 58,500
					EPCM Costs	\$ 931,481.79

21.1.1.5 Risk Allowances

A contingency amount of 10% has been allowed for Phase 1, and 12.5% for Phase 2.

This estimate is categorised as a "Feasibility Type" with a combination of detailed, and semi-detailed costs.

The estimate has been produced using Vendor firm and budget quotations and market data

21.1.2 Base Date

The base date for the Capital Cost Estimate is May 2019

21.1.3 Base Currency

The estimate has been presented in the US Dollar currency in present day terms. Prices obtained in other currencies have been converted to US Dollars using the applicable exchange rates.

Fluctuations in the exchange rates from the date of quotation to the date of indention have not been catered for.

21.1.4 Scope

The scope of the estimate covers the total capital cost of the plant areas and the related infrastructure as more fully described in the body of the main study documents. Owner's costs are all included.

21.1.5 Escalation

Escalation has not been considered in the Capital Cost Estimate.

21.1.6 Transport

Transport costs were included in the modular plant as well as mining equipment costing where appropriate.

21.1.7 Overall Philosophy

The general approach to estimating was to measure / quantify each cost element from the engineering drawings, PFD's, mechanical equipment list, infrastructure equipment list, and motor lists.

21.1.8 Assumptions

The estimate for the plant has been based on an assumption of a continuous engineering, procurement and construction effort with no interruption of the implementation programme after funding approval has been obtained.

21.1.9 Spares, First Fill and Consumables

Allowances have been made for the First Fill and for Consumables based on the supplier and the process design engineer estimates.

21.2 OPEX

The operating costs per tonne of finished graphite flake concentrate delivered on a FOB basis at the Port of Fort Dauphin, Madagascar are outlined in Table 55. The reduction in mining and G&A cost per unit produced between Phase 1 and Phase 2 results from the fact that those costs include fixed costs which do not change with the increase in production and therefore leads to a reduction in unit cost.

TABLE 55: OPERATING COSTS PER TONNE OF FINISHED GRAPHITE CONCENTRATE

Category	Phase 1	Phase 2
	Operating cost	
Mining (US\$/T)	102.81	65.34
Processing (US\$/T)	265.82	265.82
Trucking to local port / Ft. Dauphin (US\$/T)	133.01	133.01
General and Administration (US\$/T)	64.29	50.00
TOTAL	\$565.93	\$514.17

Please note that these operating costs assume that the plant is able to successfully handle the variability in the ore body, as shown by the SGS test work discussed in detail in Section 13.

21.2.1 Process Operating Cost Summary

Table 56 below shows the operating costs for the concentrator process plant. The breakdown of the cost is presented graphically in Figure 89. As was expected, electrical power contributes most of the plant operating costs at 42.3%. This is because of the use of generators for power generation.

TABLE 56: PROCESS PLANT OPEX ESTIMATE

Description	Annual Cost US\$/annum	Unit Cost US\$/t milled	% Total
Power	2 126 991	8.86	42.3%
Water	10 417	0.04	0.2%
Reagents	366 590	1.53	7.3%
Milling	436 925	1.82	8.7%
Product Dryer	97 408	0.41	1.9%
Tailing	56 924	0.24	1.1%
Labour	1 423 772	5.93	28.3%
Maintenance	227 239	0.95	4.5%
Laboratory	186 594	0.78	3.7%
Miscellaneous Operating Cost	92 502	0.39	1.8%
Total Operating Cost	5 025 362.84	20.9	100.0%

- Note 1: Process plant operating costs are assumed to be the same for Phase 1 and Phase 2 when in reality, economies of scale should result in a reduction of costs for Phase 2.

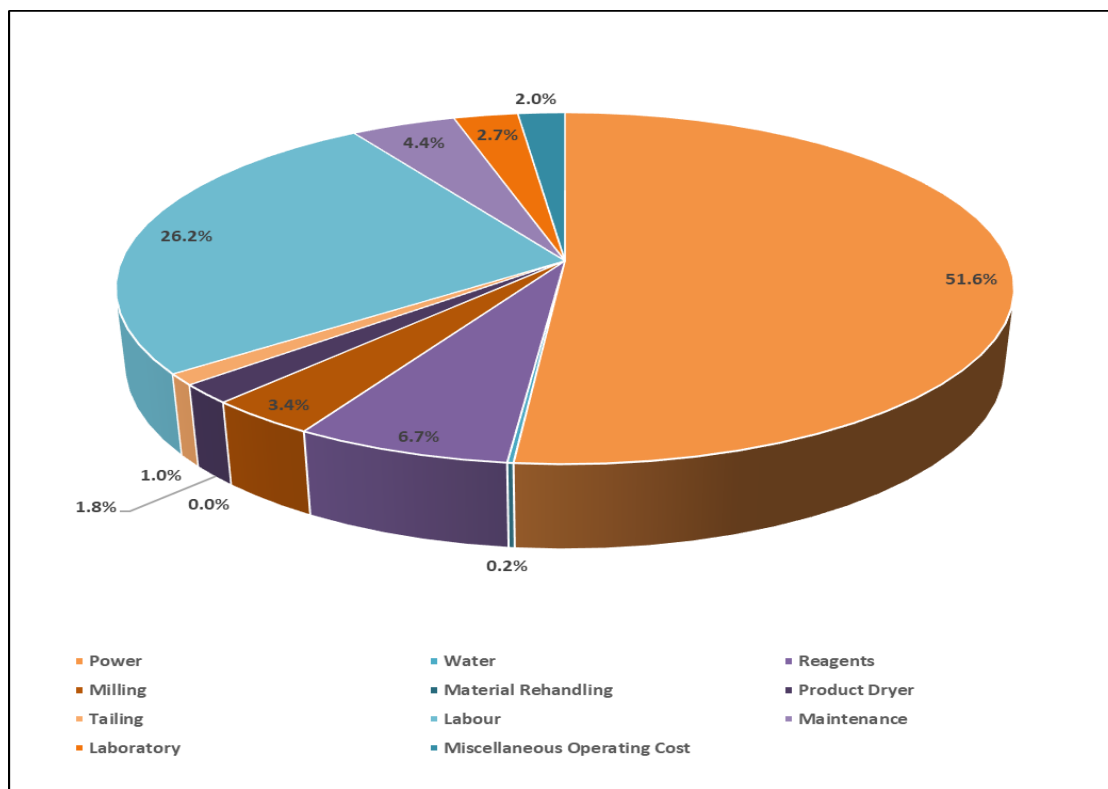


FIGURE 89: PROCESS PLANT OPERATING COST BREAKDOWN

21.2.1.1 Methodology

The processing plant operating cost (OPEX) estimate is based on averaged, steady-state operating conditions of a 240 ktpa run-of-mine (ROM) feedrate and 15 ktpa of final product. Calculations to derive the estimate were informed by the process flowsheets, mass balance outputs, process design criteria and required mechanical equipment.

21.2.1.2 Labour

Labour costs for the plant are based on the staff organogram shown in Figure 90 below. The Process Manager (DU) will be an expatriate appointment while the other employees may be sourced locally.

The staff complement for the project is 79 (see Table 57). The labour cost per ton of ore milled was found to be USD 5.0. The organogram and staffing specifications were based on Met63's experience in the operation of concentrator plants.

The table of rates was not adjusted and is still abased on the Molo 2017 240ktpa FS, but the final amount has been adjusted for escalation.

