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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

Prepared by DRA Projects (Pty) Limited for

ENERGIZER RESOURCES INC.

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Effective Date: 6 February 2015

IMPORTANT NOTICE

This notice is an integral component of the Molo 2015 Feasibility Study ("Molo 2015 FS") technical report and should be read in its entirety and must accompany every copy made of the Molo 2015 FS. The Molo 2015 FS has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43 101").

The Molo 2015 FS has been prepared for Energizer Resources Inc. ("Energizer") by Caracle Creek International, GCS Water and Environment("GCS"), Agetipa SRK Consulting Inc ("SRK"), Metpro Management Inc., and DRA Projects (Pty) Ltd ("DRA") as the Report Contributors. The Molo 2015 FS is based on information and data supplied to the Report Contributors by Energizer and other parties. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in the services of the Report Contributors, based on:

(i) Information available at the time of preparation.

(ii) Data supplied by outside sources, and

(iii) The assumptions, conditions, and qualifications set forth in this report.

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The results of the Molo 2015 FS represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Molo 2015 FS. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

The conclusions and estimates stated in the Molo 2015 FS are to the accuracy stated in the Molo 2015 FS only and rely on assumptions stated in the Molo 2015 FS. The results of further work may indicate that the conclusions, estimates, and assumptions in the Molo 2015 FS need to be revised or reviewed.

The Report Contributors have used their experience and industry expertise to produce the estimates and approximations in the Molo 2015 FS. Where the Report Contributors have made those estimates and approximations, they are subject to qualifications and assumptions and it should also be noted that all estimates and approximations contained in the Molo 2015 FS will be prone to fluctuations with time and changing industry circumstances.

The Molo 2015 FS should be construed in light of the methods, procedures, and techniques used to prepare the Molo 2015 FS. Sections or parts of the Molo 2015 FS should not be read in isolation of, or removed from their original context.

Date and Signature Page

This report titled "Molo Feasibility Study - National Instrument 43-101 Technical Report" was prepared on behalf of Energizer Resources Inc. by the following authors:

Effective date: 6 February 2015

John Hancox, PhD (Geology), General Manager - Africa and Australasia, Caracle Creek International Consulting (Pty) Limited, was responsible for: 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.Signed 23 March 2015.

Desmond Subramani, B.Sc. Honours (Geology), Principal Geologist – Mineral Resource Estimation, Caracle Creek International Consulting (Pty) Limited, was responsible for: Section 1.6, Section 12, Section 14. Signed 23 March 2015.

Dave Thompson, B.Tech (Mining), Mining Consultant, DRA Projects (Pty) Ltd, was responsible for: Sections 1.8, 1.18.2, 1.19.2, Section 15, Section 16, Sections 25.2, 26.2, 27.2. Signed 23 March 2015.

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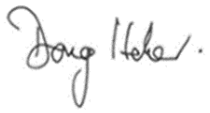

John Stanbury, B.Sc. (Industrial Engineering), Director, Cresco Project Finance, was responsible for: Sections 1.4, 1.12, 1.16, Section 19, Section 22, Section 25.10. Signed 23 March 2015.

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Abbreviations, Symbols, Acronyms and Units of Measure

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
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
Dry metric tonnes	dmt
Dry metric tonnes per hour	dmt/h
Environmental Commitment Permit	RIM
Feet	‘ or ft
Free air delivery	FAD
Greater than	>
Greater than or equal to	≥

Grams	g
Grams per tonne	g/t
Hectares	ha
High density polyethylene	HDPE
Hours	h
Inductively coupled plasma (assay)	ICP
Intermediate bulk container	IBC
Internal diameter	ID
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
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
Original equipment manufacturer	OEM
Particle size distribution where 80% of particles in a stream are larger than the size indicated	P_{80}
Parts per million	PPM
PEG Mining Consultants Inc.	PEG
Percent	%
Percent graphitic carbon	% C(g)
Percent total carbon	% C(t)
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
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
Universal Transverse Mercator	UTM
Uranium Star Minerals SARL	USM
Whole Rock Analysis	WRA
X-Ray Fluorescence	XRF

1 SUMMARY

1.1 Introduction

Energizer Resources Inc. (formerly Uranium Star Corp.) ("Energizer" or the "Company") is a mineral exploration and development Company based in Toronto, Canada, which is currently focused on the exploration and development of its 100% owned, flagship Molo Graphite 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 Bekikiy 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 is one of several surficial graphite trends discovered by 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 Preliminary Economic Assessment (the "PEA"), which was undertaken by DRA Mineral Projects in 2013. The positive outcome of this assessment lead Energizer to undertake another phase of exploratory drilling and sampling in 2014, which was done under the supervision of Caracle Creek International Consulting (Pty) Limited ("Caracle Creek" or "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). Caracle Creek 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 forms the basis for this Molo 2015 FS.

1.2 Project Location

The Molo deposit is located some 160 km SE of Madagascar's administrative capital (and port city) of Toliara, in the Tulear region and about 220 km NW of Fort Dauphin. It is approximately 13 km NE of the local village of Fotadrevo. See Figure 1 below.

Figure 1: Project Location

1.3 Project Description

The proposed development of the Molo Graphite Project includes the construction of a green fields open pit mine, a processing plant with a capacity of 862,000 tonnes of ore per annum and all supporting infrastructure including water, fuel, power, tailings, buildings and permanent accommodation.

The mine will utilize four 2 megawatt diesel generators, with three running and one standby and water is supplied from a well field which has been defined by drilling and detailed geo-hydrological modelling. The processing plant will consist of conventional crushing, milling and flotation circuits followed by concentrate

filtering, drying and screening. The waste heat generated by the power station will be utilized for the drying of the concentrate.

The tailings storage facility, in the form of a valley dam layout, is located approximately 1.5 kilometers to the west of the process plant and is designed to accommodate the run-of-mine tonnage for the 26 year Life of Mine (LOM)**.**

See Figure 2 below for the proposed layout of the site.

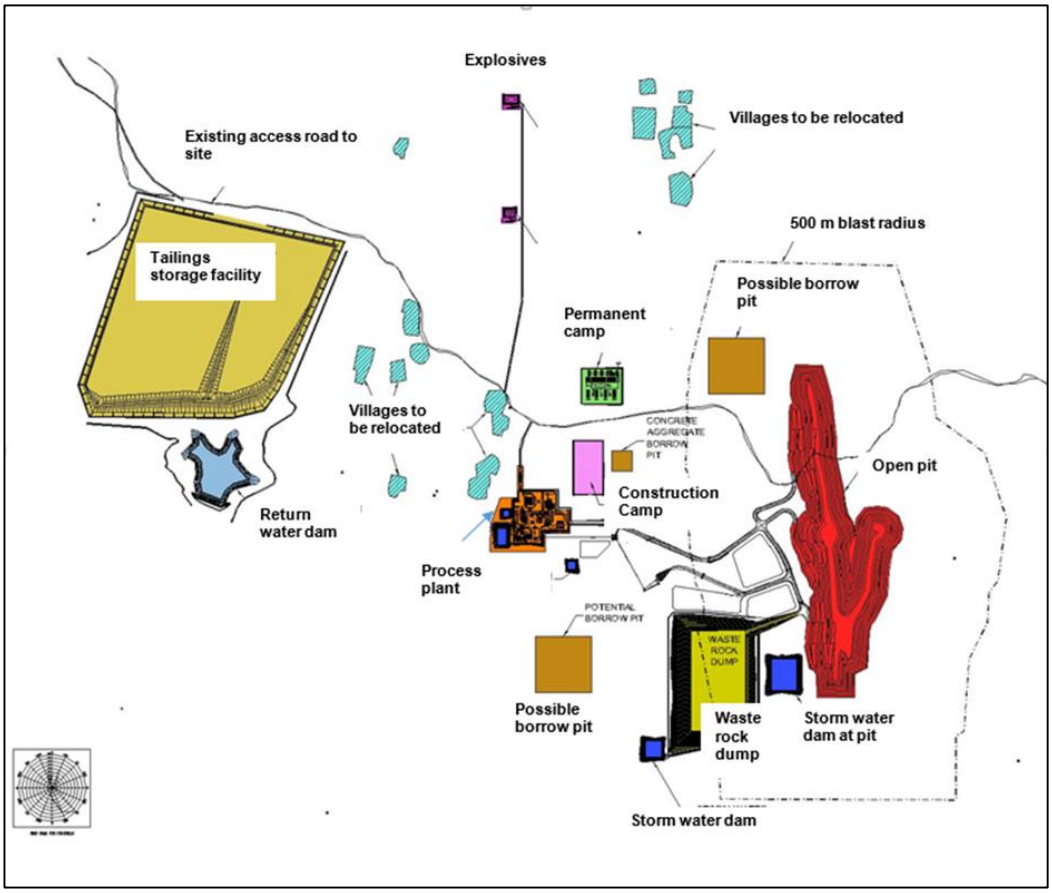


Figure 2: Site layout

1.4 Summary of financial results

Table 1 below summarizes the financial results of the Molo 2015 FS. These are based on a discounted flow analysis of the project using nominal cash flows, which include the effect of inflation.

Table 1: Summary of Financial Results

1	Post-tax: NPV (10% Discount Cash Flow)(1)(2)	US$389,797,113
2	Post-tax: IRR (1)(2)	31.2%
3	Payback (2)	4.84 years
4	Capital cost (“CAPEX”)	US$149.9 million
5	Design Development Allowance (to cover potential quantity and rate changes during detailed design and execution)	US$13.8 million
6	Owners Contingency	US$24.6 million
7	On-site Operating Costs (“OPEX”) per tonne of concentrate, (year 3 onward)	US$353
8	Transportation per tonne of concentrate (from mine site to Madagascar Port year 3 onward)	US$182
9	Transportation per tonne of concentrate (from Madagascar Port to European Customer Port from year 3 onward)	US$155
10	Average annual production of concentrate	53,017 tonnes
11	Life of Mine (“LOM”)	26 years
12	Graphite concentrate sale price (US$/tonne at Start Up - 2017)	US$1,689 per tonne
13	Average Head Grade	7.04%
14	Average ore mined per annum over Life of Mine	856,701 tonnes
15	Average stripping ratio	0.81:1
16	Average carbon recovery	87.80%

Notes

- Note 1: Assumes project is financed with a 50% debt and 50% equity.
- Note 2: Values shown are based on nominal cash flows, which include the effect of inflation. Costs are increased on an annual basis by the relevant inflation index.

Table 2 below summarizes key mine and process data.

Table 2: Mine & Process Data

Proven reserves	14,170,000	Tonnes @ 7.0% C grade
Probable reserves	8,367,000	Tonnes @ 7.04% C grade
Grade (graphitic carbon)	7.04%	Average plant head feed over LOM
Waste to ore ratio	0.81:1	
Processing rate	856,701	Tonnes per annum
Mine life	26 years	
Recovery	87.8%	
Average annual product tonnes	53,017	

1.5 Property Description and Ownership

1.5.1 Property description

The Molo Graphite Project is contained in a portion of Exploration Permit #3432**.** The Project is centred on UTM coordinates 413,390 Easting 7,345,713 Northing (UTM 38S, WGS 84 datum). The Molo Graphite Project is located 11.5 km ENE of the town of Fotadrevo and covers an area of 62.5 hectares (“ha”). The Government of Madagascar designates individual claims by a central LaBorde UTM location point, comprising a square with an area of 6.25 km^2.

1.5.2 Ownership

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, 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").

The Molo Graphite Project is located within Exploration Permit #3432 as issued by the Bureau de Cadastre Minier de Madagascar ("BCMM") pursuant to the Mining Code 1999 (as amended) and its implementing decrees.

CCIC has 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 Energizer or one of its Madagascar subsidiaries**.**

Energizer holds the exclusive right to explore for a defined group of industrial minerals within the permits listed above. 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, and Zeolites.

Reporting requirements of exploration activities carried out by the titleholder on a Research 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. Upon establishment of a mineral resource, Research Permits may be converted into Exploitation Permits by application. CCIC is of the opinion that Energizer is compliant in terms of its commitments under these reporting requirements.

The Molo Graphite 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 Energizer positional data is collected in WGS 84, and if necessary converted back to LaBorde.

1.5.3 Royalties

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

1.5.4 Permits

Exploration Permit #3432 is currently held under the name of a subsidiary of Malagasy Minerals called, Mada-Aust Sarl. The transformation or amendment of exploration and research mining permits within the country continues to be suspended from the time that Madagascar was run by a president who was not democratically elected. This current permit expired on August 17, 2011.

Energizer's Madagascar domiciled subsidiary companies and Mada-Aust Sarl has continued to pay all taxes and administrative fees to the Madagascan government and its mining ministry with respect to all the mining permits held in country. These taxes and administrative fee payments have been acknowledged and accepted by the Madagascan government. In addition, Energizer continues to diligently work with the Madagascan government to obtain the necessary permits in its name as the country clears its backlog of applications and amendments.

The research permit will be converted into an exploitation permit in due course. When the permit is transformed from a research permit to an exploitation permit, the exploitation permit will be issued in the name of Energizer. The exploitation permit is required to advance the Molo Graphite Project to the developmental stage.

1.6 Mineral Resource Estimate

The Molo Graphite Project hosts the following resources:

- Measured Mineral Resource of 23.62 MT (Million tonnes) 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 the 14th August 2014. The Mineral Resources are classified according to the Canadian Institute of Mining, Metallury 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, Metallury 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	8.40	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, Metallurgy and Petroleum definitions.
- "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/m^3) was assigned to the mineralised zones for the resource tonnage estimation.

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

1.7 Exploration

No further exploration is currently planned.

1.8 Mineral Reserve Estimate

As a result of the Molo 2015 FS, the following maiden proven and probable mineral reserves are declared, see Table 4 below.