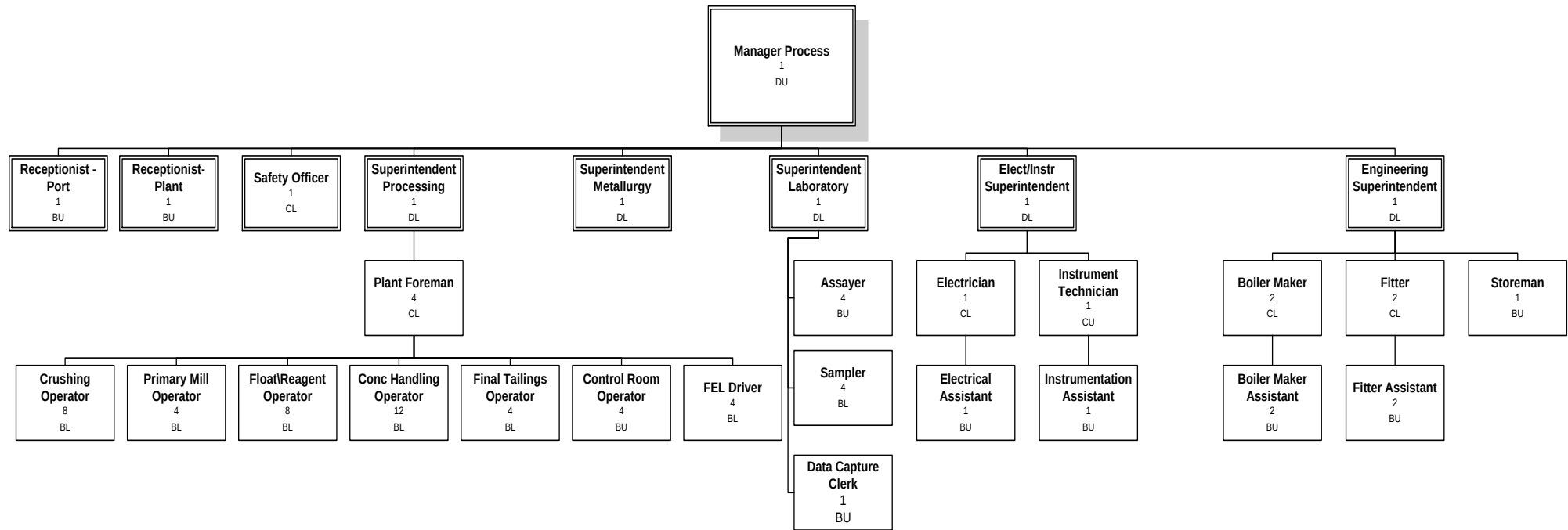


FIGURE 90: MOLO 240 KTPA GRAPHITE PROJECT STAFF ORGANOGRAM

TABLE 57: LABOUR COSTS

Occupation	Site	Department	Grade	Number	Total	Total Monthly Salary US\$	Total Monthly Medical Cost US\$	Total Monthly Transport Cost US\$	Total Monthly PPE Cost	Monthly Cost US\$	Annual Cost US\$	Unit Cost US\$/t
Manager Process	Plant	Management	DU	1	1	10 327	1 033	100	9	11 469	137 623	0.6
Administrator/Receptionist	Plant	Management	BU	1	1	689	69	100	9	867	10 402	0.0
Adminstrator/Receptionist	Port	Management	BU	0	1	689	69	100	9	867	10 402	0.0
Elect/Instr Superintendent	Plant	Electrical / Instrumentation	DL	1	1	5 969	597	100	9	6 674	80 091	0.3
Electrician	Plant	Electrical / Instrumentation	CL	1	1	1 165	117	100	9	1 390	16 685	0.1
Instrument Technician	Plant	Electrical / Instrumentation	CU	1	1	1 936	194	100	9	2 239	26 862	0.1
Electrical Assistant	Plant	Electrical / Instrumentation	BU	1	1	689	69	100	9	867	10 402	0.0
Instrumentation Assistant	Plant	Electrical / Instrumentation	BU	1	1	689	69	100	9	867	10 402	0.0
Engineering Superintendent	Plant	Engineering	DL	1	1	5 969	597	100	9	6 674	80 091	0.3
Fitter	Plant	Engineering	CL	1	2	2 330	233	200	18	2 781	33 370	0.1
Fitter	Port	Engineering	CL	0	0	-	-	-	-	-	-	-
Boiler Maker	Plant	Engineering	CL	1	2	2 330	233	200	18	2 781	33 370	0.1
Storeman	Plant	Engineering	BU	1	1	689	69	100	9	867	10 402	0.0
Boiler Maker Assistant	Plant	Engineering	BU	1	2	1 378	138	200	18	1 734	20 804	0.1
Fitter Assistant	Plant	Engineering	BU	1	2	1 378	138	200	18	1 734	20 804	0.1
Superintendent Laboratory	Plant	Laboratory	DL	1	1	5 969	597	100	9	6 674	80 091	0.3
Sampler	Plant	Laboratory	BL	1	4	1 644	164	400	36	2 244	26 929	0.1
Sampler/Assayer	Port	Laboratory	BL	0	0	-	-	-	-	-	-	-
Assayer	Plant	Laboratory	BU	1	4	2 756	276	400	36	3 467	41 607	0.2
Data Capture Clerk	Plant	Laboratory	BU	1	1	689	69	100	9	867	10 402	0.0
Superintendent Processing	Plant	Operations	DL	1	1	5 969	597	100	9	6 674	80 091	0.3
Plant Foreman	Plant	Operations	CL	1	4	4 660	466	400	36	5 562	66 740	0.3
Plant Supervisor	Port	Operations	CL	0	0	-	-	-	-	-	-	-
Crushing Operator	Plant	Operations	BL	2	8	3 288	329	800	71	4 488	53 858	0.2
Primary Milling Operator	Plant	Operations	BL	1	4	1 644	164	400	36	2 244	26 929	0.1
Float/Reagents Operator	Plant	Operations	BL	2	8	3 288	329	800	71	4 488	53 858	0.2
Final Tailings Operator	Plant	Operations	BL	1	4	1 644	164	400	36	2 244	26 929	0.1
Concentrate Handling Operator	Port	Operations	BL	3	12	4 932	493	1 200	107	6 732	80 786	0.3
Control Room Operator	Plant	Operations	BL	1	4	1 644	164	400	36	2 244	26 929	0.1
FEL Driver	Plant	Operations	BL	1	4	1 644	164	400	36	2 244	26 929	0.1
Superintendent Metallurgy	Plant	R&D	DL	1	1	5 969	597	100	9	6 674	80 091	0.3
Safety Officer	Plant	Safety	CL	1	1	1 165	117	100	9	1 390	16 685	0.1
Total				32	79	83 130	8 313	7 900	704	100 047	1 200 562	5.0

21.2.1.3 Power

The breakdown of the power cost is shown in Table 58 below. The total cost per metric ton was found to be USD 8.86/t milled. As was expected, milling, flotation and tailings handling contribute a larger percentage to the overall power consumption. The power cost calculations are based on the use of generator power at a cost of USD 0.22 per kWh.

TABLE 58: BREAKDOWN OF THE OPERATING COST FOR POWER

Fixed/Demand Charge		
Max Demand US\$/KW		US\$/kW/year
Installed Capacity - Crushing	186	kW
Installed Capacity - Mill/Float/Tailings	607	kW
Installed Capacity - Drying/Screen/Bagging	293	kW
Total Variable Power Cost		US\$/year
Total Variable Power Cost		US\$/t
Crushing		
Crushing Unit Power Cost	0.26	US\$/kWh
Crushing Power Consumption	897 774	kWh/year
Crushing Total Variable Power Cost	234 232	US\$/year
Crushing Total Variable Power Cost	0.98	US\$/t
Mill/Float/Tailings		
Mill/Float/Tailings Unit Power Cost	0.26	US\$/kWh
Mill/Float/Tailings Power Consumption	4 893 244	kWh/year
Mill/Float/Tailings Total Variable Power Cost	1 276 660	US\$/year
Mill/Float/Tailings Total Variable Power Cost	5.32	US\$/t
Drying/Screen/Bagging		
Drying/Screen/Bagging Unit Power Cost	0.26	US\$/kWh
Drying/Screen/Bagging Power Consumption	2 361 414	kWh/year
Drying/Screen/Bagging Total Variable Power Cost	616 099	US\$/year
Drying/Screen/Bagging Total Variable Power Cost	2.57	US\$/t
Power		
Annual Power Cost	2 126 991	US\$/year
Monthly Power Cost	177 249	US\$/month
Power Total Unit Cost	8.86	US\$/t

21.2.1.4 Water

The operating costs for water were based on potable water usage of 8 784 m³/year at a unit cost of USD 1/m³. Table 59 below shows the water operating costs.

TABLE 59: OPERATING COSTS FOR WATER

Potable Water Cost		
Potable Water Cost	1.2	US\$/m ³
Potable Water Consumption	8 784	m ³ /year
Potable Water Cost	10 417	US\$/year
Potable Water Cost	0.04	US\$/t
Water		
Annual Water Cost	10 417	US\$/year
Monthly Water Cost	868	US\$/month
Water Total Unit Cost	0.04	US\$/t

21.2.1.5 Reagent Costs

The consumption figures used in determining the reagent operating cost estimate were based on the process design criteria information and are supported by the plant mass balance. The reagent costs used were based on imported supply rates obtained from reagent suppliers FOB: Faux Cap, Madagascar. Table 60 shows the reagent operating costs.

TABLE 60: REAGENT OPERATING COSTS

Collector		
Cost	1157.459	t/year
Dosage Rate	111	g/t
Consumption	26.6	t/year
Annual Cost	30835	US\$/year
Monthly Cost	2570	US\$/month
Unit Cost	0.13	US\$/t
Frother		
Cost	3506.768	t/year
Dosage Rate	165	g/t
Consumption	39.6	t/year
Annual Cost	138868	US\$/year
Monthly Cost	11572	US\$/month
Unit Cost	0.58	US\$/t
Flocculant (Concentrate)		
Cost	2400.304	t/year
Dosage Rate	1	g/t
Consumption	0.2	t/year
Annual Cost	576	US\$/year
Monthly Cost	48	US\$/month
Unit Cost	0.0024	US\$/t
Flocculant (Tailings)		
Cost	2400.304	t/year
Dosage Rate	125	g/t
Consumption	30.0	t/year
Annual Cost	72009	US\$/year
Monthly Cost	6001	US\$/month
Unit Cost	0.30	US\$/t
Coagulant		
Cost	3046.631	t/year
Dosage Rate	170	g/t
Consumption	40.8	t/year
Annual Cost	124303	US\$/year
Monthly Cost	10359	US\$/month
Unit Cost	0.52	US\$/t
Reagents		
Annual Reagent Cost	366 590	US\$/year
Monthly Reagent Cost	30 549	US\$/month
Reagent Total Unit Cost	1.53	US\$/t

21.2.1.6 Milling

The milling costs (shown in Table 61) consist of the grinding media costs as well as the costs for re-lining.

TABLE 61: MILLING COSTS

Milling		
Balls Unit Cost	1.60	US\$/t
Liner Unit Cost	0.22	US\$/t
Balls Annual Cost	384389	US\$/year
Liner Annual Cost	52536	US\$/year
Total Annual Cost	436925	US\$/month

21.2.1.7 Product Drying

The cost of product drying (shown in Table 62) is USD 0.41/t; this is the same figure as used in the Molo 2015 FS.

TABLE 62: PRODUCT DRYING COSTS

Product Drying		
Product Drying Cost	0.41	US\$/t
Annual Cost	97408	US\$/year
Monthly Cost	8117	US\$/month
Unit Cost	0.41	US\$/t

21.2.1.8 Final Tailings Handling

Table 63 shows the cost of tailings handling. The tailings handling unit cost is USD 0.24/t.

TABLE 63: FINAL TAILINGS HANDLING COSTS

Final Tailings Handling		
Final Tailings Handling Cost	0.24	t/year
Annual Cost	56924	US\$/year
Monthly Cost	4744	US\$/month
Unit Cost	0.24	US\$/t

21.2.1.9 Maintenance Costs

The maintenance costs (see Table 64) were estimated according to the capital cost of the mechanical equipment for the crusher, wet plant and dry circuits. The cost was estimated to be 7.5% of the mechanical equipment cost for each circuit.

TABLE 64: MAINTENANCE COSTS

Crusher		
Capital Equipment Cost	289 971	US\$/t
Spares Allowance	7,5%	
Annual Cost	21 748	US\$/year
Monthly Cost	1 812	US\$/month
Unit Cost	0.09	US\$/t
Wet Plant		
Capital Equipment Cost	2 284 140	US\$/t
Spares Allowance	7,0%	
Annual Cost	159 890	US\$/year
Monthly Cost	13 324	US\$/month
Unit Cost	0.67	US\$/t
Dry Plant		
Capital Equipment Cost	912 023	US\$/t
Spares Allowance	5,0%	
Annual Cost	45 601	US\$/year
Monthly Cost	3 800	US\$/month
Unit Cost	0.19	US\$/t
Total		
Annual Cost	227 239	US\$/year
Monthly Cost	18 937	US\$/month
Total Unit Cost	0.95	US\$/t

21.2.1.10 Laboratory Costs

Table 65 below presents the breakdown of the laboratory costs for the project. The estimated cost to process a single sample is USD 1.0. A total of 144 616 laboratory samples will be processed on a yearly basis, which will result in a laboratory cost per ton of USD 0.6.

TABLE 65: LABORATORY COST BREAKDOWN

Laboratory Plant Sample		
No of Samples	54 750	/year
Cost per sample	1.20	US\$/t
Annual Cost	64 929	US\$/year
Monthly Cost	5 411	US\$/month
Unit Cost	0.27	US\$/t
Grade Control Samples		
No of Samples	81 162	/year
Cost per sample	1.20	US\$/t
Annual Cost	96 252	US\$/year
Monthly Cost	8 021	US\$/month
Unit Cost	0.40	US\$/t
Concentrate Sample		
No of Samples	21 429	/year
Cost per sample	1.2	US\$/t
Annual Cost	25 413	US\$/year
Monthly Cost	2 118	US\$/month
Unit Cost	0.11	US\$/t
Total		
Annual Cost	186 594	US\$/year
Monthly Cost	15 550	US\$/month
Total Unit Cost	0.78	US\$/t

21.2.1.11 Miscellaneous

The miscellaneous costs (Table 66) include stationery and office equipment, lubricants, workshop tools and computer software. The costs were based on Met63 process plant experience and were discussed with the Company.