Table 4: Mineral Reserves

Category	Tonnage	C Grade (%)
Proven	14,170,000	7.00
Probable	8,367,000	7.04
Proven and Probable	22,437,000	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.

1.9 Metallurgical Test Work

The Molo 2015 FS is based on a full suite of metallurgical test work performed by SGS Canada Metallurgical Services Inc. (“SGS”) which is based 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 extraction has been successfully demonstrated in the laboratory and pilot plant campaigns.

The tests 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 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 of the first 5 years of operation is 87.8% based on the metallurgical response of composites using samples from all drill holes within the five year pit design. The average composition of the combined concentrate grade is presented in the table below.

The area composites were generated by splitting the footprint of the five year mine plan into five zones of approximately the same size. All drill holes within one specific zone were then combined to form an area composite. A total of fifteen area composites were generated for metallurgical evaluation, (five zones with three depth intervals per zone). All assays were completed using control quality analysis and cross checks were completed during the mass balancing process to verify that the results were within the estimated measurement uncertainly of up to 1.7% relative for graphite concentrate grades greater than 90% total carbon.

Table 5: Metallurgical Data - Flake Size Distribution and Product Grade

Product Size	Mass 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	97.2
+150 mesh (medium flake)	15.5	97.3
+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	Mass 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.

1.10 Recovery methods

The process design is based on an annual production capacity of 862 kilotonnes of plant feed material at a nominal head grade of 7.04% C(t) producing an estimated average of 53 kilotonnes per annum (ktpa) of final concentrate.

The ore processing circuit consists of three-stage crushing followed by primary milling and classification, a flotation separation and concentrate upgrading circuit, and graphite product and tailings effluent handling facilities.

The crushing circuit is designed to operate 365 days per annum for 24 hours per day at ±68% utilization and comprises a primary jaw crusher, a secondary cone crusher and a tertiary cone crusher in closed-circuit with a double-deck classifying screen. The crushed product (P80 of approximately13 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 91% utilization. A single stage primary ball milling circuit is employed, incorporating a closed circuit linear classifying screen and a scalping screen ahead of the mill. The scalping screen undersize feeds a single flash flotation cell before combining with the mill discharge material. Scalping and linear screen oversize are the feed to the primary mill. The primary ball mill size is 4.3m diameter (inside new liners) x 4.6m (EGL) with an installed motor power of 1000 kW.

Primary milling is followed by rougher flotation which, along with flash flotation, recovers the graphite to concentrate from the main stream. Rougher flotation employs seven forced-draught trough cells.

The primary, fine-flake and attritioning cleaning circuits upgrade the concentrate to the final product grade of above 94% C(t). Concentrate from the main stream feeds into the primary cleaning circuit consisting 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 65 mesh classifying screen, from where the large-flake oversize is directed to a high rate thickener located ahead of a final concentrate attritioning circuit. Primary cleaner classifying screen 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 mill, a column flotation cell and flotation cleaner/cleaner scavenger trough cells. Fine flake column concentrate merges with the +65 mesh primary cleaner classifying screen oversize as it feeds the attritioning circuit thickener. Attritioning circuit column concentrate comprises the final concentrate stream feeding the final concentrate thickener.

Combined rougher and cleaner flotation final tailings are pumped to a guard de-gritting cyclone installation ahead of a high rate final tailings thickener. Cyclone overflow feeds the thickener. Cyclone and thickener underflows combine and are pumped for final disposal to the tailings storage facility (“TSF”).

Thickened final concentrate is pumped to a linear vacuum belt filter for further dewatering before the filter cake is fed into a rotary kiln drying circuit.

A three-stage, twin stream sifting plant screens the dry concentrate (dryer product) into the pre-determined size classes. A bagging plant is employed to weigh, sample and bag the different size fractions discretely for loading into sea freight containers for shipment.

Chemical reagents are used throughout the primary recovery and upgrading processes. Diesel fuel collector and liquid frother are added to various points-of-use within the flotation circuits.

Diesel collector is pumped from the main tank farm to a bulk tank at the plant, 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 a plant reagent store in 1m^3 IBC’s, or 210 litre steel drums. The drums are collected by forklift as required and the contents pumped into a frother storage tank. A manifold system on the storage tank supplies multiple variable speed peristaltic pumps, which discretely pump the frother at set rates to the various points-of-use within the flotation circuits.

Flocculant powder (Magnafloc 919 and Magnafloc 24 for concentrate and tailings thickening facilitation respectively) 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, one each being dedicated to supplying the concentrate and tailings thickeners due to the flocculant

types required being different for each application. Variable speed peristaltic pumps discretely pump the flocculant at set rates to the thickeners' points-of-use.

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

Figure 3 below gives a high level overview of the project and Figure 4 below provides a block diagram depicting the basic process flow.

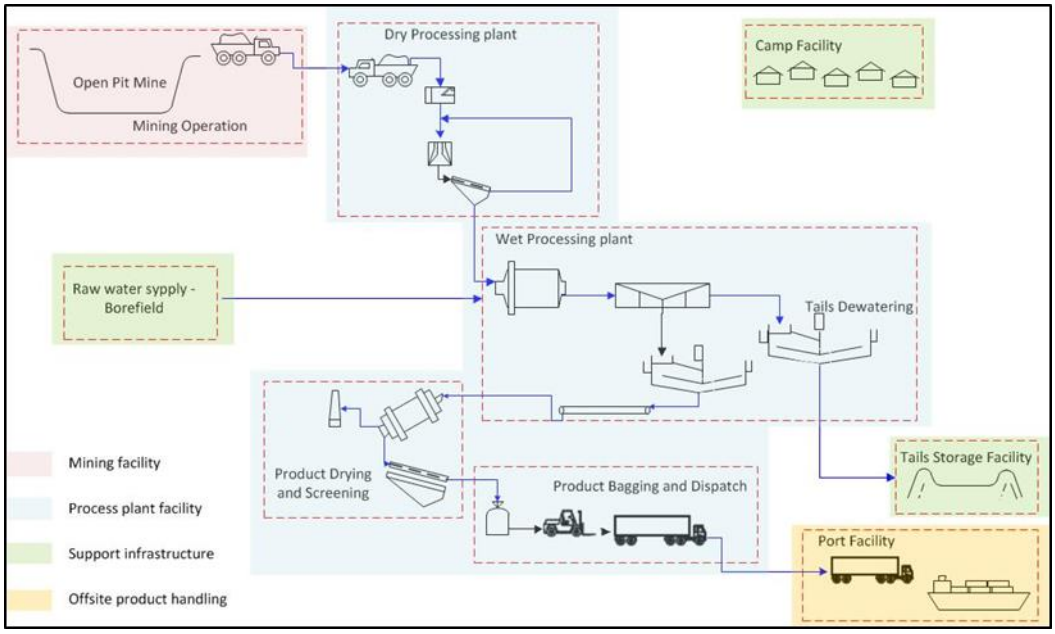


Figure 3: Project summary

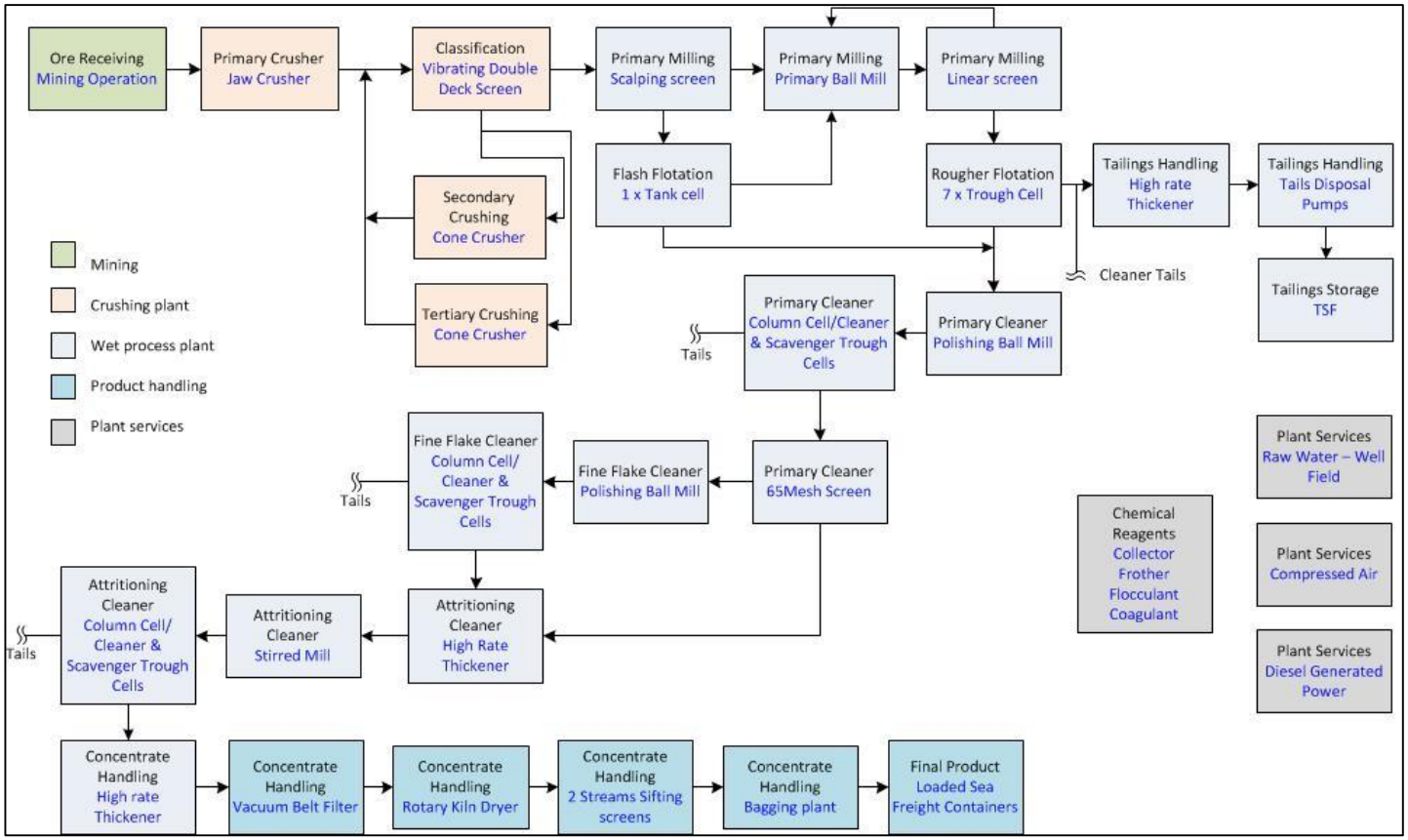


Figure 4: Block Flow Diagram

1.11 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 no infrastructure on site and everything 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
- Temporary accommodation during construction
- Permanent accommodation (340 people)
- All permanent buildings (offices, workshops, stores, laboratory)
- All buried services (potable water, sewage, stormwater, electrical reticulation)
- In plant roads
- Haul road
- Tailings Storage Facility
- Tailings pipe line to the TSF
- Return water pipe line from the TSF back to the plant
- Rock dumps and Run of Mine Ore (ROM) pads

See Figure 2 in section 1.3 for the site layout.

1.11.1 Raw Water Supply

Water is supplied by a net work of boreholes. A detailed water demand and supply analysis was done as part of the Molo 2015 FS, and this has shown that the water demands of the plant can be accommodated by boreholes within a radius of 5km from the plant. The daily steady state raw water make up requirement is estimated to be 561m^3 per day.

1.11.2 Power Supply

Power is supplied by four 2 MW diesel generators. The running load for the plant is estimated to be 2.7 MW with an additional 0.8 MW for the permanent camp and all mine infrastructure. Under normal operation there will only be two units running, with a third allowed to assist with mill starting, and the fourth unit as a spare for maintenance.

1.12 Product Pricing

Graphite prices are based on current quotes and projected estimates provided by UK-based Roskill Consulting Group Ltd ("Roskill"), recognized as a leader in providing independent and unbiased market research, pricing trends, and demand and supply analysis for the natural flake graphite market.

The weighted average price per tonne of graphite concentrate in December 2014 was US$1,375. This is a basket price and reflects the contribution of the different flake sizes and carbon grades to the overall price. The start-up price (in 2017 terms) for a tonne of Molo graphite concentrate is a projection based on Roskill information. The graphite price then escalates in the financial model based on Roskill's forecasts for supply and demand. The reader is cautioned that these are forecasts and may change subject to market dynamics.

1.13 Logistics

The cost to transport one tonne of dry concentrate (0.5% moisture content) from Molo to Rotterdam via Fort Dauphin, Madagascar, in December 2014 terms is 337 USD / tonne. This is based on shipping 26 tonnes of concentrate in 1 m^3 bags placed inside a 40 ft. container.

The route from Molo to Fort Dauphin runs either via the RN 10 or the RN 13. Both these routes vary from reasonable to 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.

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.

Figure 5 below shows a picture of the Port of Ehoala at Fort Dauphin.

Figure 5: Port of Ehoala at Fort Dauphin

Figure 6, Figure 7, and Figure 8 below give some insight into the varying road conditions between Molo and Fort Dauphin.

Figure 6: Road Conditions (1)

Figure 7: Road Conditions (2)

Figure 8: Road Conditions (3)

1.14 Capital costs

The capital cost for the project is estimated to be 188.2 million USD, including a contingency of 24.5 million USD. Competitive bids were obtained for most mechanical equipment, and for the earthworks, civils, structural steel, mechanical erection, piping and electrical, control and instrumentation detailed Bills of Quantities were issued for competitive pricing.