TABLE 66: MISCELLANEOUS COSTS

Description	Monthly Cost US\$	Annual Cost US\$	Unit Cost US\$/t
Lubricant	4 744	56 924.20	0.24
Workshop Tools	1 186	14 231.05	0.06
Computer Software	1 779	21 346.57	0.09
Total	7 708	92 502	0.39

22 ECONOMIC ANALYSIS

22.1 GENERAL

The economic analysis is based on the results of cash flow forecast financial modelling prepared in Microsoft Excel. Inputs to the financial model were obtained from the various technical experts where possible, or from third parties considered experts in their field.

Values shown are rounded to an appropriate level; all financial values shown in this report are based in United States Dollars (US \$) and nominal terms. Returns are presented on both a pre- and post-tax basis. No provision has been made for debt financing.

As the construction of the processing facility contemplates a modular build out process, the Company will have the ability to expand the processing capabilities of the plant based on market demand, and transition from Phase 1 to Phase 2 by tripling the process plant capabilities in the third year of production. The economic analysis was constructed under this assumption.

22.2 ASSUMPTIONS

22.2.1 Product Price

Historical flake graphite pricing was provided by Benchmark Mineral Intelligence for the 12-month period between March 2018 and March 2019 (Figure 86). This historical pricing was then applied to a tonne of average MOLO graphite for the Life of Mine to arrive at a blended average/"basket price" of US \$1207.55/tonne. No escalation factors were applied to this price in the financial models, with the price "flat-lined" for the Life of Mine.

22.2.2 Mineral Royalties

The project is subject to a 2% Revenue Royalty and a 1.5% Net Smelter Royalty.

22.2.3 Technical Assumptions

The primary technical assumptions used in the financial analyses are presented in **Error! Reference source not found.** below.

TABLE 67: SCENARIO 2 TECHNICAL ASSUMPTIONS FOR FINANCIAL ANALYSIS

Item	Value
Average ore mined per year	725,000 T
Average stripping ratio (waste to ore ratio)	0.52
Life of project	30 years
Average mill feed/head grade (%C)	7.1%
Average concentrate grade (%C)	97.3%
Average mill recovery (%)	88.3%

Item	Value
Average tonnes of concentrate production per year	45,000 T

- Note 1: Assumes Phase 1 CAPEX, OPEX and production for first 2 years of operation, with a CAPEX increase to accommodate Phase 2 in the second year of production.

22.2.4 Economic Assumptions

The primary economic assumptions used in the financial analyses are presented in Table 68 below.

TABLE 68: ECONOMIC ASSUMPTIONS FOR FINANCIAL ANALYSIS

Item	
Currency	US Dollar
Discount Rate (% per year)	8%
Average graphite price	USD \$ 1,208/T
Equity	100%
Inflation	0%
Government royalty	2%
Net Smelter Royalty (NSR)	1.5%
Madagascar corporate tax rate	20%
Average production costs of graphite concentrate (at plant)	USD \$378.15/T
Average production costs of graphite concentrate (FOB Fort Dauphin)	USD \$511.16/T

22.3 FINANCIAL MODEL AND RESULTS

Error! Reference source not found. summarizes the total revenues and expenditures calculated for the life of mine (30 years). All values are expressed in millions of USD.

TABLE 69: - FINANCIAL ANALYSIS SUMMARY

Metric	Value
Revenue	1,658.5
Government Royalties	33.2
Mining	114.6
Processing	374.6
Transportation	184.4
G&A and Admin	34.9
Net Smelter Return	13.8
Initial Capital Expenditure	60.1
Initial Working Capital	2.5
Income Taxes	169.5
Closure and Rehabilitation Costs	1.9

The financial model results are provided in **Error! Reference source not found.**

TABLE 70: FINANCIAL ANALYSIS RESULTS

Year		-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
	Total																																
Revenue	1,605.8	-	17.2	19.1	55.9	61.1	60.9	59.5	54.3	53.9	53.8	53.7	53.7	53.9	53.9	53.4	52.7	52.4	52.7	53.1	53.5	53.4	53.6	53.6	53.3	53.0	52.4	52.3	52.5	53.0	53.4	53.4	53.4
Government Royalties	(32.1)	-	(0.3)	(0.4)	(1.1)	(1.2)	(1.2)	(1.2)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.0)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.0)	(1.0)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)
Mining	(112.0)	-	(1.7)	(1.7)	(4.0)	(4.5)	(4.9)	(4.8)	(4.6)	(3.8)	(3.9)	(3.4)	(3.3)	(3.6)	(3.8)	(3.6)	(3.5)	(3.7)	(3.4)	(3.9)	(3.9)	(3.8)	(4.2)	(3.9)	(4.3)	(4.1)	(4.0)	(3.7)	(2.9)	(2.9)	(3.0)	(2.6)	(2.5)
Processing	(359.8)	-	(4.0)	(4.5)	(12.2)	(12.2)	(12.0)	(12.2)	(12.0)	(12.2)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)	(12.1)
Transportation	(176.9)	-	(1.9)	(2.1)	(6.2)	(6.7)	(6.7)	(6.6)	(6.0)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.8)	(5.8)	(5.8)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.9)	(5.8)	(5.8)	(5.8)	(5.8)	(5.8)	(5.9)	(5.9)	(5.9)
G&A and Admin	(33.8)	-	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)
Net Smelter Return	(13.4)	-	(0.1)	(0.1)	(0.5)	(0.5)	(0.5)	(0.5)	(0.4)	(0.4)	(0.4)	(0.5)	(0.5)	(0.5)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.5)	(0.5)	(0.5)
Working Capital Changes	(10.8)	-	(4.4)	0.7	(7.5)	(1.1)	0.1	0.3	1.1	0.0	0.0	(0.0)	(0.0)	(0.0)	0.0	0.1	0.2	0.1	(0.1)	(0.1)	(0.1)	0.0	(0.0)	(0.0)	0.1	0.0	0.1	(0.0)	(0.1)	(0.1)	(0.0)	(0.0)	(0.0)
Initial Capital Expenditure	(60.1)	(17.0)	(31.0)	(12.1)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure and Rehabilitation Costs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Project Cash Flows (pre-tax)	806.9	(17.0)	(27.4)	(2.1)	23.4	33.7	34.6	33.5	30.2	29.5	29.3	29.6	29.7	29.6	29.5	29.3	28.8	28.3	28.7	28.6	28.8	29.0	28.7	29.0	28.5	28.4	28.1	28.2	29.0	29.4	29.6	30.1	30.2
Corporate Taxes	(163.5)	-	(1.3)	(0.9)	(5.0)	(5.8)	(5.7)	(5.4)	(4.6)	(4.7)	(4.6)	(4.7)	(5.0)	(5.7)	(5.9)	(5.8)	(5.7)	(5.6)	(5.8)	(5.7)	(5.8)	(5.8)	(5.8)	(5.7)	(5.8)	(5.7)	(5.6)	(5.6)	(5.8)	(5.9)	(5.9)	(6.0)	(6.0)
Project Cash Flows (post-tax)	643.4	(17.0)	(28.8)	(3.0)	18.4	27.9	28.9	28.0	25.6	24.8	24.6	24.9	24.7	23.9	23.6	23.4	23.1	22.7	22.9	22.9	23.1	23.2	23.0	23.2	22.8	22.7	22.5	22.6	23.2	23.5	23.7	24.1	24.2

The financial model was also run at discount rates of 10% and 12%. A summary of the project economics based on various discount rates are provided in **Error! Reference source not found.** below.

TABLE 71: ECONOMIC METRICS WITH VARYING DISCOUNTS

Metric	Unit	Value
Before Tax		
Total Project Cash Flows	USDm	841.0
NPV @ 8%	USDm	237.1
NPV @ 10%	USDm	182.9
NPV @ 12%	USDm	143.3
IRR	%	43.1%
Payback Period	year	3.4
After Tax		
Total Project Cash Flows	USDm	671.6
NPV @ 8%	USDm	184.3
NPV @ 10%	USDm	140.5
NPV @ 12%	USDm	108.4
IRR	%	36.2%
Payback Period	year	3.8

SENSITIVITY ANALYSIS

Table 72 presents the key project sensitivities for. Ranked in order of sensitivity (highest to least sensitive), they are:

- Selling Price
- Operating Expenses
- Capital Expenditure

TABLE 72: IRR SENSITIVITY

Item assessed	Effect on Project IRR	New Project IRR
No changes (reference point)		36.2%
Operating expenses up 10%	-3.6%	39.8%
Selling Price down 10%	-5.8%	30.4%
Capital expenditure up 10%	-2.9%	33.3%

22.3.1 Spider Sensitivity Analysis

Capital expenditure, operating expenses and selling price sensitivities on project IRR are shown graphically below for both pre-tax and post-tax analyses.

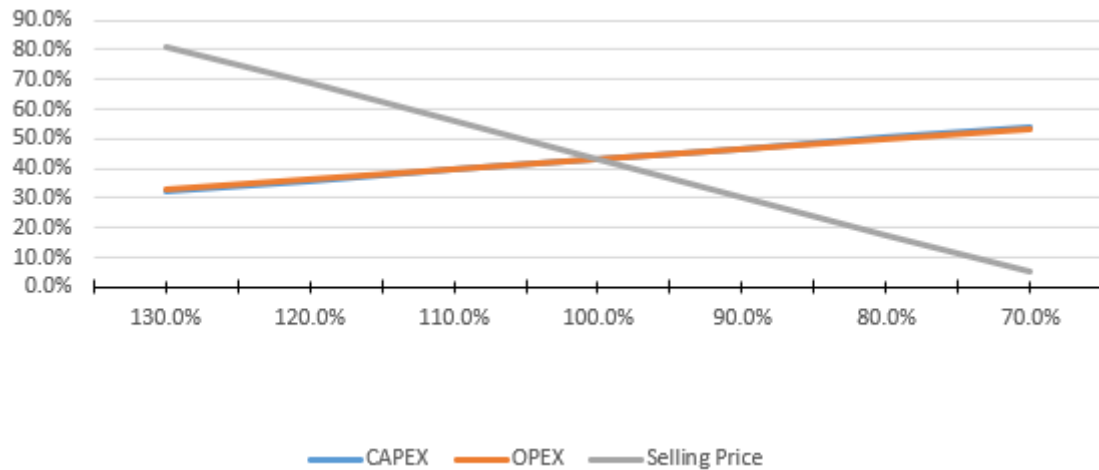


FIGURE 91: SENSITIVITY OF PROJECT IRR (PRE-TAX) TO CAPEX, OPEX AND PRICE

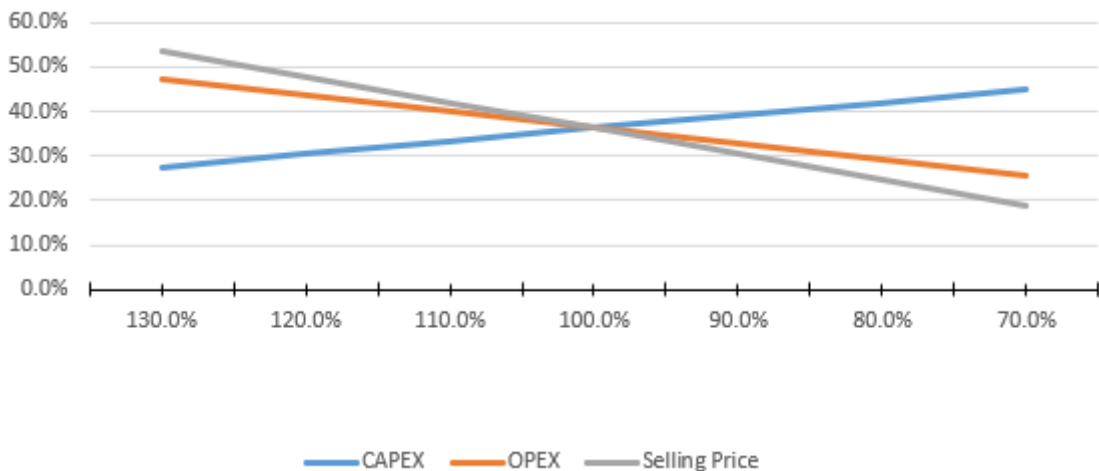


FIGURE 92: SENSITIVITY OF PROJECT IRR (POST-TAX) TO CAPEX, OPEX AND PRICE

Similar to the above, capital expenditure, operating expenses and selling price sensitivities on project NPV (8%) are shown graphically below for the post-tax analyses.

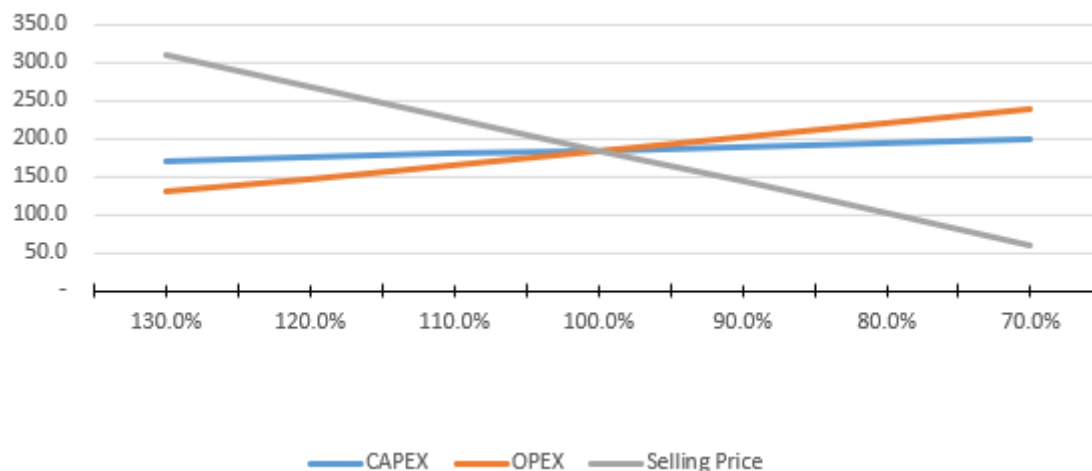


FIGURE 93: SENSITIVITY OF PROJECT NPV (8) (POST-TAX) TO CAPEX, OPEX AND PRICE

23 ADJACENT PROPERTIES

No similar graphite deposits are known in south-western Madagascar other than those already detailed in this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any other information on the properties that would affect their interpretations, or conclusions regarding the Molo Graphite Deposit. It should, however be noted that Molo is only one of nineteen graphite prospects on the greater Property, only seven of which have been drill tested.

24.1 REPORT ON THE HAZID (HAZARD STUDY 2) STUDY CARRIED OUT ON THE PROPOSED MOLO GRAPHITE PLANT IN MADAGASCAR

Two Hazid reports, namely, Hazid on Graphite Dust Exposure and Hazid on Silica Dust Exposure, were completed by Globesight, based on the Hazard study for the proposed project.

24.1.1 Executive Summary

A new graphite mine and processing plant is to be established in the southern part of Madagascar on behalf of the Company. The plant will process ROM ore and produce pure graphite material for export. The process consists essentially of the following steps:

- Crushing
- Milling and flash flotation
- Rougher flotation
- Cleaner circuits

- Thickening
- Filtration and drying
- Bagging
- Tailings handling
- Reagent storage and supply
- Supply of services

Rod Prior, an experienced Professional Process Safety Engineer, was asked to facilitate the Hazard and Operability (HAZOP) study on the new plant. The HAZOP study was replaced by a HAZID study because of time constraints. HAZID is a higher-level technique which does not examine every pipeline.

The HAZID study was held on the 21 April 2017 and included representatives from the designers, consultants and the owners. A standard HAZID worksheet was used where the possibility of various hazardous events was considered. Where an event was considered credible an analysis followed.

The analysis considered the following:

- Possible causes
- Consequences
- Preventative avoidance measures
- Protective mitigating measures
- Actions needed if the measures were inadequate

The analysis is a form of structured brainstorming with a high reliance on experience.

There were no fundamental issues which required rectification. There are several issues associated with setting up a facility in a remote part of the country. However, the risk of a graphite dust explosion in the Drying and Bagging section of the plant could be significant and requires a detailed study (HAZOP and Risk assessment) to ensure adequate controls are put in place. Attention needs to be paid to workforce training and supervision as these workers will be inexperienced and unfamiliar with industrial practices. There is considerable scope for human error in many aspects of the operations as the plant is not highly automated.

A full HAZOP should be done on the plant when the design is fully completed. Vendor HAZOPs are needed because of the large number of vendor packages used.

24.1.2 Introduction

A plant to produce graphite is to be built in Southern Madagascar. The feed will be high quality graphite ore mined in an adjacent mine. The plant will use standard processing techniques but will use a novel concept of importing the plant in modules. The Company will own and will operate the facility. The plant has been designed by SYNC of China with MET 63 providing process engineering support.

Initially, a HAZARD AND OPERABILITY (HAZOP) Study was requested to identify the safety, health, environmental and operability hazards and additional controls needed to ensure that the risks posed by the hazards are acceptable. To HAZOP 39 Piping and Instrumentation Drawings would require 3 days. It was agreed that a higher-level Hazard Study HAZID (Hazard Identification) or Hazard Study 2 would be carried out on the 18 Process Flow Diagrams. This could be done in a day. HAZID is effectively a structured brainstorming approach aimed at identifying significant hazards and associated controls.

Rod Prior, an experienced Process Safety Engineer, was invited to facilitate the HAZID. The HAZID sessions were held on the 21 April 2017 at the Waterfall Club in Johannesburg.

The mining operations were first discussed with Dave Thompson, mining consultant, presenting the information.

The High-Level Risk and Opportunity Analysis carried out in 2016 was reviewed by the HAZID team and a number of changes were made. Some of the risks were typical HAZID risks.

24.1.3 Process Description

24.1.3.1 *Ore Receiving and Crushing*

The crusher circuit is fed with ROM ore using a frontend loader at a rate of 50tph. The material is screened before being fed to a jaw crusher. Size is reduced to < 120mm. The crusher product is fed to a classification screen. Various fractions are fed to secondary and tertiary crushers. Some material is recycled. A single stream is fed to the Milling Section

24.1.3.2 *Primary Milling and Flash Flotation*

Material is fed to the primary mill (ball) where size is reduced to 80%<500 μ . Coarse material at -2mm is fed to the flash flotation cell where diesel and MIBC are added. Froth (concentrate) and tailing streams are produced.

24.1.3.3 *Rougher Flotation*

The rougher flotation cell bank concentrates the graphite stream. Reagents are added as before.

24.1.3.4 *Primary Concentrate Cleaning Circuit*

Concentrate is passed through the primary polishing mill to the primary cleaner flotation cell where further concentration takes place.

24.1.3.5 *Fine Flake Cleaning Circuit*

Concentrate is passed through the fine flake polishing mill where flakes are separated rather than reduced in size. A flotation cell produces further concentration. Tailings from the cell are subjected to cleaner and scavenging flotation cells.

24.1.3.6 *Attrition Cleaning Circuit*

The cleaner concentrate stream is thickened in the attrition concentrate thickener. The underflow product is processed in the attrition mill. The material is further concentrated in a flotation cell. Tailings are processed in cleaner and scavenging flotation cells.

24.1.3.7 *Final Tailings Handling*

Tailing streams are combined and processed in the final tailings thickener. The underflow (tailings) into a plate and frame filter. The filter cake is stockpiled and moved to a dump via frontend loader.

24.1.3.8 *Final Concentrate Handling Circuit*

Combined concentrate is thickened in a thickener. The underflow is dried on a filter belt to produce a filter cake with <20% moisture. The filter cake is dried in a diesel-fired filter to a moisture content of <1%. The concentrate is screened into four size fractions before being bagged.

24.1.3.9 *Services*

Water is pumped from a well-field. Two water systems feed the plant. Dirty water is supplied to various process applications. Clean water is produced via a water purification plant and is available for drinking purposes and other applications.

Compressed air from two compressors for instrument air duty and filtration. A blower provides low pressure air for the flotation cells.

Diesel and MIBC are stored in small isotainers and pumped to the process.

Flocculant powder is made up into solution in a mixing tank. It is pumped to the two thickeners via dosing pumps.

Coagulant powder is made up into solution in a mixing tank. It is fed into the three thickeners via dosing pumps.

Electricity is generated on site by a diesel driven generator. There is a standby generator to supply a partial supply.

24.1.4 **Hazid Procedure**

The HAZID procedure identifies safety, health, environmental and operability hazards and associated controls through a structured questioning approach triggered by considering possible incident scenarios. The procedure is normally applied before the detailed design is available and is based on examining the PFDs (Process Flow Sheets). In this study, the HAZID approach was chosen because of limited time available to do a full HAZOP

The following hazardous events (scenarios) were used in the HAZID Study.

EXTERNAL FIRE
INTERNAL FIRE
INTERNAL EXPLOSION
CONFINED EXPLOSION
UNCONFINED EXPLOSION
RUPTURE/BURST
INTEGRITY LOSS
ACUTE HARMFUL/NOXIOUS EXPOSURE
CONTAMINATION / COMPOSITION
SPILLS
POLLUTION
NOISE
VIOLENT RELEASE OF ENERGY
CORROSION / EROSION
POOR QUALITY
MOMENTUM CHANGE
UNAVAILABILITY
OPERABILITY PROBLEM
HUMAN FACTORS
SECURITY BREACH
FINANCIAL PROBLEM

For each hazardous event (scenario), the possibility of the event occurring in the section is considered. This comes from experience or an understanding of the situation. Should a credible scenario be agreed then the following steps are followed:

- 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

24.1.5 Assumptions

It was assumed that South African Standards for the Plant would be at least equal, if not higher, than that required in Madagascar.