The base date for the capital costs is December 2014 and no provision has been made for escalation. The accuracy of capital costs is considered to be with +/- 10%

Table 7: Construction Capital Costs

Category	Cost (USD Million)
Capital Cost	149.9
Design Development Allowance	13.8
Subtotal	*163.7*
Contingency	24.5
Total	**188.2***
*Excludes taxes, tariffs, duties and interest	

Table 8: Initial Capital Cost Summary

Cost Centres	Cost (USD Million)
Pre-production	37.3
Tailings Storage Facility	24.3
Mechanicals	20.8
Electrical, Control & Instrumentation	20.8
External services	17.9
Earthworks	11.8
Piping	7.4
Structural	5.6
Transport	5.5
Vendor packages	3.4
Civil works	2.5

Consumables and spares	2.4
Buildings, fittings	2.1
Plate work	1.9
Total Capital Costs	**163.7**

Future capital expenditure expected to be incurred has been allowed for in the financial model to cover the expansion of the TSF in year 2, the replacement of the mine fleet, the replacement of the power plant, and for rehabilitation at the end of the project. Over the life of mine this accounts for an additional 38.3 million USD with 7 million USD spent on the TSF expansion in year 2.

1.15 Operating costs

The average cash operating costs from year 3 onwards, after the expatriate staff complement is reduced, are estimated to be 21.7 USD per mineralized tonne processed and 353 USD per tonne of concentrate produced. The cash operating costs includes mine operations (owner operated), process plant operations and general and administrative charges. They do not include shipping from Molo to the end user, or any downstream processing costs.

Table 9: Operating Costs per Tonne of Feed

Category	Year 3 onwards
Mining	US$3.9
Processing	US$11.0
General and Administrative	US$6.8
Total OPEX per Tonne of Feed	US$21.7
Costs have been rounded.	

Table 10: Operating Costs per Tonne of Concentrate produced at the Mine Site

Category	Year 3 onwards
Mining	US$64
Processing	US$179
General and Administrative	US$110
Total OPEX cost per Tonne of Concentrate at Mine Site	US$353
Costs have been rounded	

The operating costs expressed above are considered to be accurate to +/- 10%, and assume a varying USD inflation rate of 1.6% in 2015 and escalating to 2.0% from 2017 onward. Currency inflation rates were also considered in the financial model and were applied to the South African Rand and Malagasy Ariary portions of the opex costs.

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

1.16 Economic analysis

Table 11 below summarizes the economic analysis of the project using discounted cash flow methods.

Table 11: Economic Analysis of the Project

Category	Value
Average price / tonne of concentrate (at start up, 2017)	US$1,689
Internal Rate of Return ("IRR") - Project Equity	31.2%
NPV @ 8% Discounted Cash Flow	US$521,602,408
NPV @ 10% Discounted Cash Flow	US$389,797,113
NPV @ 12% Discounted Cash Flow	US$293,649,899
Project Payback Period	4.84 years
* Assumes that the project is financed through 50% equity finance and 50% corporate debt. The debt assumptions used in the model assumes a rate of 5.75% over LIBOR, with LIBOR forecast to escalate to 3.54% by 2022. An arranging fee is also assumed.	

Note

All values in the above table do not account for inflation and assume that a satisfactory investment agreement is negotiated under Madagascar's LGIM (Loi Sur les Grands Investissements Miniers) tax laws covering large scale mining investments, for which this project qualifies. Also included in the above table are forecasted prices for 2017, which coincides with the year the Molo mine is expected to be in production.

The exchange rates used in the financial model are as follows:

- 11.31 South African Rand ("ZAR") to US$1, moving in line with purchasing power parity
- 0.833 Euro to US$1, fixed for the modelled period
- 2,746 Malagasy Ariary ("MGA") to US$1, moving in line with purchasing power parity

1.17 Environmental & Permitting

A comprehensive Environmental and Social Impact Assessment ("ESIA") was completed to local Malagasy, Equator Principles, Word Bank and International Finance Corporation (IFC) standards. The process was preceded by an Environmental Legal Review and an Environmental and Social Screening Assessment; both providing crucial information to align the project development and design with international best practice on sustainable project development.

The ESIA submission is subject to approval of the investment amount by Madagascar's Ministry of Mines. The application was submitted on 30th January 2015 and the approval of the investment amount is in progress. Energizer will receive a Global Environmental Permit upon approval of the ESIA, a process which is expected to take six months from date of submission.

A comprehensive permitting register is in place and additional sectorial permit applications will form part of the early execution phase. Approval of the sectorial applications is expected within the same six month period as the ESIA review.

No material issues were identified in relation to Environmental, Social and Permitting processes and through the stakeholder engagement process the local and regional community has expressed a desire for the project to move forward.

1.18 Conclusions

1.18.1 Geology

Energizer'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, updated resource statement for the Molo deposit, which is stated in accordance with the CIM Guidelines.

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 from the Indicated to Measured Resource

category by additional infill drilling from the Indicated to Measured Resource category.

1.18.2 *Mining*

Maiden mineral reserves of 22 437 000 tonnes have been declared for the Molo Graphite Project at an average grade of 7.02% and based on the information contained in the Molo 2015 FS it is possible to economically mine this deposit.

1.18.3 *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.

1.18.4 *Recovery Methods*

The laboratory, pilot and vendor test work conducted prior to and during the study defined the required process flow sheet. This was duly translated into a full-scale production plant flow sheet as described within this report. The flow sheet unit processes were populated and individual component equipment selected according to either pilot plant precedents or, where these were not available, proven practice within the industry, in conjunction with suitably experienced vendors. All process designs and selections were based on conventional, proven mineral processing practices.

The processing selections and configurations built into the design are adequately suited to the requirements. Based on the mining and metallurgical test work information presented elsewhere within the Molo 2015 FS, and assuming within specification ROM (Run of Mine) ore is fed to the plant, the required recovery is expected to be attainable at the throughput stated. Note that this recovery is based on lab and pilot scale test work and may reduce slightly on a full scale plant due to operational inefficiencies. This possible reduction has not been taken into account in the financial analysis.

1.18.5 *Infrastructure*

All infrastructure required for the project has to be installed from scratch and has been allowed for in the project budget.

1.18.6 *Water*

The detailed hydrogeological analysis has concluded that the plant can be supplied from a well field.

1.18.7 *Environmental, Social*

A comprehensive Environmental and Social Impact Assessment has been done, and is in the process of being submitted to Madagascan government for approval.

1.18.8 *Permitting*

Various permits will have to be obtained for the project including an Environmental Permit and a Mining permit. The most urgent permit is for Energizer to renew the exploration permit covering the project.

1.18.9 *Tailings*

It is possible to construct the required tailings storage facility and a suitable site has been identified. Geochemical and hydrogeological test work has shown that the facility does not need to be lined.

1.18.10 *Risks*

The qualitative risk assessment identified 56 risks of which 9 were extremely high before controls. After controls were applied the number of extremely high risks was reduced to two. These risks are:

1. The exploration permit covering the Molo pit expired in 2011 and has yet to be officially renewed (Exploration Permit #3432 is the permit in question).
2. Current delays in issuing new mining permits.

After controls were applied the remaining high risks are as follows, (reduced from 39 to 18):

1. Requirement that all voids / excavations be backfilled without exception.
2. Inaccurate landownership data.
3. The unit costs of moving product are high.
4. Project NPV and IRR lower than the Preliminary Economic Assessment (PEA)
5. Theft during construction & operation (diesel, cable, etc.)
6. No off take agreements signed yet or formal product specifications received.
7. The current execution strategy calls for contracts to be placed before permits are granted.
8. The project has modelled the diesel price at 0.8 USD / litre.
9. ESIA review timeframes could extend past the planned project start date - indications are 6-9 months for ESIA approval from date of submission to the O.N.E (The Madagascan Government department of the Environment)
10. The process design may not achieve the optimal balance between the competing requirements of:
    a. Maximizing coarse flake recovery
    b. Maximizing product carbon grade
    c. Maximizing overall recovery
11. Future Land Claims (Ancestral Rights).

12. The process plant may not achieve a consistent on spec product, especially as the feed grade to the plant varies and this may make process control difficult.
13. Madagascan political situation remains potentially unstable.
14. Difficult logistics getting material on and off the island plus very bad roads.
15. Contractors P&G's high due to locality.
16. The projects returns are reliant on a real term increase in the price of graphite.
17. Implementation of the preferential taxation arrangement may be difficult.
18. The debt funding assumptions may not be achievable.

1.19 Recommendations

1.19.1 Geology

No further recommendations at present.

1.19.2 Mining

The long mine life of the Molo Graphite 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 Molo Graphite 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 Molo 2015 FS.

1.19.3 Metallurgical Test Work

The following recommendations are made for additional metallurgical testwork prior to the detailed engineering stage:

- Evaluate a range of different attrition mill media 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; and
- Complete a series of flotation tests on samples covering the mine life past the initial 5 years.

1.19.4 Recovery Methods

Optimization and refinement opportunities exist regarding the process design which could reveal benefits over the equipment selections and unit process detail

within the current design. The latter are based essentially on test work outcomes pursued and reported on thus far for study purposes.

Appropriate test work is recommended prior to the initiation or during the course of a detailed design phase preceding construction. This would include the following:

- Bulk material flow test work;
- Additional test work, in conjunction with vendors and in line with ongoing technical developments, aimed at further refinement of the polishing and attrition milling processes;
- Concentrate attritioning circuit static and dynamic thickening tests, including reagent scoping and optimization trials;
- Further investigation into potentially replacing the final tailings disposal positive displacement pumps with more common centrifugal pump trains by reducing the slurry solids concentration for overland pumping. This will include examination into whether the overall water balance and supply system can reasonably accommodate such a change.

1.19.5 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 (“RWD”). It is proposed that all the material excavated from the RWD 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

1.19.6 *Water*

The following is recommended during the detailed design phase:

- Updating the current dynamic water balance including a dynamic TSF water balance. The current water balance only assumed average monthly inflows from the TSF into the RWD. It would be recommended to confirm the water availability on the Molo Graphite Project if drought conditions occur and the TSF model element is included in the dynamic water balance
- 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 and scavenger wells 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 Molo Graphite Project site should be done as soon as possible.
- The installation and testing of the additional well field boreholes must be undertaken. The groundwater resource model must be updated to include site specific borehole data.
- The environmental geochemical test work of the Molo 2015 FS should be confirmed by selective testing of samples from the latest exploration and metallurgical test programs. The geochemical model should be updated accordingly.

1.19.7 *Environmental, Social*

- GCS recommends the installation of a suitable weather station at or as near as possible to the proposed project site, even before construction commences. 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 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 Green House Gas (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

1.19.8 *Permitting*

- An application for the exploration permit in Energizer's name is a critical step in the larger permitting and licensing regime and requires early attention and dedicated involvement
- Security of land tenure is a process and is estimated to take 7 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 etc should be undertaken within the ESIA review period (6 months), which is expected to be from March till August 2015
- Compilation of a comprehensive legal register
- Municipal elections are scheduled for July 2015. It is recommended that all above-mentioned permitting processes should commence prior to and in anticipation of these elections

The permitting and licensing of the proposed Molo Graphite 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.

1.19.9 *Tailings*

Additional work required during detailed design of the TSF and adjacent Return Water Dam (RWD) is as follows:

- The full rheology and beaching characteristics for the tailings are not known which leaves uncertainties regarding the optimum deposition design. This will need to be investigated via large scale tests once suitably sized pilot process plant samples are available. It should be noted that such large scale tests will also provide additional more representative samples which can be used to carry out further testing of other tailings characteristics, such as consolidation, permeability and shear strength, which should be used to validate / revise the assumptions made for the stability assessments, seepage / drainage assessments and water balance
- The geotechnical investigation was carried out for the general TSF area only, and was not focused on the specific design elements as the location of these

was not known at the time. Additional focused geotechnical investigations will be required to confirm the geotechnical conditions at specific locations

- The depth to groundwater is not known in the immediate vicinity of the RWD. In the event that ground water is shallow, it may not be possible to excavate the RWD basin to the required depth without employing dewatering measures, or alternatively constructing an additional RWD downstream. The depth to groundwater and any seasonal fluctuations will need to be investigated by installation of a groundwater monitoring borehole, which must be monitored during the wet season
- Water quality data is required over a period of time to provide a baseline for comparison once the TSF is commissioned. To provide the necessary baseline data, regular ground and surface water monitoring must be carried out leading up to the date when the TSF is commissioned
- The overall design will need to be developed to a level required for construction and to optimise the design with regard to technical, environmental and economic considerations, whilst taking due cognisance of additional information made available, including the additional studies detailed

2 INTRODUCTION

2.1 The Issuer

This Technical Report (the "Report") has been prepared for Energizer Resources Inc. (http://www.energizerresources.com/) ("Energizer", or "the Company"), a mineral exploration and development company based in Toronto, Canada. Energizer is currently focused on the exploration and development of its Molo graphite deposit ("Molo", the "Molo Graphite Project" or "the Project") in south-western Madagascar.

2.2 Technical Team

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

- Caracle Creek International Consulting (Pty) Limited – Geology
- GCS Water and Environment (Pty) Ltd / Agetipa – Water, Environmental, Social, Permitting
- SGS Lakefield – Metallurgical Test work
- Metpro Management Inc. – Metallurgical Test work analysis
- SRK – Tailings Storage Facility, Civil Geotechnical work
- DRA Projects (Pty) Ltd – Mining, Process, Infrastructure, Capex, Opex
- SDV, Panalpina, Unitrans, Strang – Logistics
- OHMS – Mine Geotech
- RLH – Port Trade Off Study
- Cresco Project Financing – Financial Modelling

2.3 Report History

The report history for this Project is as follows:

- Maiden resource statement – December 2012
- Molo Graphite Project, Fotadrevo, Province Of Toliara, Madagascar, Preliminary Economic Assessment, Technical Report Update, (NI 43-101), 12 April 2013, DRA Mineral Projects (Pty) Ltd
- Molo Graphite Project, Fotadrevo, Province of Toliara, Madagascar Technical Report NI 43-101 – 25 September 2014 (Hancox and Subramani)

2.4 Terms of Reference and Purpose

The terms of reference for this phase were to build on the earlier resources statement and PEA and complete a full Feasibility Study on the Molo Graphite Project deposit. The work covered the following key activities:

- Drilling of a further 2 063 metres (32 diamond cored boreholes) and trenching an additional 1 876 metres (9 trenches) to increase the total Molo resource and upgrade as much as possible into the Measured and Indicated categories in order to provide a basis for conversion to Proven and Probable Mineral Reserves. The entire database on which this new model and resource estimate is based now contains 80 drill holes (totalling 11,660 metres) and 35 trenches (totalling 8,492 metres).
- To generate a mine design and pit schedule and declare maiden Proven and Probable Mineral Reserves including understanding the geotechnical conditions and geohydrology of the pit
- To do further process test work (optimization and variability) and finalize a flow sheet to be used for process design and costing
- To engineer the process plant
- To engineer all supporting infrastructure including water, power, buildings, accommodation, tailings dam etc.
- To complete a full ESIA for submission to the Malagasy government as part of the permitting process
- To understand the permitting regime in Madagascar
- To understand inbound and outbound logistics and logistics costs
- To compile Capex and Opex estimates for the project
- To understand the graphite market
- 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 the Molo 2015 FS. See

Table 12 below:

Table 12: Sources of Information

Report of Document	Author / Organization
Physical Data	
Site Topographical Data	Energizer
Exploration Permit #3432	Malagasy Government
Civil Geotechnical Reports	SRK 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	DRA
Contract Mining Tenders	Various
Equipment Tenders	Various
Process Plant and Metallurgy	
Process Design Criteria (PDC)	DRA
Process Flow Diagrams (PFD)	DRA
Mechanical Equipment List (MEL)	DRA
Process Control Diagrams (PCD)	DRA
Metallurgical Test Work Reports	SGS
Metallurgical Test Work Analysis	Metpro
Vendor Test Work Reports	Ouotec, FLSmidth, Heyl and Patterson, Derrick, BASF, Senmin, Great Western
Hazop Studies (Hazop 1 and Hazop 2 Reports)	DRA

Report of Document	Author / Organization
Plant Layout Drawings	DRA
Block Plan	DRA
Rheology Test Work	SGS
Production Schedule	DRA
Tailings Disposal and Storage	
Site Selection Report	SRK Consulting
TSF Report (including TSF Geotechnical Study)	SRK Consulting
Infrastructure	
Site Plan	DRA
Area Plan	DRA
Architectural Drawings	DRA
Infrastructure Drawings	DRA
Infrastructure Equipment List (IEL)	DRA
Plant and Camp Geotechnical Report	SRK Consulting
Water Supply Dam Geotechnical Report	InRoads Consulting
Surface Water Supply Report - ANT_2 Dam	GCS
Power	
Power Plant Layout Drawings	DRA
Running Load Estimate	DRA
Single Line Diagram	DRA
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	Energizer
Malagasy Labour Rates	Energizer
Staffing Plan	DRA
Transport and Logistics	
Route Surveys	Unitrans, SDV, Colas

2.6 Personal inspections

The following personal inspections were undertaken on the property:

1. Dr. Philip John Hancox visited the site between the 8th and 11th of May 2012. During this visit the graphite occurrences at Molo 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.
2. 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.
3. Dave Thompson visited the Molo site on 11-13th March 2014. The aim of this visit was to assess the site from a mining perspective.
4. Doug Heher visited the Molo site between the 11-13th March 2014 and again on 19th August 2014. The purpose of the visit in March was to get familiar with all aspects of the site including the proposed plant location and general infrastructure siting. The visit in August was to take the South African Department of Trade and Industry to site as they subsidised a portion of the study.
5. Oliver Peters and John Stanbury have not visited the site as this was not required for their role on the Project.
6. 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 Report has been prepared for Energizer 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 6th of February, 2015, other than in concerning the mineral resource and mineral reserve estimates which are effective 14th of August 2014 and January 2015, respectively.)
- Assumptions, conditions, and qualifications as set forth in this Report, and
- Data, reports, and other information as supplied by Energizer and other third party sources

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

Project information in this Report is sourced from previous works supplied by Energizer 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 Energizer. To the extent possible under the mandate of NI 43-101, the data have been verified with regard to the material facts.

DRA have received specialist input from the following organizations in the preparation of this Report and place full reliance on this information:

- SRK (Tailings Storage Facility)
- SRK (Site Civil Geotechnical Information)
- 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)

Except for the purposes legislated under Canadian 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.

4.1.1 Country Overview

Madagascar is the largest island in the Indian Ocean (Figure 7) 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 km^2 and a 5,000 km long coastline.

Figure 9: 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 ("m amsl").

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 Molo Graphite Project is located.

4.1.2 Government Policy and Outlook Regarding the Mining Industry

The government of Madagascar embarked on an economic revival plan in 2000. At that time the Ministry of Energy and Mines ("Ministry") had already initiated reform through the Projet de 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.

4.2 Location

The Molo Graphite Project is located approximately 160 km southeast of Madagascar's port city of Toliara, in the Tulear region and about 220 km northwest of Fort Dauphin. See Figure 10 below.

Figure 10: Map of Southern Madagascar

4.3 Property area

The Molo Graphite Project is contained in a portion of Research Permit #3432. See Figure 11 below. The Project is centered on UTM coordinates 495,289 Easting 7,345,473 Northing (UTM 38S, WGS 84 datum). The Molo Graphite Project is located 11.5 km east-northeast of the town of Fotadrevo and covers an area of 62.5 hectares ("ha") located inside Exploration Permit #3432. The Government of Madagascar designates individual claims by a central LaBorde UTM location point, comprising a square with an area of 6.25 km^2.

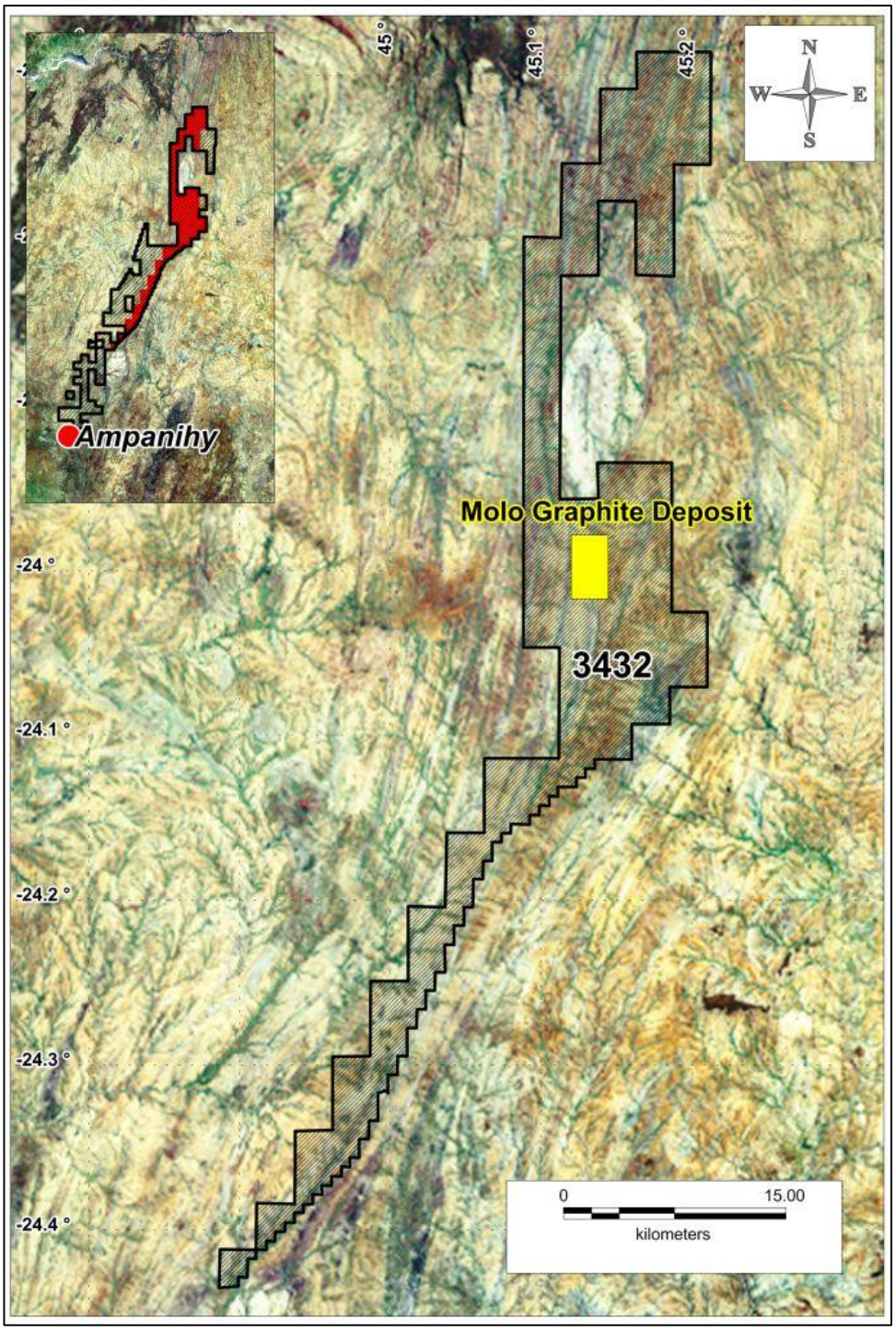


Figure 11: Molo Project - Exploration Permit #3432

Source: Energizer Resources

4.4 Mineral Tenure and Title

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

On April 16, 2014, 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").

The Molo Graphite Project is located within Exploration Permit #3432 as issued by the Bureau de Cadastre Minier de Madagascar ("BCMM") pursuant to the Mining Code 1999 (as amended) and its implementing decrees.

CCIC has 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 Energizer or one of it's Madagascar subsidiaries**.**

Energizer holds the exclusive right to explore for a defined group of industrial minerals within the permits listed above. 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, and Zeolites.

Companies in Madagascar first apply for an exploration mining permit with the Bureau de Cadastre Minier de Madagascar ("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. Research 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 on the 31st 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 a Research 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. Upon establishment of a mineral resource, Research Permits may be converted into Exploitation Permits by application. CCIC is of the opinion that Energizer is compliant in terms of its commitments under these reporting requirements.

The Molo Graphite 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 Energizer positional data is collected in WGS 84, and if necessary converted back to LaBorde.

4.5 Royalties

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

4.6 Permits

Exploration Permit #3432 (425 km^2) is currently held under the name of a subsidiary of Malagasy Minerals called, Mada-Aust Sarl. The transformation or amendment of exploration and research mining permits within the country continues to be suspended from the time that Madagascar was run by a president who was not democratically elected. This current permit expired on August 17, 2011. Energizer's Madagascar domiciled subsidiary companies and Mada-Aust Sarl have continued to pay all taxes and administrative fees to the Madagascar government and its mining ministry with respect to all the mining permits held in country. These taxes and administrative fee payments have been acknowledged and accepted by the Madagascar government. In addition, Energizer continues to diligently work with the Madagascar government to obtain the necessary permits in its name as the country clears its backlog of applications and amendments.

The research permit will be converted into an exploitation permit in due course. When the permit is transformed from a research permit to an exploitation permit, the exploitation permit will be issued in the name of Energizer. The exploitation permit is required to advance the Molo Project to the developmental stage.

The suspension of transformation or amendment of exploration and research permits within Madagascar is the only factor that the authour is aware of that may affect access, title, or the right or ability to perform work on the Project.

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 12). From Andranovory, secondary all-season roads continue to Betioky, a distance of 93 km. From Betioky the Project 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 track.

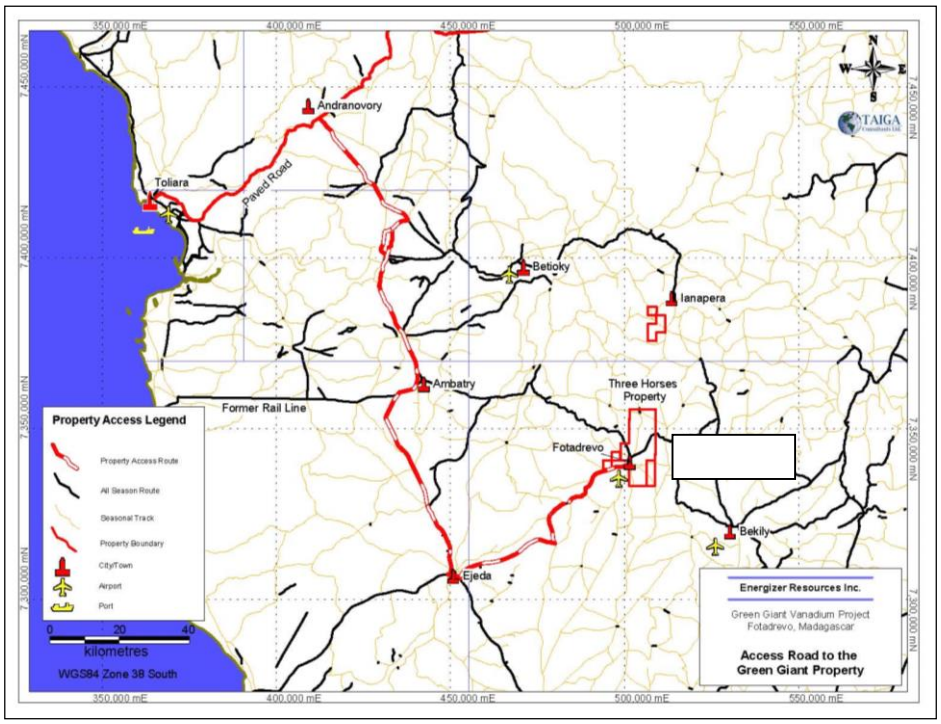


Figure 12: Road Access to the Molo Area from the Town of Toliara

With the upgrading of an existing airstrip at Fotadrevo (Figure 13) to an all-weather airstrip during the 2008 exploration programme, the Molo 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 13: 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 14) 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 (“m amsl”).