24.1.6 Recommendations

NODE TITLE	REC NO	ACTION REQUIRED
Ore receiving and Crushing	1	Specify appropriate lube oil to minimize fire risk
	2	Dust need to be analysed particularly for harmful silica content
	3	Refer to ESIA Report for information on spills, pollution & noise
	4	Mine plan for Grade control to be followed
	5	Check that there is sufficient design allowance for loader impact on crusher equipment.
	6	Apply practical solutions (common sense) to load the crushers when wet material found
	7	Use HR auditing to check that local inexperienced employees are performing tasks correctly
	8	Develop Security Protocols to deal with theft & other issues
Primary Milling and Flotation	9	All dust spillage to be contained in bund areas
	10	Ops Procedures to specify PPE and medical tests for Mill noise
	11	Confirm Mill design (handrails, barrier etc.) from a safety/access point of view
	12	Additional testing by SGS to confirm that flow blockages will be minimized
Rougher Flotation	13	Lightning and surges to be catered for in design
Cleaner Flotation	14	Fire detection in the MCC to be linked to the incomer trip
	15	PPE requirements to be specified for both flocculants
Tailings Thickening	16	Consider spillage compartments in process water tank
	17	Ensure that the PM system covers the abrasion effect of coarse silica
	18	Review functional spec of the thickener with respect to the auto rake lift
	19	To prevent drownings, consider signage, training and a buddy system
Tailings Filtration	20	Check vendor package for hydraulic pack for filter having an interlock between pressure release and power feed
	21	Evaluate wet tailings
	22	Additional training on filter on filter operation (reliability issue)
Concentrate Filtration	23	Consider layout change as very close to Drier (fire risk)

NODE TITLE	REC NO	ACTION REQUIRED
	24	Emphasize cleaning of filter cloths in training and procedures (blinding)
Concentrate Drying and Bagging	25	Measures to prevent fires & explosions in the burner (Burner Management System) must be reviewed in the Vendor HAZOP
	26	The risk of a major graphite dust explosion must be thoroughly explored in a specific HAZOP (enclosed building)
	27	Establish the health risk implications (PPE etc.) of graphite dust inhalation
Water System	28	Review of the potable treatment philosophy and ops manuals
Compressed Air Circuits	29	Conduct vendor HAZOP studies on this and all other vendor packages
Reagent Storage & Dosing (Diesel & MIBC)	30	Review storage location and layout to minimize any external fires leading to tank heating
	31	Develop a diesel management plan
	32	Develop a security strategy to deal with theft (diesel and others)
Flocculant Storage & Dosing	33	Conduct vendor HAZOPs on these vendor packages
	34	Emphasize the slipping risk in spillages for operators in training

24.1.7 HAZID Conclusions

The HAZOP team could identify a limited number of hazards and operating issues that required attention. No major safety or operability issues were recorded.

A significant uncertainty was detected with respect to the heat exchanger that cools the motors. The drawings provided were unable to show how the devices worked and need to be examined further with the vendor.

25 INTERPRETATIONS AND CONCLUSIONS

25.1 GEOLOGY

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.

25.2 MINING

Mineral reserves of 22 436 685 tonnes (Proven - 14 169 741 T at 7.00% C and Probable - 8 266 944 T at 7.04% C) have been declared for the Molo 2019 720ktpa FS at an average grade of 7.02% C and based on the information contained in the FS it is possible to economically mine this deposit.

25.3 TAILINGS

Due to the substantially reduced tonnages for the project as envisaged, tailings will be dried and co-disposed with the waste rock generated as part of the open cast mining. Despite this co-disposal approach, a detailed design has been completed, complete with environmental and social impact assessment and closure to allow for the upgrade to a more conventional, cyclone facility, should the throughput be increased during the life of the mine. This approach has been pursued to ensure that sufficient flexibility is built into the project development strategy to accommodate the anticipated increase in market demand.

25.4 RISKS

In addition to the qualitative risk assessment completed during the Molo 2015 FS, a comprehensive HAZID study was completed as part of this study.

25.5 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.

25.6 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.

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.

26 RECOMMENDATIONS

26.1 GEOLOGY

No further recommendations.

26.2 MINING

The Project will allow for potential optimisation of drilling and blasting designs during execution that could reduce operating costs slightly.

From a pure mining perspective, the Project is very small and provided reasonable levels of short-term planning are applied it should have very few challenges in delivering the required tonnages at the required grade to meet the production targets set out in this study.

26.3 METALLURGICAL TEST WORK

The following recommendations are made for the detailed engineering stage:

- Investigate the metallurgical impact of different attrition mill technologies such as stirred media mills or attrition scrubbers;
- Evaluate a range of different grinding media (e.g. different size, shape, material) to determine if flake degradation can be reduced without affecting the concentrate grade;
- Develop a grinding energy versus concentrate grade relationship for the best grinding media. This will allow a more accurate prediction of the required attrition mill grinding energy as a function of the final concentrate grade;
- Conduct attrition mill vendor tests to aid in the sizing of the equipment;
- Carry out vendor testing on graphite tailings using the optimized reagent regime proposed by the reagent supplier.

- Complete a series of flotation tests on samples covering mine life intervals for the 2017 FS pit design.

26.4 RECOVERY METHODS

The process plant has been designed to easily optimize the final product grade, this is achieved by having two options in the attrition cleaning step. It is however recommended that additional laboratory testwork be conducted to test the current plant configuration for treatment for higher feed grade material.

26.5 INFRASTRUCTURE

The following are recommended prior to the detailed design stage:

- Additional geotechnical investigations at the proposed new construction and permanent camp site, particularly at the location of the new potable water storage tanks
- A detailed geotechnical investigation will need to be undertaken to identify and confirm suitable sources of concrete aggregate and concrete sand materials at the location of the project site. This testing will need to include for concrete material testing and the production of concrete trial mixes with the material identified
- The geotechnical information will also need to confirm the suitability for construction of all the material to be excavated from the Return Water Dam. It is proposed that all the material excavated from the Return Water Dam is utilised in the works as processed fill material
- Confirmation as to whether the material from the proposed borrow pit near Fotadrevo (which will be used to supply all fill material for the TSF starter wall construction) can be utilised as fill material, or if this material can be stabilized in some manner and used in the works
- A detailed topographical survey will need to be undertaken of the proposed construction site, borrow pit areas and the access road between Fotadrevo and the mine site. This information is required prior to the final detailed design of the plant layout and associated earthworks

26.6 WATER

The following is recommended during the detailed design phase:

- 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. Additionally, proposed 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

- The installation of a weather station on the Project site should be done as soon as possible.
- Quantitative and predictive water balance, groundwater and geochemical analyses should be undertaken on regular intervals in order to update the water management plan.

26.7 ENVIRONMENTAL, SOCIAL

- The installation of a suitable weather station at or as near as possible to the proposed project site, even before construction commences, is recommended. Accurate, local weather data is almost non-existent in Madagascar. This data will prove invaluable for model calibration, improvement in baseline understanding and for future energy supply options which could utilise wind and or solar power generation.
- Clean and or renewable energy supply should be considered as a medium to long term target.
- Appointment of a community representative and the establishment of a mandate to sensitise the local communities prior to any project activities.
- Monitoring and auditing to commence at project preparation phase.
- Compilation of Standard Operating Procedures for Environmental and Social aspects requiring direct management and intervention.
- It is recommended that actual activity data, (e.g. kilometres travelled, or litres of diesel consumed) for a financial year is used when a GHG Assessment is being calculated. Given that this project involves an estimation of a future GHG assessment for activities yet to begin, a series of assumptions have been made in order to obtain the activity data required to undertake this calculation.
- Community recruitment, skills development and training should begin at project preparation phase.

26.8 PERMITTING

- Security of land tenure is a process and is estimated to take 6-9 months, thus this process should be commissioned as early as possible.
- Compilation of a comprehensive legal register.

The permitting and licensing of the proposed Project requires dedicated attention to ensure consistent momentum in application for and delivery of permits and licenses. This is extremely relevant within the Malagasy context.

26.9 WORK PROGRAM

A monthly payment schedule, as well as cost estimate with timing, is recommended in **Error! Reference source not found.** and **Error! Reference source not found.**

TABLE 73: RECOMMENDED MONTHLY PAYMENT SCHEDULE

Month	0	1	2	3	4	5	6	7	8	9	10	11	12
Buildings						\$577,304.72	\$288,652.36	\$288,652.36					
Civil & Infrastructure						\$1,051,836.10	\$525,918.05	\$525,918.05					
Construction Services		\$663,100.00	\$35,000.00	\$35,000.00	\$35,000.00	\$304,175.11	\$35,000.00	\$35,000.00	\$297,500.00	\$35,000.00			
Electrical Infrastructure							\$128,804.10						
Environmental & Permitting costs			\$377,748.00		\$312,961.06						\$39,118.94		
Indirect Costs		\$41,416.00	\$41,416.00	\$41,416.00	\$41,417.00	\$41,417.00	\$41,417.00	\$41,417.00	\$41,417.00	\$41,417.00			
Mining							\$2,574,143.85						
Owner's Costs	\$119,700.00	\$119,700.00	\$119,700.00	\$119,700.00	\$119,700.00	\$119,700.00	\$119,700.00	\$119,700.00	\$119,700.00	\$119,700.00			\$299,250.00
Process Equipment	\$2,250,000.00	\$2,250,000.00					\$2,250,000.00	\$938,609.00	\$750,000.00				\$4,500,000.00
Project Services	\$78,720.00	\$78,720.00	\$52,290.00	\$141,292.00	\$111,292.00	\$111,292.00	\$114,292.00	\$114,292.00	\$114,292.00	\$5,000.00	\$10,000.00		\$157,439.95
Sub-total	\$ 2,448,420	\$ 3,152,936	\$ 626,154	\$ 337,408	\$ 620,370	\$ 2,205,725	\$ 6,077,927	\$ 1,124,979	\$ 1,511,518	\$ 951,117	\$ 49,119	\$ -	\$ 4,956,690
Contingency (10%)	\$244,842.00	\$315,293.60	\$62,615.40	\$33,740.80	\$62,037.01	\$220,572.49	\$607,792.74	\$112,497.94	\$151,151.80	\$95,111.70	\$4,911.89		\$619,586.24
Working Capital												\$1,033,333.33	\$1,033,333.33
TOTAL	\$ 2,693,262	\$ 3,468,230	\$ 688,769	\$ 371,149	\$ 682,407	\$ 2,426,297	\$ 6,685,720	\$ 1,237,477	\$ 1,662,670	\$ 1,046,229	\$ 54,031	\$ 1,033,333	\$ 6,609,610
Cumulative	\$ 2,693,262	\$ 6,161,492	\$ 6,850,261	\$ 7,221,410	\$ 7,903,817	\$ 10,330,114	\$ 17,015,834	\$ 18,253,312	\$ 19,915,982	\$ 20,962,210	\$ 21,016,241	\$ 22,049,574	\$ 28,039,598
Month	13	14	15	16	17	18	19	20	21	22	23	24	25
Buildings						\$865,957.08	\$432,978.54	\$432,978.54					
Civil & Infrastructure						\$2,278,672.21	\$1,139,336.10	\$1,139,336.10					
Construction Services	\$994,650.00	\$52,500.00	\$52,500.00	\$52,500.00	\$456,262.67	\$52,500.00	\$52,500.00	\$446,250.00	\$52,500.00				
Electrical Infrastructure						\$257,608.20							
Environmental & Permitting costs		\$377,747.94		\$312,961.01						\$39,118.93			
Indirect Costs	\$82,832.00	\$82,832.00	\$82,832.00	\$82,834.00	\$82,834.00	\$82,834.00	\$82,834.00	\$82,834.00	\$82,834.00				
Mining										\$2,339,197.53			
Owner's Costs	\$299,250.00	\$299,250.00	\$299,250.00	\$299,250.00	\$299,250.00	\$299,250.00	\$299,250.00	\$299,250.00	\$299,250.00				
Process Equipment	\$4,500,000.00					\$4,500,000.00			\$1,877,218.00	\$1,500,000.00			
Project Services	\$157,439.95	\$104,579.96	\$282,583.91	\$222,583.93	\$222,583.93	\$228,583.92	\$228,583.92	\$228,583.92	\$10,000.00	\$19,999.99			
Sub-total	\$ 6,034,172	\$ 916,910	\$ 717,166	\$ 970,129	\$ 1,060,931	\$ 8,565,405	\$ 2,235,483	\$ 2,629,233	\$ 4,661,000	\$ 1,559,119	\$ -	\$ -	\$ -
Contingency (12.5%)	\$754,271.49	\$114,613.74	\$89,645.74	\$121,266.12	\$132,616.32	\$1,619,289.68	\$279,435.32	\$328,654.07	\$582,624.94	\$194,889.87			
Working Capital	\$1,033,333.33										\$1,400,000.00	\$1,400,000.00	\$1,400,000.00
TOTAL	\$ 7,821,777	\$ 1,031,524	\$ 806,812	\$ 1,091,395	\$ 1,193,547	\$ 10,184,695	\$ 2,514,918	\$ 2,957,887	\$ 5,243,624	\$ 1,754,009	\$ 1,400,000	\$ 1,400,000	\$ 1,400,000
Cumulative	\$ 35,861,374	\$ 36,892,898	\$ 37,699,710	\$ 38,791,105	\$ 39,984,652	\$ 50,169,347	\$ 52,684,265	\$ 55,642,151	\$ 60,885,776	\$ 62,639,785	\$ 64,039,785	\$ 65,439,785	\$ 66,839,785