Figure 14: View of the Molo Project Area showing the Nature of the Vegetation

Typical of the tropics, the surface is subject to lateritic weathering. Full laterite profiles are, however, rarely developed within the southern climatic zone. Previous drilling on the Project 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 southeastern 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 Energizer has its base camp) is located 12 km 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 Energizer 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.

The land in the vicinity of the Molo Graphite Deposit is open and unpopulated with sparse shrub and grassland cover. As such, there is more than sufficient area for mining operations, tailings storage, waste disposal area, and processing plant. The land is non-titled, so negotiations for infrastructure surface rights will be conducted with local communities, and regional and national authorities.

5.5 Security

As Madagascar is an island it has no border issues. Furthermore there are no conflicts 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 Energizer, to ensure security on the Project, on the work site and for the Company's equipment.

6 HISTORY

The region around the Project 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 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 the BRGM, and therefore these volumes are key to retracing any historical data. Archival research by Energizer has however not revealed any evidence of mineral exploration in the past fifty years within the Molo Project area.

6.1 Property-Scale Exploration History

Prior to the exploration work completed by Energizer there is no record of any previous exploration activity within the Project area and no historical resource estimates exist for Molo. Table 13 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 13: Historical Activities on the Project

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

Exploration work undertaken in 2011 led to the discovery of the Molo deposit, which then became the focus of the 2012-2014 exploration programmes addressed in this 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 15 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 Supergroup successions of sub-Saharan Africa, which was widespread in the former supercontinent of Gondwana.

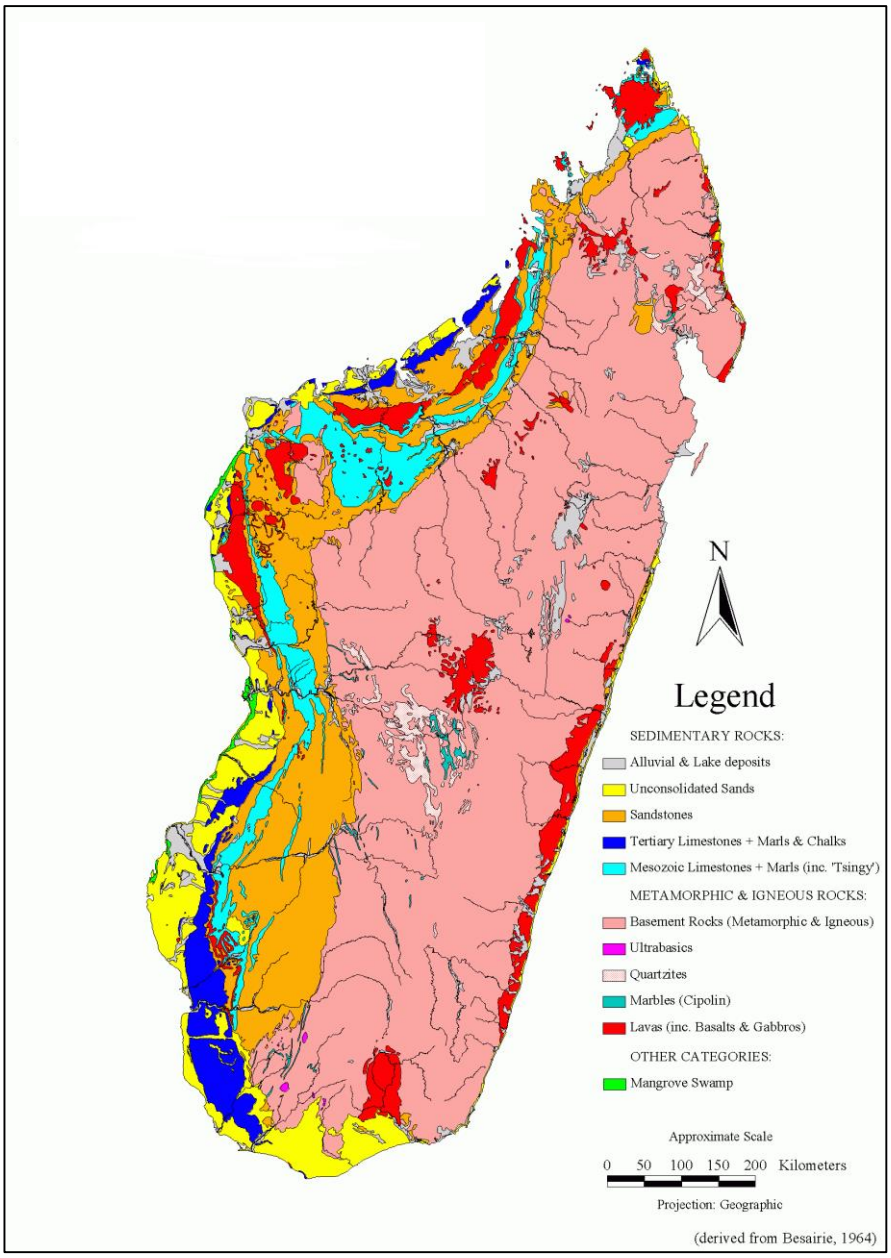


Figure 15: Geological Map of Madagascar showing the Distinctive Crystalline Basement and Sedimentary Basins in the West

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 16). 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 17). 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.

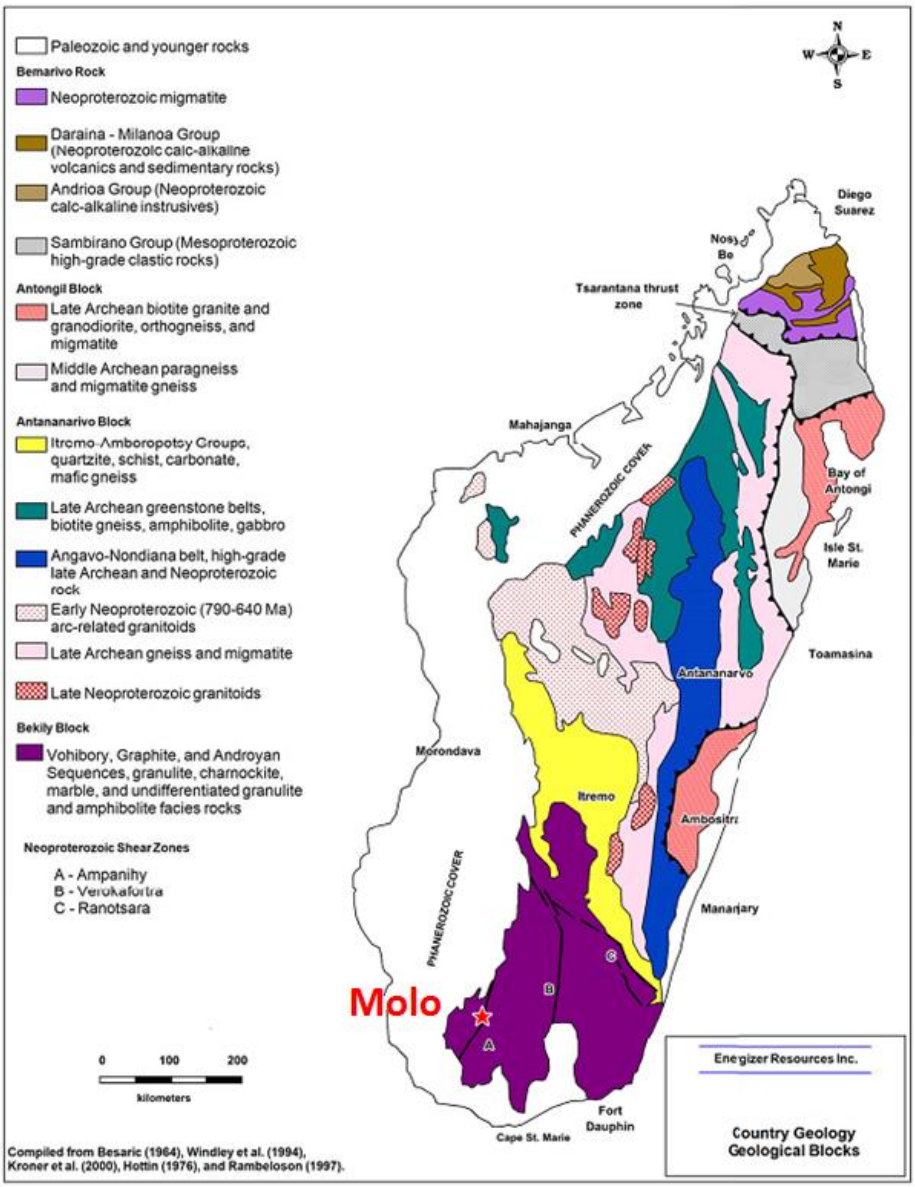


Figure 16: Country Geology: Geological Blocks

Source: AGP Mining Consultants (2011).

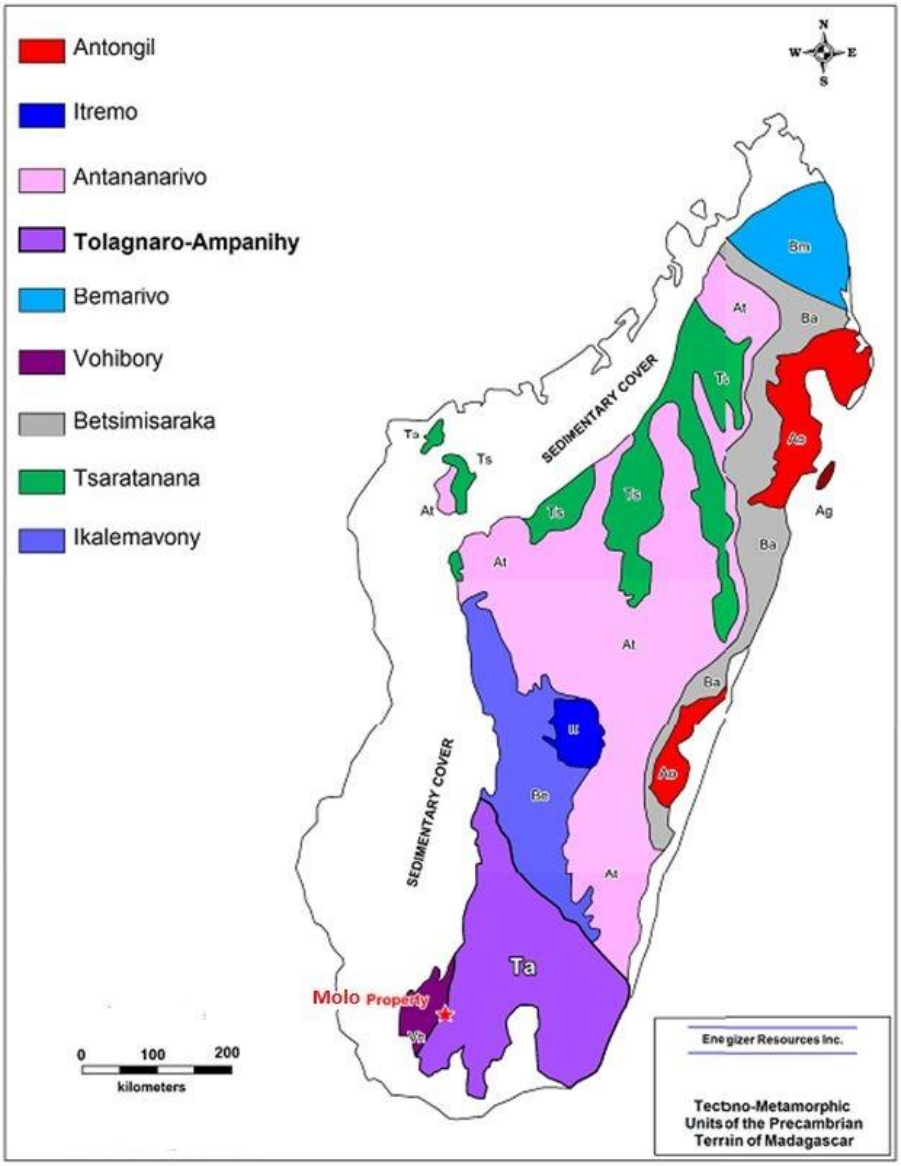


Figure 17: Tectono-metamorphic Units of the Precambrian Terrain of Madagascar

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 18). 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 from 3 to 11 kilobars ("K bar") 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 (D_1 and D_1), during which northeast verging recumbent sheath folds and ductile thrusts were formed, are dated between 647 to 627 Ma. Early prolate mineral fabrics (L_1/L_2) are preserved in massif-type anorthosite bodies and their marginal country rocks.

D_1 and D_2 deformation was followed by a 10 to15 Ma period of static, annealing metamorphism when bulk shortening (D_3) took place. D_2 and D_3 deformations are coaxial, but are separated in time by leucocratic dykes that intruded between 620-610 Ma. Between 609 and 607 Ma, D_3 deformation was focused zonally, forming the prominent north–south shear zones. Oblate strain resulted in a strong composite D_2/D_3 fabric defined by sub-vertical S-tectonites and sub-horizontal intersection lineations.