TABLE 74: COST ESTIMATE AND TIMING

Time (months from financing)	0-1	1-4	5-7	8-10	11-13	14-17	18-20	21-22	23-25
Financing required	\$ 2,693,262	\$ 5,210,555	\$ 10,349,495	\$ 2,762,929	\$ 15,464,720	\$ 4,123,277	\$ 15,657,500	\$ 6,997,633	\$ 4,200,000
Milestone	Detailed Design	Plant procurement	Module shipping	VAT	3 months working capital	Expansion plant procurement	Expansion module shipment	Expansion VAT	3 months working capital for expansion
	Long-lead Procurement	Plant Fabrication	Site construction	Final payments	Expansion Detailed Design	Expansion plant fabrication	Expansion site construction	Final expansion plant payment	
	Detailed construction planning	Plant Assembly	Mining fleet	1 yr enviro monitoring		Expansion plant assembly	Expansion mining fleet	1 yr environmental monitoring	
		FAT				Expansion FAT			
		First fill consumables				Spares allowance			
		Spares allowance							
		Relocation Action Plan							
		Permitting							

27 REFERENCES

27.1 GEOLOGY

Aussant, C.H., Scherba, C. and Maynard, R. (2010). Summary Report for the (April to July 2009 and September to December 2009) Exploration Programs on the Three Horses Property February 2010.

Besairie, H. (1964). Madagascar, Carte Géologique 1/1,000,000. In three sheets, Antananarivo.

Besairie, H. (1967). The Precambrian of Madagascar. IN: Rankama (Ed). The Precambrian, 3. Wiley Interscience, London, 133-142.

BRGM (1985). Plan Directeur d'Actions Pour la Mise en Valeur des Ressources du Sol et du Sous-Sol de Madagascar, Direction des Mines et de la Géologie, Contrat d'étude No 01/84/MIEM-DME/FED, Three volumes: Premier Phase-Premier partie 573 pp; Deuxieme partie 418 pp; Deuxieme Phase 146 pp.

Collins, A.S. (2006). Madagascar and the amalgamation of Central Gondwana. Gondwana Research 9, 3-16.

Desautels, P., McCracken, T. and Holloway, A. (2011). Technical Report Update NI 43-101, Fotadrevo, Province of Toliara, Madagascar. Report for Energizer Resources by AGP Mining Consultants Inc, 247 pp.

De Wit, M.J., Bowring, S.A., Ashwal, L.D., Randrianasolo, L.G., Morel, V.P. and Rabeloson, R.A. (2001). Age and tectonic evolution of Neoproterozoic ductile shear zones in southwestern Madagascar, with implications for Gondwana studies. Tectonics 20(1), 1-45.

Fugro Airborne Surveys Corporation (2008). DIGHEM Survey for Madagascar Mining Investments, Three Horse Property, Tulear, Madagascar Map Sheets G59, C60, H59, H60, April 23, 2008, Report #07090.

Hancox, P.J. and Subramani, D. (2013). Molo Graphite Project, Fotadrevo, Province of Toliara, Madagascar Technical Report NI 43-101. Report for Energizer Resources, 121 pp.

Kogel, J.E., Trivedi, N.C., Barker, J.M., Krukowski, S.T. (2006) Industrial minerals & rocks: commodities, markets, and uses (7th edition). Society for Mining, Metallurgy, and Exploration (U.S.), Littleton, Colorado, 379 pp.

Levinson, A.A. (1974). Introduction to Exploration Geochemistry. Applied Publishing, Wilmette, IL, 924 pp.

Martelat, J-E., Lardeaux, J-M., Nicollet, C and Rakotonirafy, R. (2000). Strain pattern and late Precambrian deformation history in southern Madagascar. Precambrian research 102, 1-20.

McCracken, T. and Holloway, A. (2009). Technical Report Update NI 43-101, Fotadrevo, Province of Toliara, Madagascar. Report for Energizer Resources.

Paquette, J.L., Nedelec, A., Moine, B. and Rakotondrazafy, M. (1994). U-Pb, Single Zircon Pb-Evaporation, and Sm-Nd Isotopic Study of a granulite Domain in SE Madagascar. *Journal of Geology* 102, 523-538.

Pitfield, P., Bauer, W., Schofield, D., Tusky, T. and Randriamananjana, T. (2006). Reappraisal of the geology and structural evolution of the Precambrian basement in north and east-central Madagascar. *Geophysical Research Abstracts* 8.

Pili, E., Ricard, Y., Lardeaux, J.M. and Sheppard, S.M.F. (1997). Lithospheric shear zones and mantle-crust connections. *Tectonophysics* 280, 15-29.

Rakotondrazafy, A. M. R., Giuliani, G., Ohnenstetter, D., Fallick, A.E., Rakotosamizanany, S., Andriamamonjy, A., Ralantoarison, T., Razanatseheno, M., Offant, Y., Garnier, V., Maluski, H., Dunaigre, C., Schwarz, D. and Ratrimo, V. (2008). Gem Corundum Deposits of Madagascar: A Review. *Ore Geology Reviews* 34: 134-154.

Razafindralambo G., Gaylord L. (2005). Madagascar: National Environmental Action Plan, MfDR Principles in Action: Sourcebook on Emerging Good Practices, pp. 75-82.

Scherba, C.S. and Chisholm, R.E. (2008). Geological Evaluation of the Three Horses Property. Report prepared for Uranium Star Corporation; 81 pp.

27.2 MINING

Hancox, P.J. and Subramani, D. (2013). Molo Graphite Project, Fotadrevo, Province of Toliara, Madagascar- Technical Report NI 43-101. Report for Energizer Resources, 121 pp.

L Cilliers/ M Steinmann – Open House Management Solutions (Pty) Ltd. Geotechnical Engineer Report for Energizer Resource.

Molo production schedule model molo_fs6_2017_rt

Molo Model Note_10-03-2015

27.3 PROCESS

Brodzik, P. (2014). Dry Screening of Graphite - Molo Concentrator Project. Derrick Corp. test report prepared for Energizer Resources.

Brodzik, P. (2014). Wet Screening of Graphite - Molo Concentrator Project. Derrick Corp. test report prepared for Energizer Resources.

Chegwidden, A. (2014). Molo 750 kpta Graphite Processing Plant Drill Core Sample Selection for Metallurgical Variability Test work. Technical Note C8375-PROC-TW-0001.

Chegwidden, A. (2014). Molo 750 kpta Graphite Mine Feasibility Study Concentrator Flowsheet Refinement. Technical Note C8375-PROC-TW-0002.

Daubermann, R. (2014). Flocculant Screening Tests on Slurry Sample as Provided. Senmin report 1408002.

Ford, E., Mudau, P., and Sithole, V. (2014). Bench Scale Comminution Testwork on graphite Composites from Molo Deposit, Madagascar. Mintek External Report 7039 prepared for SGS Canada.

Grammatikopoulos, T. (2015). An Investigation into High Definition Mineralogy into the Mineralogical Characteristics of Seven Metallurgical Samples from the Molo Graphite Deposit, Madagascar. SGS Report 14119-005, MI5018-JAN15.

Grammatikopoulos, T. (2015). An Investigation into High Definition Mineralogy into the Mineralogical Characteristics of Six Metallurgical Samples from the Molo Graphite Deposit, Madagascar. SGS Report 14119-005, MI5018-JAN15.

Grammatikopoulos, T. (2013). The Mineralogical Characteristics of a Low and High Grade Samples from the Molo Graphite Deposit, Madagascar. SGS Report 14119-001 MI5010-JUN13.

Ho, D. (2014). Filtration Test Report 111376T1. Outotec test report prepared for Energizer Resources.

Ho, D. (2014). Thickening Test Report 111376T1. Outotec test report prepared for Energizer Resources.

Lee, N. (2014). Sedimentation and Filtration Tests for Molo Deposit Project. FLSmidth laboratory testing report.

Mortimer, J. (2014). Preliminary Assessment of BASF Solid-Liquid Separation Reagents for Energizer Graphite Tailings. BASF Mining Solutions technical service summary report.

Peters, O., Imeson, D. (2015). An Investigation into the Process Development for Two Composites from the Molo Graphite Prospect. SGS Report 14119-001.

Peters, O., Imeson, D. (2014). An Investigation into the Recovery of Graphite from a 200 Tonne Bulk Sample from the Molo Deposit. SGS Report 14119-002.

Peters, O., Imeson, D. (2015). An Investigation into the Process Optimization and Variability Testing of Molo Ore Composites. SGS Report 14119-005 - Report 1.

Peters, O., Imeson, D. (2015). An Investigation into the Variability Testing of Molo Ore Composites and Concentrate Upgrading. SGS Report 14119-005 – report 2.

Peters, O. (2014). Summary of Preliminary Metallurgical Results of Variability Composite Testing. Metpro Management Report for DRA Global.

Peters, O. (2014). Summary of Metallurgical Results of Variability Composite Testing and Concentrate Upgrading. Metpro Management Report for DRA Global.

Restifo, C. (2014). Laboratory Test Report 103055. Heyl & Patterson test report prepared for Energizer Resources.