A variety of post-D_3 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 D_4 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.

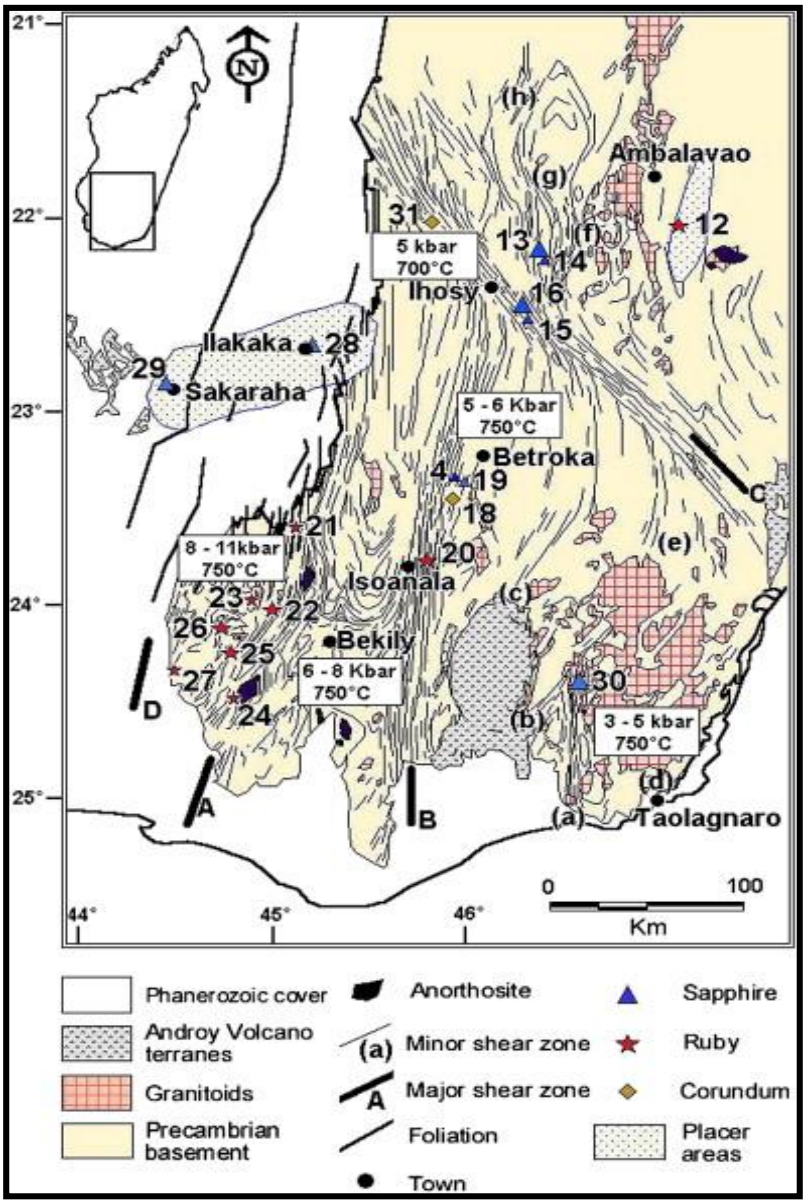


Figure 18: Structural and Lithological Sketch Map of South-east Madagascar showing the positions of the Major Shear Zones

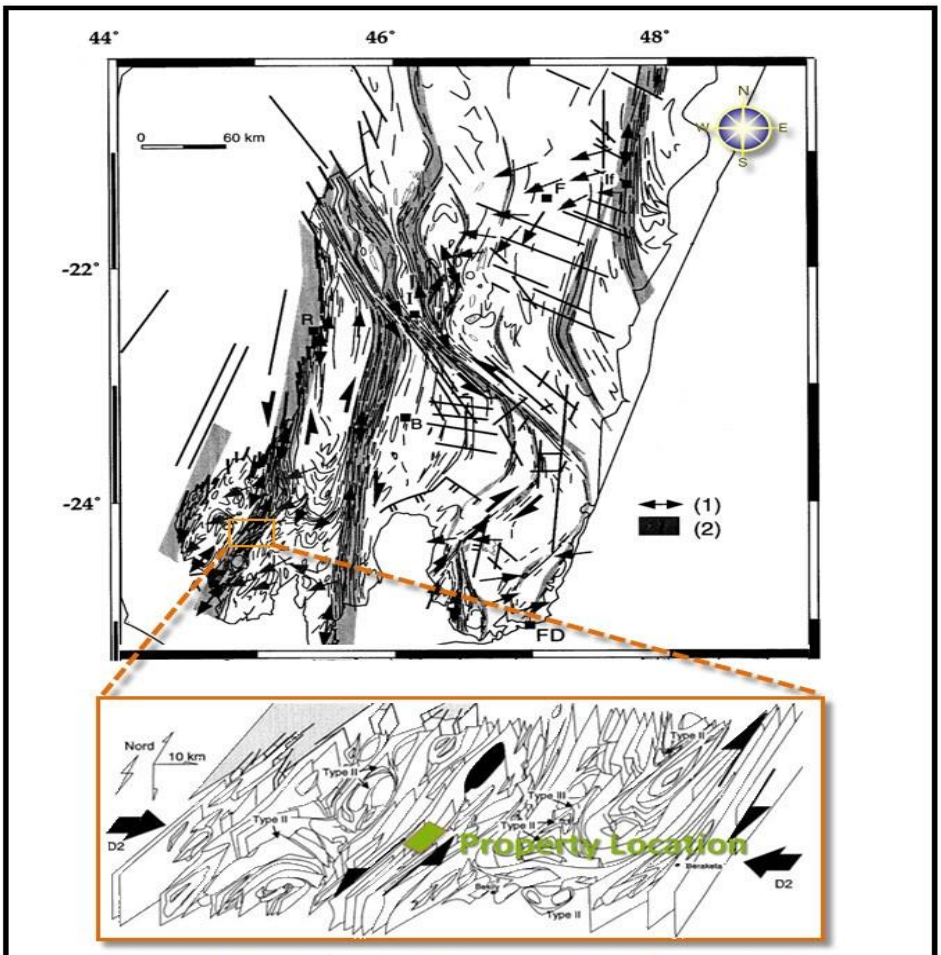
From: Rakotondrazafy et al. (2008).

7.2 Regional Geology as it Relates to the Molo Deposit

The Molo deposit occurs within the regional Ampanihy Shear Zone (Figure 19 and Figure 20). 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 19 below illustrates the general position of Molo relative to the D2 regional strain pattern, and the resulting Ampanihy Shear zone.

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



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

Modified from Martela, 2000

Figure 19: Position of the Molo Project Area within the Overall Strain Pattern Documented for Southern Madagascar

Descriptions of the individual lithological units identified by Energizer, 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 (≤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-feldspatic gneiss comprises a mixture of fundamental constituent lithologies, dependent on the relative abundance, or absence of biotite and garnet.

7.2.1.7 Feldspathic Gneiss

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

7.2.2 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 Project 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:

1. Zones where magnetic units are parallel, or near parallel to the walls of the domain. In these regions, the shearing has reduced intrafolial folds into sheared out 'tectonic fish'. There are also broad zones where a low content of magnetite (± pyrrhotite) may be present which most likely represent a different metamorphic mineral assemblage, (different pressure and temperature conditions), or pre-metamorphic alteration and/or rock types.
2. 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.

3. 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 20). Outcrop mapping and trenching on Molo has shown the surface geology to be dominated by resistant ridges of graphitic schist (Figure 21) and graphitic gneiss, with fracture-lined vanadium mineralisation, as well as abundant graphitic schist float.

Geological modelling (Figure 22) has shown that the deposit consists of various zones of mineralised graphitic gneiss, with a barren footwall composed of garnetiferous gneiss (Figure 23). The host rock of the mineralised zones is graphitic gneiss.

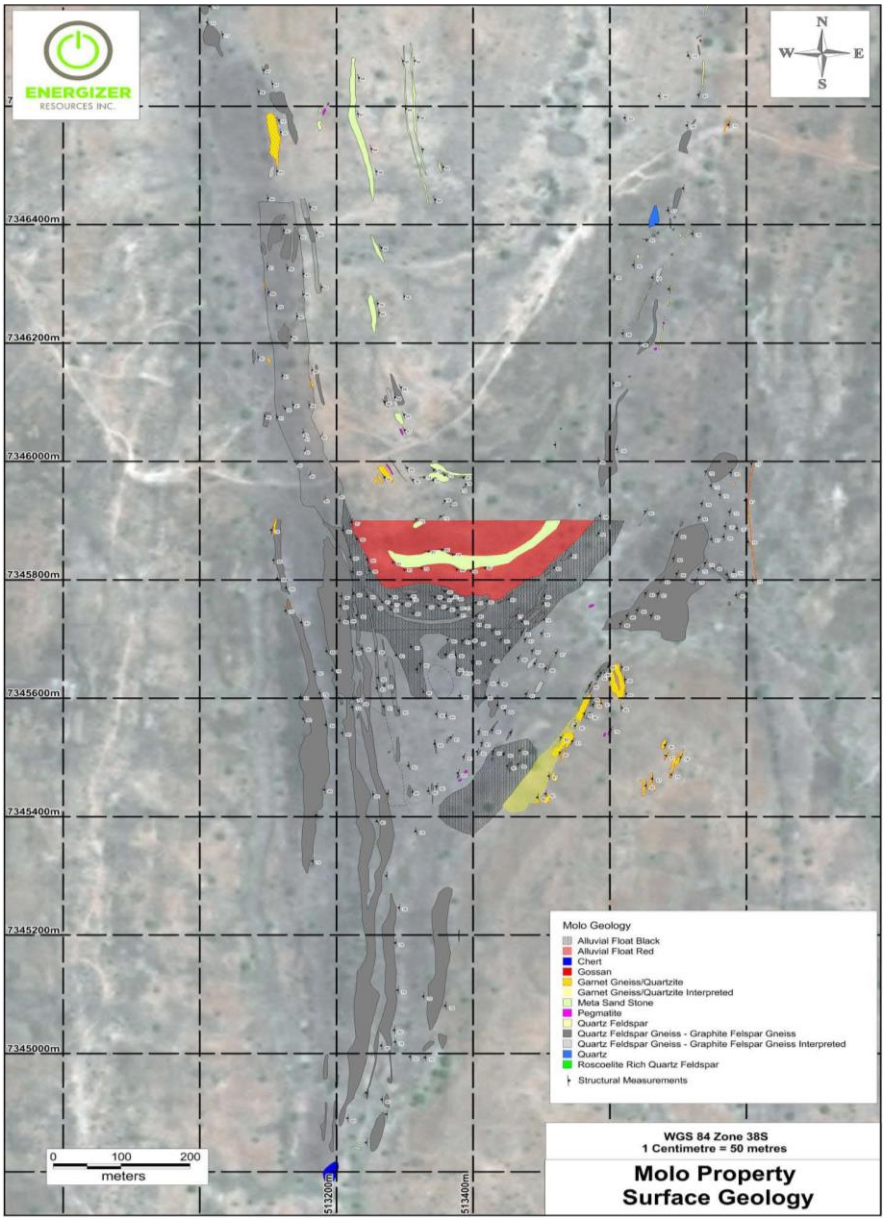


Figure 20: Map showing the Surface Geology of the Molo Deposit

Figure 21: Outcrop Exposure of Graphitic Schist on the Molo Project

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

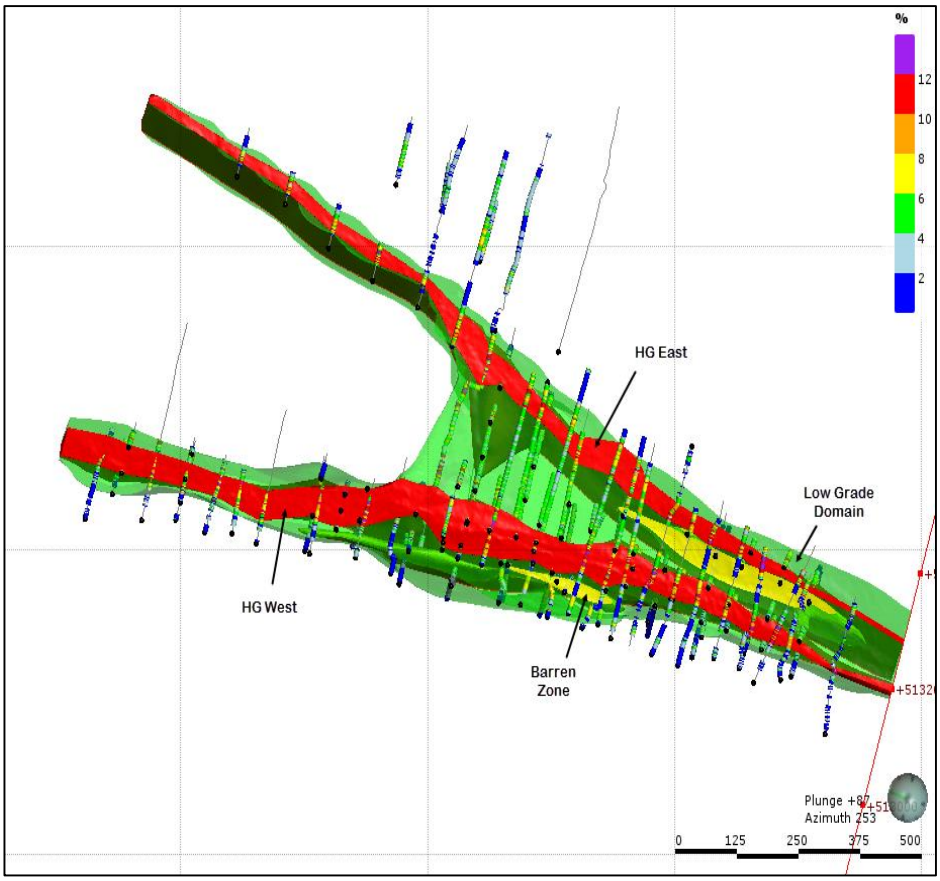


Figure 22: Geological Model of the Molo Deposit showing the Nature of the Two Tightly Oppressed 'High'' Grade (HG East and HG West) Mineralized Zones

Figure 23: 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 scientifically 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 Item 14 “Mineral Resource Estimates”.