Teme, K., Mahlabe, N., Haumann, E. Lombard, A., and Mhlanga, N. (2012). Mineralogical and Metallurgical Scoping Study on Graphite Bearing Ore Deposit from Green Giant Project in Madagascar. Mintek External Report 6399 prepared for Energizer Resources.

27.4 INFRASTRUCTURE

27.4.1 Plant Geotechnical Information

SRK Report - 474874_Molo_Geotechnical Investigation - Proposal, March 2014 (Final).

27.4.2 Water

ASTM International, 2010. Standard Guide for Application of a Groundwater Flow Model to a Site-Specific Problem, D5447-04. United States.

British Geological Survey (BGS), 2011. Groundwater Productivity Map of Africa. NERC, United Kingdom.

Davies, J., 2008. Chapter 14 Hydrogeology, In: British Geological Survey, United States

Geological Survey and Cabinet GLW Conseil, 2008. Republique de Madagascar Ministère de L'Energie et des Mines (MEM/SG/DG/UCP/PGRM); Projet de Gouvernance des Ressources Minérales (PGRM).

GCS, 2014. Molo Graphite Project - ESIA: Hydrogeology Report. GCS Report No. 13-723.

GCS, 2014. Molo Graphite Project - ESIA: Hydrology Report. GCS Report No. 13-723.

GCS, 2014. Molo Graphite Project - Feasibility Study: Climate Report. GCS Project No.13-723.

Hammarstroma, J.M., Seal Ila, R.R., Meierb, A.L. Kornfeldc, J.M. (2005). Secondary sulfate minerals associated with acid drainage in the eastern US: recycling of metals and acidity in surficial environments. Chemical Geology 215 (2005) 407 – 431.

Heath, R.C., 1983. Basic Ground-Water Hydrology. United States Geological Survey (USGS) Water Supply Paper: 2220. USGS, United States of America.

Kottek, M., Grieser, J., Beck, C., Rudolf, B. and Rubel, F. (2006) World Map of the Köpen-Geiger Climate Classification Updated. Gebrüder Borntraeger. Meteorologische Zeitschrift,15 (3), 259-263.

Miller, S., Robertson, A. and Donahue, T. (1997). Advances in Acid Drainage Prediction using the Net Acid Generation (NAG) Test. Proc. 4th International Conference on Acid Rock Drainage, Vancouver, BC, 0533-549.

Nimbol (2014). Data obtained from Madagascar Meteorology Department.

NOAA [National Oceanic and Atmospheric Administration]. (2013) Downloadable Climate.

Satellite Data. United States Department of Commerce. Available from www.noaa.gov.

Palandri, J.L. and Kharaka, Y.K. (2004). A compilation of rate parameters of water-mineral interaction kinetics for application to geochemical modeling. USGS Open file report 2004-1068.

Price, W.A. (1997). DRAFT Guidelines and Recommended Methods for the prediction of Metal leaching and Acid Rock Drainage at Minesites in British Columbia. British Columbia Ministry of Employment and Investment, Energy and Minerals Division, Smithers, BC, p.143.

Rudnick, R.L. and Gao, S. (2003). The Composition of the Continental Crust. In: The Crust (ed. R.L. Rudnick), Treatise on Geochemistry (eds. H.D. Holland and K.K. Turekian). Elsevier-Pergamon, Oxford. 3, 1-64.

Soregaroli, B.A. and Lawrence, R.W. (1998). Update on Waste Characterization Studies. Proc. Mine Design, Operations and Closure Conference, Polson, Montana.

UNESCO, 2008. Groundwater Resources of Africa Map. BGR Hannover/UNESCO Paris.

Van Tonder, G., Kunstmann, H. and Xu, Y., 1998. Estimation of the Sustainable Yield of a Borehole Including Boundary Information, Drawdown Derivatives and Uncertainty Propagation. University of the Free State, Bloemfontein.

World health organisation (2011). Guidelines for drinking-water quality - 4th ed. http://whqlibdoc.who.int/publications/2011/9789241548151_eng.pdf?ua=1

Zahrai, S.K., Madden, M.E.E., Madden, A.S., Rimstidt, J.D. (2013). Na-jarosite dissolution rates: The effect of mineral composition on jarosite lifetimes. *Icarus* 223 (2013) 438–443.

27.4.3 Tailings and Plant

SANS 1200. (2007). Standardized Specification for Civil Engineering Construction (All Parts). South African Bureau of Standards.

South African Institute of Engineering Geologists, South African Institute of Civil Engineers, Geotechnical Division, 2nd Impression (2002). Guidelines for Soil and Rock Logging in SA.

TRH14:1985. Guidelines for Road Construction Materials. Committee for State Road Authorities, Department of Transport, Pretoria.

TMH1:1979. Standard Methods of Testing Road Construction Materials. National Institute for Transport & Road Research C.S.I.R. Pretoria.

World Health Organisation Regional Office for Africa. We site <http://www.who-eatlas.org/africa/countries/madagascar/madagascar-seismic-map.html>. Accessed 19 June 2014.

27.5 MARKET STUDIES

Benchmark Mineral Intelligence. (2017a). Glossary, Part 2: Battery Raw Materials, Graphite (<http://benchmarkminerals.com/glossary/#2>).

Benchmark Mineral Intelligence. (2017b). Rise of anode megafactories and the impact on graphite, Graphite Supply 2017 Conference presentation.

Benchmark Mineral Intelligence. (2019). Graphite pricing subscription service. (<https://www.benchmarkminerals.com>).

BP Global. (2017). Energy Economics, The impact of electric cars on oil demand (<https://www.bp.com/en/global/corporate/energy-economics/energy-outlook/electric-cars-and-oil-demand.html>).

Clarke, M. (2017). Welcome to Graphite Supply Chain 2017, Graphite Supply 2017 Conference presentation.

Industrial Minerals. (2017). Graphite, Natural graphite supply (<http://www.indmin.com/Graphite.html>).

International Energy Agency, Clean Energy Ministerial. (2017). Global EV Outlook 2017: Two million and counting (<https://www.iea.org/publications/freepublications/publication/GlobalEVOutlook2017.pdf>).

Moore, S., O'Driscoll, M., and Russell, R. (2012). Natural Graphite Report 2012: London, Industrial Minerals, 394 p.

Olson, D., 2014, U.S. Geological Survey. (2014). Minerals Yearbook, Graphite (<https://minerals.usgs.gov/minerals/pubs/commodity/graphite/myb1-2014-graph.pdf>).

Olson, D.W. (2017). Graphite, Natural, U.S. Geological Survey, Mineral Commodity Summaries, January 2017 (<https://minerals.usgs.gov/minerals/pubs/commodity/graphite/mcs-2017-graph.pdf>).

Organization of the Petroleum Exporting Countries. (2017). World Oil Outlook, Critical Uncertainties (<https://woo.opec.org/index.php/critical-uncertainties>).

Reuters. (2017). BHP, World's largest miner, says 2017 is 'tipping point' for electric cars (<https://www.reuters.com/article/us-bhp-strategy/bhp-worlds-largest-miner-says-2017-is-tipping-point-for-electric-cars-idUSKCN1C10HO>).

Robinson, G.R. Jr., Hammarstrom, J.M., and Olson, D.W. (2017). Graphite, chap. J, in Schulz, K.J., DeYoung, J.H., Jr., Seal, R.R. II, and Bradley, D.C., eds., Critical mineral resources of the United States – Economic and environmental geology and prospects for future supply: U.S. Geological Survey Professional Paper 1802, p. J1-J24, <https://doi.org/10.3133/pp1802J>.

Roskill Consulting Group Ltd. (2017). NextSource Materials Inc. graphite market study and price forecast, p. 122.

Scherba, C.S., Montreuil, J.F., and Barrie, C.T. (2018). Geology and economics of the giant Molo graphite deposit, southern Madagascar: Society of Economic Geologists, Special Publications, no. 21, pp. 347-363.

Taylor, H.A. Jr. (2006). Graphite, in Kogel, J.E., ed., Industrial Minerals & Rocks: Commodities, Markets, and Uses: New York, Society for Mining, Metallurgy, and Exploration (U.S.), p. 507-518.

27.6 ENVIRONMENTAL, SOCIAL, PERMITTING

Agetipa, 2014. Molo Project Queries by GCS on April 11, 2014. Madagascar.

Agetipa, 2014. Permits and Stakeholders Register. Madagascar.

Agetipa (2015). Molo Graphite Project: Draft Resettlement Action Plan, Madagascar.

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.

Equator Principals III, 2013. A financial industry benchmark for determining, assessing and managing environmental and social risk in projects. Available from http://www.equator-principles.com/resources/equator_principles_III.pdf

GCS Water and Environment (GCS), 2015. Molo Graphite Project - Environmental and Social Impact Assessment. South Africa.

GCS Water and Environment (GCS), 2013. Madagascar Environmental Legal Protocol. South Africa.

International Council on Mining and Metals (ICMM), 2008. Planning for Integrated Mine Closure: Toolkit. Available from <http://www.icmm.com/document/310>.

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/>.

28 **CERTIFICATES OF AUTHOR**

ERUDITE CERTIFICATE OF AUTHOR
STRATEGIES | PROJECTS | ENGINEERING

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Johann de Bruin

9 George Storrar Drive, Groenkloof, Tshwane, 0181, South Africa

Director

b) Title and Effective Date of Technical Report:

Molo Feasibility Study, National Instrument 43-101 Technical Report

Effective date: 31 May 2019.

c) 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)

d) Site Inspection:

I visited the site on various dates in 2015, 2016 and 2017.

e) Responsibilities:

I was responsible for the following: Sections 1.1 to 1.3, 1.4, 1.8, 1.10, 1.11, 1.12, 1.13 to 1.17 to 1.22, Section 2, Section 3, Section 15, Section 16, Section 17, Section 18, Section 19, Section 20, Section 21, Section 22, Section 24, Section 25.2, 25.4 to 25.6, Section 26.2, 26.4 to 26.9, Section 27.2, 27.4 to 27.5.

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I have no prior involvement with this Project.

h) 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.

i) 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: 31 May 2019



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 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Oliver Peters

Metpro Management Inc., 102 Milroy Drive, Peterborough, Ontario, K9H 7T2, Canada

Principal Metallurgist

b) Title and Effective Date of Technical Report:

Molo Feasibility Study, National Instrument 43-101 Technical Report

Effective date: 31 May 2019

c) Qualifications:

I graduated with a M.Sc. degree in mineral processing from the RWTH Aachen University.

I am a Registered Professional Engineer with the Professional Engineers of Ontario – Reg. No. 100078050

I have completed various graphite studies as summarized below. I have practiced my profession from 1998 to now.

I have read the definition of “qualified person” set out by National Instrument 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purpose of NI 43-101. My relevant experience includes 17 years as a mineral processing engineer and project manager.

d) Site Inspection:

I did not visit the site

e) Responsibilities:

I was responsible for the following sections: 1.9, 1.18.3, 1.19.3, Section 13, Sections 25.3, 26.3, 27.3

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I was responsible for managing the all laboratory and pilot scale programs at SGS Lakefield in the role of a consulting metallurgist.

h) 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.

i) 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: 31 May 2019



Oliver Peters

Principal Metallurgist

Summary of selected recent graphite experience:

Year	Client	Type Of Study	Project Description
2011	Northern Graphite	Laboratory	Process development
2011	Northern Graphite	Pilot Plant	Validation of flowsheet, collection of engineering data
2012/2013	Focus Graphite	Laboratory	Process development, variability study
2012	Focus Graphite	Pilot Plant	Validation of Flowsheet, collection of engineering data
2012	Magnesita	Laboratory	Process development, variability study
2012	Mason Graphite	Laboratory	Process development
2013	Canada Carbon	Laboratory	Process development
2013	Graphite One	Laboratory	Process development
2014	Alabama Graphite	Laboratory	Process development
2014	Sovereign Metals	Laboratory	Process development
2014	Nouveau Monde	Laboratory	Process development
2014	Dalgraphite	Laboratory	Process development
2014	Canada Carbon	Pilot Plant	Validation of Flowsheet, collection of engineering data
2013-2015	NextSource Materials	Laboratory	Process development, variability study
2014	NextSource Materials	Pilot Plant	Validation of Flowsheet, collection of engineering data

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 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Sivanesan (Desmond) Subramani

30 7th Avenue, Parktown North, Randburg, Gauteng, South Africa

Principal Resource Geologist

b) Title and Effective Date of Technical Report:

Molo Feasibility Study, National Instrument 43-101 Technical Report

Effective date: 31 May 2019

c) Qualifications:

I graduated with a B.Sc. Honours (Geology and Economic Geology) degree from the University of KwaZulu Natal - Durban in 1994.

I am a member in good standing of the South African Council for Natural Scientific Professions (SACNASP No. 400184/06) as well as a Member of the Geological Society of South Africa.

I have practiced my profession from 1995.

I have read the definition of “qualified person” set out by National Instrument 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purpose of NI 43-101. My relevant experience includes 19 years as a Geologist, of which the last 8 years focused on geostatistical resource modelling and estimation.

d) Site Inspection:

I visited the site 15th to 19th February 2014.

e) Responsibilities:

I was responsible for the following sections: Section 1.6, Section 12, Section 14.

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I was responsible for the sections 12 and 14 of the previous National Instrument 43-101, filed on 12th April 2013

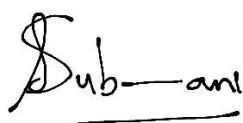
h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) 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: 31 May 2019



Desmond Subramani

Principal Resource Geologist

Summary of recent experience:

Year	Client	Commodity	Type Of Study	Project Description
2005 - 2006	Katanga Mining	Copper & Cobalt	PFS	Generated Cu and Co Mineral Resources for a Prefeasibility study used in the listing on the Toronto Stock Exchange - DRC
2007	Anglo Platinum	Platinum Group Metals	PFS	Completed PGE Mineral Resource update for Ga Pasha project prefeasibility study – South Africa
2007 - 2008	Lonmin Platinum	Platinum Group Metals	Internal Report	Provided technical advice and supervision in generating mineral resources for Baobab Platinum mine. – South Africa
2011	INV Metals	Copper	Resource Estimate	Mineral Resource Estimations, plus Ni43-101 for Okohongo Cu prospect - Namibia
2011	Nkwe Platinum	Platinum Group Metals	FS	Mineral Resource Estimations for their Garatau PGE project, as part of a Feasibility Study. – South Africa
2011-2014	AngloGold Ashanti	Gold	Annual Resource Report	Annual update and reconciliation of resources for Siguiri Gold Mine. – Guinea Basao.
2011-2012	Metorex Group	Copper	PFS	Update Mineral Resources for Kinsenda Copper project, as part of an ongoing feasibility study. - DRC
2012/2014	NextSource Materials	Graphite	PFS	Mineral Resource Estimations for their Molo graphite project, as part of a Feasibility Study - Madagascar
2013/2014	Helio Resources	Gold	PFS	Mineral Resource Estimation, as part of the Ni43-101 for their SMP deposit. - Tanzania.

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 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Philip John Hancox

30 Seventh Avenue, Parktown North, Randburg, Gauteng, South Africa

General Manager

b) Title and Effective Date of Technical Report:

Molo Feasibility Study, National Instrument 43-101 Technical Report

Effective date: 31 May 2019

c) 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.

I have read the definition of “qualified person” set out by National Instrument 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purpose of NI 43-101. My relevant experience includes 17 years as a Geologist, of which the last 11 years focused on exploration.

d) Site Inspection:

I visited the site during May of 2012 and 2013.

e) Responsibilities:

I am responsible for the following sections: Sections 1.5, 1.7, 1.18.1, 1.19.1, Section 4, Section 5, Section 6, Section 7, Section 8, Section 9, Section 10, Section 11, Sections 25.1, 26.1, 27.1

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I was responsible for part of Item 1, Items 2 to 13 and items 23 to 27 of the previous National Instrument 43-101, filed on 12th April 2013.

h) 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.

i) 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: 31 May 2019



Philip John Hancox (Pr.Sci.Nat.)

General Manager Africa and Australasia

Caracle Creek International Consulting (Pty) Limited

ERUDITE CERTIFICATE OF AUTHOR
STRATEGIES | PROJECTS | ENGINEERING

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Pono Mogoera, Process Engineer
24 Adele Street, Kempton Park, 1619

b) Title and Effective Date of Technical Report:

Molo Feasibility Study, National Instrument 43-101 Technical Report
Effective date: 31 May 2019

c) Qualifications:

Pono is a registered Professional Engineer (Pr.Eng.) with ECSA with a Bachelor of Science degree in Chemical Engineering with an additional post graduate certificate in Management Advancement Program (Wits Business School).

These qualifications are complemented by over 14 years of working experience in the process engineering field across various sectors. Pono has an array of experience that ranges from the operations of mineral processing to working extensively in the project and consulting space.

Pono has previously work on two other graphite Project in the Southern African space.

d) Site Inspection:

I have not visited the site.

e) Responsibilities:

I am responsible for the following sections: Section 1.10, Section 17.1, section 18.2, Section 21.2

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

This is the first time I have been involved with this Project.

h) 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.

i) 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: 31 May 2019



Pono Mogoera (Pr Eng)

Process Engineer

Erudite Project (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 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Etienne van Heerden (Pr.Eng)
Civil and Structural Engineer
9 George Storrar Drive, Groenkloof, Pretoria

b) Title and Effective Date of Technical Report:

Molo Feasibility Study, National Instrument 43-101 Technical Report
Effective date: 31 May 2019

c) Qualifications:

Etienne is a registered Professional Engineer (Pr.Eng.) with a Bachelor of Science degree in Civil Engineering. Etienne also holds a master's degree in engineering management.

I am a Professional Engineer registered with Engineering Council of South Africa (ECSA) – 20080017.

These qualifications are complemented by over 19 years of working experience across various sectors in the civil and structural engineering industry. Etienne has an array of experience that range from the construction management to the various stages of design development of complex industrial projects. Etienne has worked on mining, petrochemical and infrastructure development projects across the African continent.

d) Site Inspection:

I have not inspected the site

e) Responsibilities:

I am responsible for the following sections:

- i. 18.2 Plant Infrastructure
- ii. 18.7 Shared Infrastructure and Services
- iii. 18.7.1 Access Roads
- iv. 18.7.2 Terraces and Bulk Earthworks
- v. 18.7.3 Gate House and Turnstile Access Control
- vi. 18.7.4 Training Centre

- vii. 18.7.5 Re-fuelling Station
- viii. 18.7.6 Waste Water Treatment Plant
- ix. 18.7.7 Portable Water Treatment Plant
- x. 18.7.8 Storm Water Drainage
- xi. 18.7.9 Storm Water Dams
- xii. 26.5 Infrastructure

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I was involved with the previous feasibility study for this project.

h) 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.

i) 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: 31 May 2019



Etienne van Heerden (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 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Hercules Albertus Smit
9 George Storrar Drive, Groenkloof, Tshwane 0181
Senior Electrical Engineer

b) Title and Effective Date of Technical Report:

Molo Feasibility Study, National Instrument 43-101 Technical Report
Effective date: 31 May 2019

c) Qualifications:

B.Eng Electrical Degree	University of Pretoria	1996
B.Eng Electrical Degree (Hons)	University of Pretoria	2001
Professional Engineer (20000338)	Engineering Council of SA	2000

d) Site Inspection:

I have not attended any site inspections

e) Responsibilities:

I was responsible for the following sections:

- i. 2.5 Source of Information
- ii. 18.8 Power Generation Facility
- iii. 18.8.1 Generator Loading
- iv. 18.8.2 Control & Instrumentation
- v. 20.3 Environmental and Social Sensitivities
- vi. 20.3.3 Electrical Supply

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

Intermittently involved since 2017.

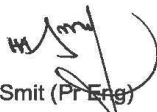
h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with same.

i) 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: 31 May 2019



H.A Smit (Pr Eng)

Senior Electrical Engineer

Erudite Strategies (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 30 June 2011, Part 8.1.

a) Name, Address, Occupation:

Albertus Wynand Christoffel (Alkie) Marais, Principal Hydrogeologist
56 Jerome Road, Lynnwood Glen, Pretoria, 0081

b) Title and Effective Date of Technical Report:

Molo Feasibility Study, National Instrument 43-101 Technical Report
Effective date: 31 May 2019

c) Qualifications:

Albertus is a specialist in groundwater services and has 23 years of experience and specialises in Mining related hydrogeological investigations, Groundwater contaminant studies, Mine dewatering studies and design, Flow and contaminant transport modelling, Hydro-geochemical waste studies, Environmental management studies, Groundwater monitoring programmes and hydrochemical analyses, Groundwater resource, aquifer assessments and Water-balance studies.

I am a registered scientist with the South African Council for Natural Scientific Professionals (SACNASP) (Registration number 400012/06)

MSc (Geohydrology), IGS (UFS) Bloemfontein, SA, 1997,
BSc (Hons)(Geology), UFS Bloemfontein, SA, 1993,
BSc (Geology, Geochemistry), UFS Bloemfontein, SA, 1992

d) Site Inspection:

March 2014.

e) Responsibilities:

- i. 17.3 Water
- ii. 17.3.4 Conceptual Storm Water Management Plan
- iii. 1.20 Environmental & Permitting
- iv. 1.20.1 Environmental and Social Impact Assessment
- v. 1.22 Recommendations
- vi. 1.22.7 Environmental, Social
- vii. 1.22.8 Permitting
- viii. 18.5 Raw Water Supply
- ix. 20 Environmental Studies, Permitting and Social Impact
- x. 20.8 Waste and Water Management
- xi. 20.8.1 Mine Waste Management

- xii. 20.8.2 Surface Water Environment
- xiii. 20.8.3 Groundwater Environment
- xiv. 20.8.4 Water Management
- xv. 27.6 Environmental, Social, Permitting

f) Independence:

I am independent of the Company in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

Involved with original Energizer Resources Study for the Molo project 2014 – 2015.

h) 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.

i) 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: 31 May 2019



Albertus W.C. (Alkie) Marais

Principal Hydrogeologist

GCS Water and Environmental Consultants (Pty) Ltd.