8 DEPOSIT TYPES

The Molo Project hosts Flake graphite deposits.

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:

1. Flake graphite disseminated in metamorphosed, silica-rich
2. Sedimentary rocks
3. Flake graphite disseminated in marble
4. Amorphous deposits formed by metamorphism of coal or
5. Carbon-rich sediments
6. Veins filling fractures, fissures, and cavities in country rock
7. 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 a marble gneiss, and schist. Each flake is separate, having crystallized as such in the rock. In many cases, pegmatitic veins have intruded the rocks.

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.

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%.

9 EXPLORATION

A reconnaissance exploration programme was undertaken on the Project 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, Energizer 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.

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 Project 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.

During the 2011 field season, a 1:1000 scale geologic map was completed over the Molo.

9.2 Trenching

During the 2011 field season trenching was completed on Molo. Initial graphitic carbon results from the 2011 trenching were encouraging in that they showed multiple graphic horizons present in each zone, of significant widths and grades. Because of this, and coupled to the size of the electro-magnetic signature, the 2012 programme focussed on Molo and an additional 22 trenches were excavated. Additional trenching was undertaken on Molo during May of 2013 as part of a bulk sampling exercise (Figure 24). Subsequently an additional nine trenches (totalling 1,876 metres) were excavated as part of the 2014 exploration programme.

Figure 24: 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 25.

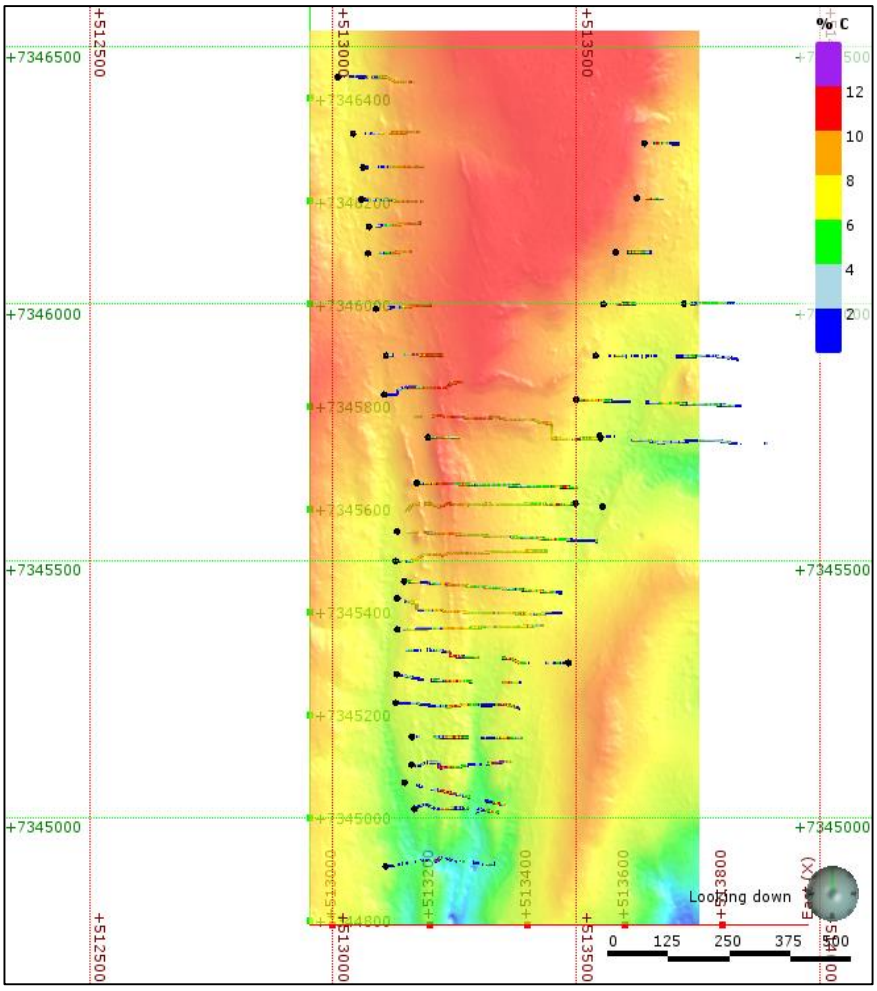


Figure 25: Plan showing the Positions and Grades of all the Trenches Excavated on the Molo Deposit, Overlain on the Topographic Elevation Map

9.2.1 Trench Sampling

Standardized sampling methods include two metre 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 14 below. This data was used in the estimation process.

Table 14: 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

Trench	From (m)	To (m)	Length (m)	C%
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. Energizer'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. This data was used to plan the original (2011) boreholes drilled on Molo (Figure 26).

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 26) 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.

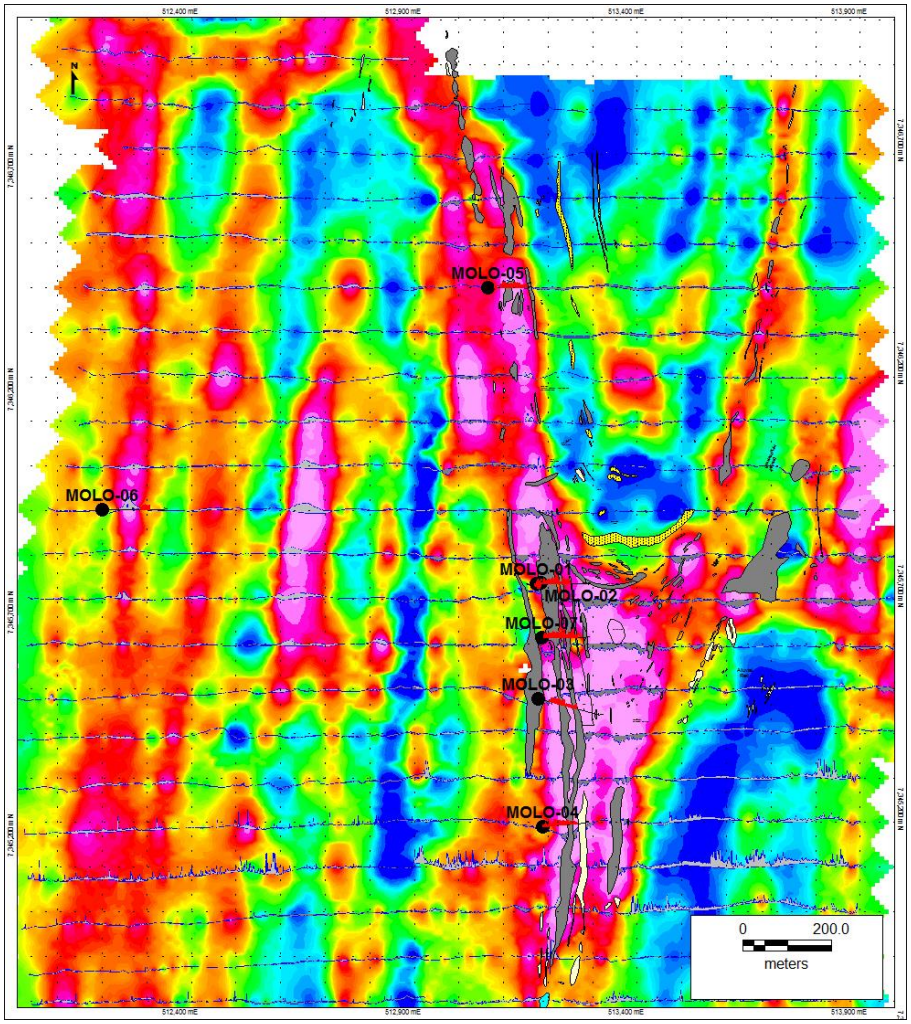


Figure 26: EM31 Geophysical Survey Map of the Molo deposit showing the positions of holes MOLO-01 to MOLO-07 drilled during 2011

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

Project 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 Project to be ‘ground truthed’ and, where applicable, sampled in an intensive manner to gain an understanding of all of the zones.

During 2011 Energizer's employees thoroughly covered the entire area of Permit #3432, with special interest in graphitic showings.. Over the course of six days Energizer'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 know historical drilling is documented on the 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 Energizer entered into a JV agreement with Malagasy and this phase therefore was re-focussed on the new graphitic prospects on the JV areas.

Seven (Molo-01 to Molo-07) wide spaced holes were drilled on Molo during the 2011 diamond drilling campaign. 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 27), (totalling 11,660 metres) have now been completed on Molo, and these were used for the mineral resource estimations.

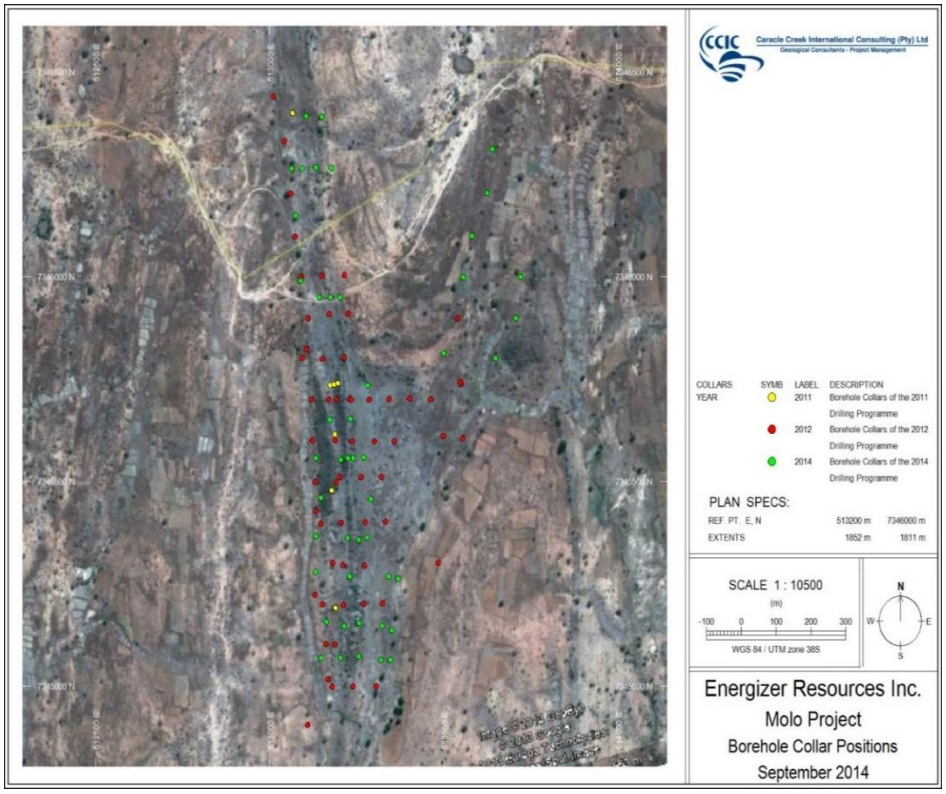


Figure 27: Borehole Collar Positions for all of the Known Boreholes Drilled on the Molo Deposit

10.1 Diamond Drill Contractor and Logistics

Between 2009 and 2014 all diamond drilling on the 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 Energizer. 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. Energizer'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 Energizer's Fotadrevo camp within a secure, 20m x 25m fenced enclosure.

10.1.2 Core Logging

Energizer 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. Energizer'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 28) and then forwarded for cutting. Core is typically photographed wet.

Figure 28: Core Photographs of Borehole MOLO-05 Core (Boxes 17 and 18) as Supplied to CCIC by Energizer

10.1.5 Collar Survey

Borehole collar locations are initially established in the field using a hand held (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 29).

Figure 29: 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 Energizer 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 follows:

- 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 Energizer's Fotadrevo campsite until shipment

10.1.9 Diamond Drill Results

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

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

BHID	Easting	Northing	Elev	Length	Dip	Azimuth
MOLO-01	513173	7345735	553.96	166	-45	90
MOLO-02	513184	7345737	554.52	47	-45	265
MOLO-03	513177	7345478	549.88	207.5	-45	105
MOLO-04	513188	7345191	545.79	137	-45	85
MOLO-05	513065	7346400	551.60	131	-45	90
MOLO-06	512199	7345902	543.51	200	-45	90
MOLO-07	513186	7345615	550.84	142.5	-45	90
MOLO-12-01	513121	7345600	550.78	463.6	-45	90
MOLO-12-02	513185	7345601	550.58	89	-45	270

BHID	Easting	Northing	Elev	Length	Dip	Azimuth
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 Energizer, or their agents.

When sufficient sample material (grab, trench, or core) has been collected, the samples are trucked, or flown to Energizer's storage location in Antananarivo, at all times accompanied by an Energizer employee. From there, samples are further shipped to either South Africa (Mintek, or Genalysis), or Canada (Activation Labs) for ICP-MS analysis. Activation Labs, Mintek and Genalysis are all ISO 17025 accredited and/or certified to 9001: 2008.

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 Project Manager, as well as Energizer executive staff, and were posted on a secure website and downloaded by Energizer 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 Energizer.

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

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

Energizer 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 see Figure 30. Blanks for the 2014 campaign shows that 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.

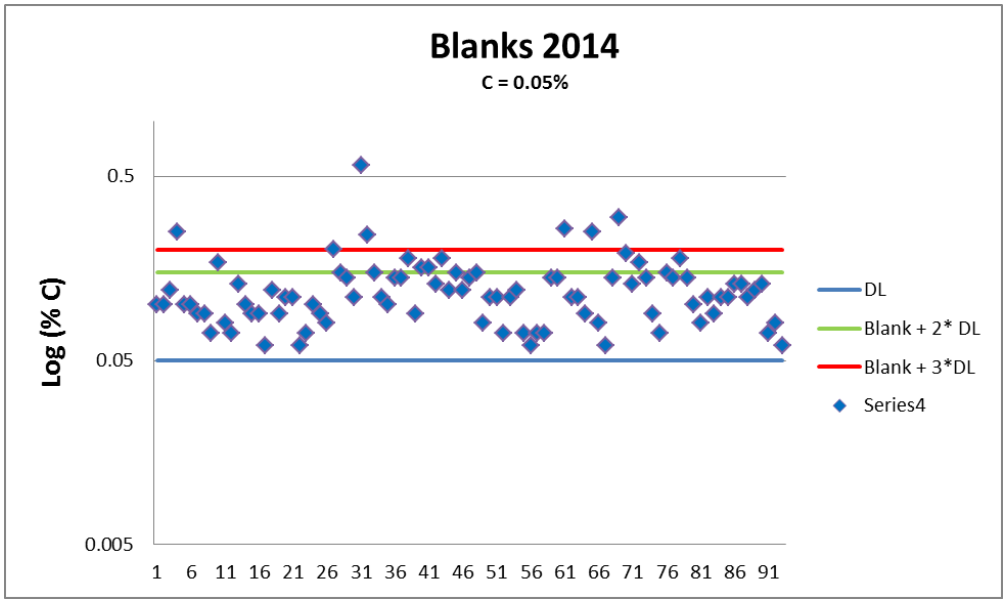


Figure 30: Plot of Log %C versus the Date of the Analysis 2014 for the Blanks

11.2.2 Standards

Because of the difficulty in sourcing certified standard reference material ("CRM") during the 2012 campaign, Energizer 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, Energizer 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 31, Figure 32 and Figure 33).

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 inside 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.

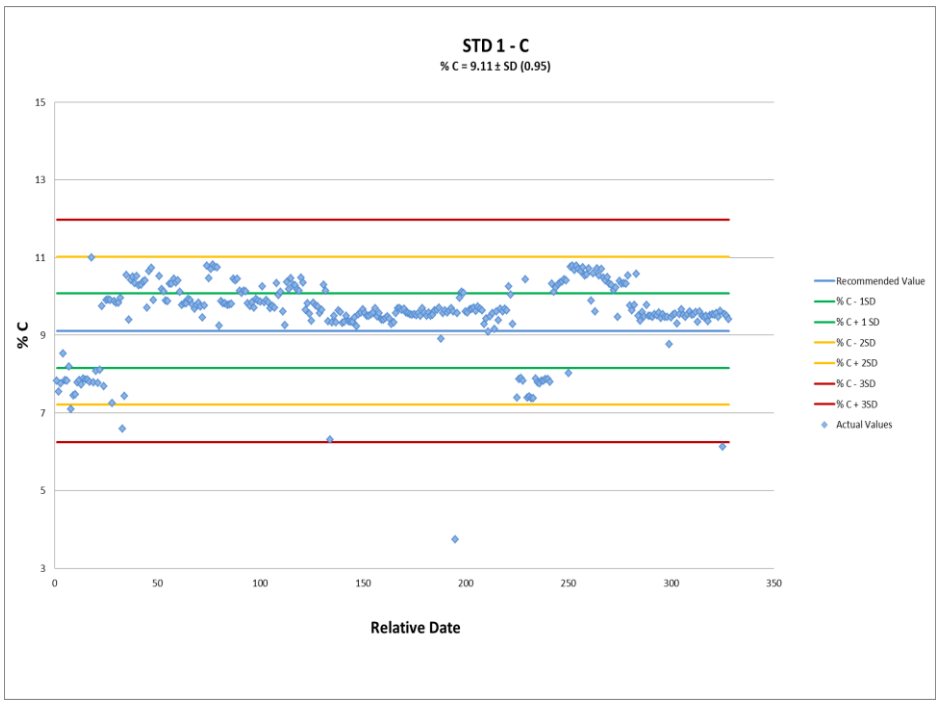


Figure 31: Graph showing Carbon Concentration as Analysed in STD 1C

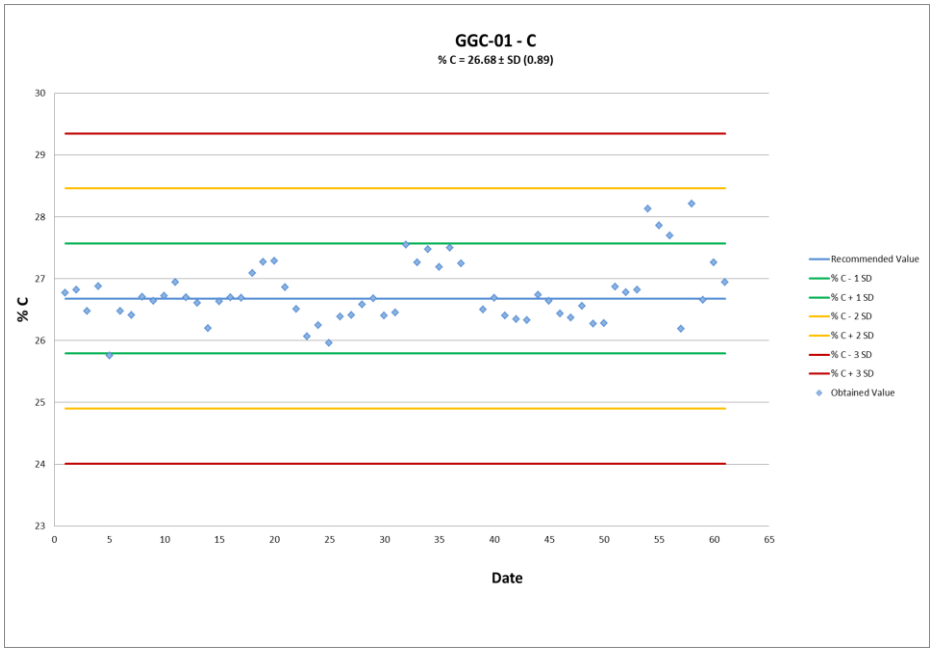


Figure 32: Graph showing Carbon Concentration as Analysed in GGC-01

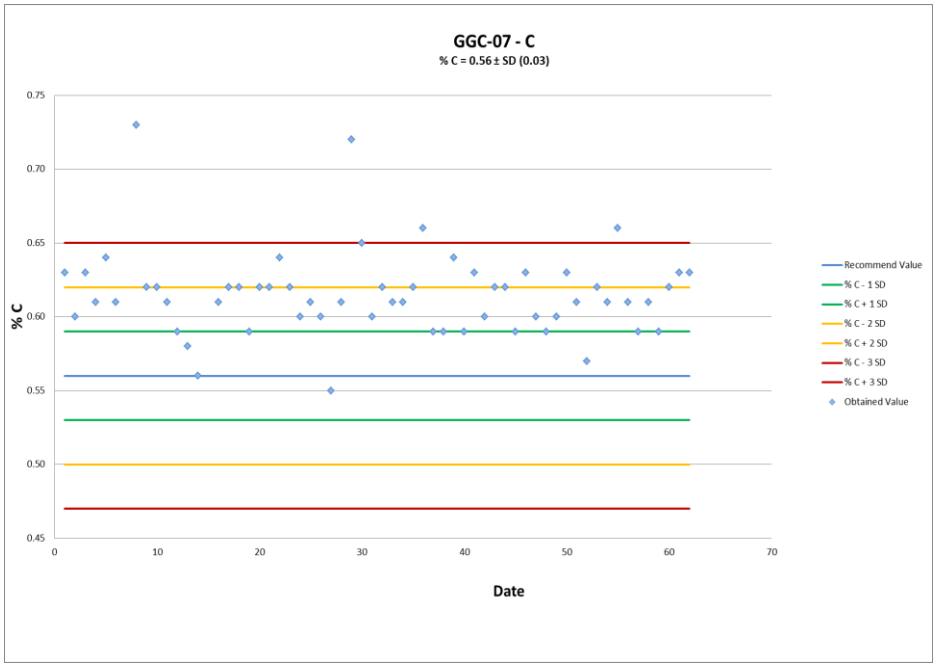


Figure 33: Graph showing Carbon Concentration as Analysed in GGC-07

11.2.3 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 34) 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 R^2 value of 0.96.

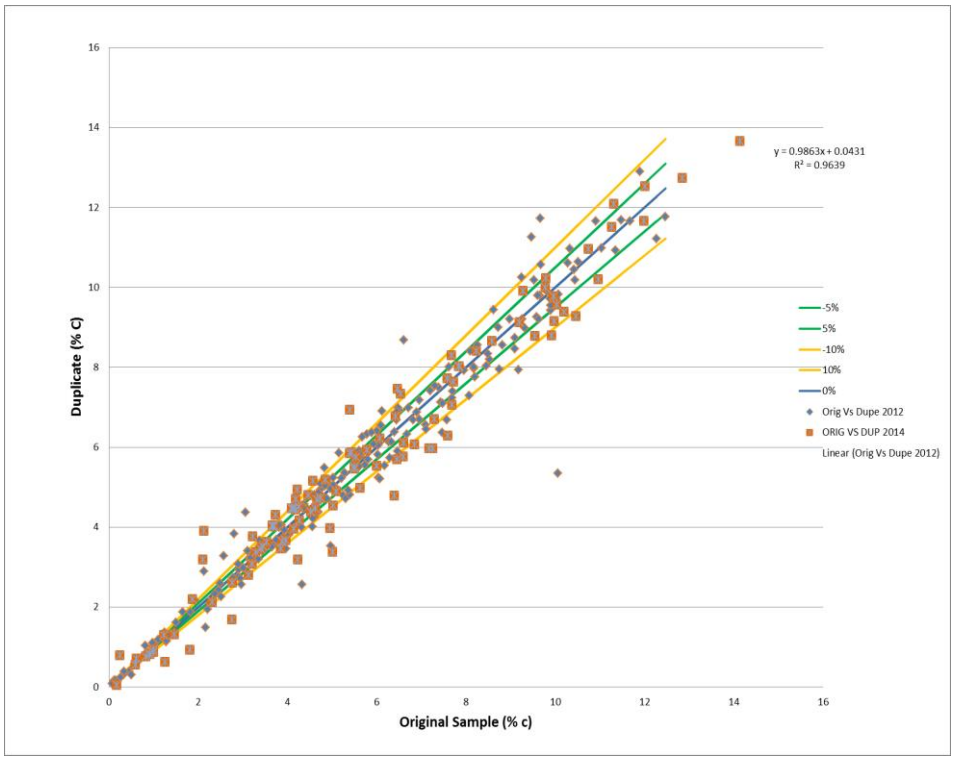


Figure 34: Original (Orig) versus Duplicate (Dupe) Plots

11.3 Relative Density

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

The process to measure the RD is as follows:

- 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 Energizer and the Molo Project, all information published regarding the 2011 exploration programme was reviewed by an independent Qualified Person as it became available.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, Energizer 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 re-surveyed surface topography of the area (Figure 35). 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.

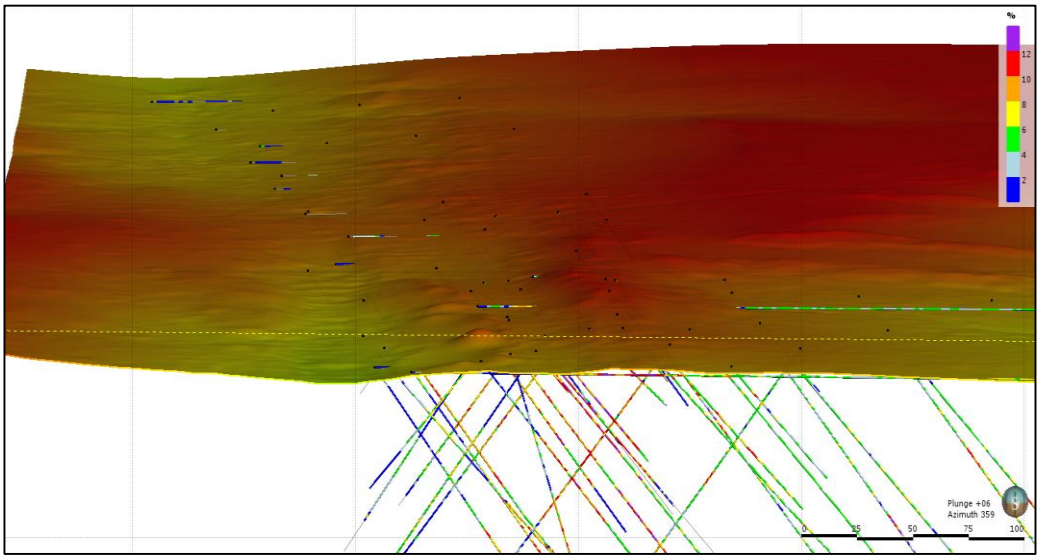


Figure 35: East-West Cross-section Showing the Collar Positions with Respect to Topography

12.2 Drill Logs

During the initial 2012 site visit Dr. Hancox randomly selected the 2011 drill hole Molo-07 to review the log's data against the drill core. The hole was 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.

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 Energizer 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 Energizer 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. The primary purpose of the programs from the perspective of the Molo material was to:

(a) Develop a robust process flowsheet that produces a combined graphite concentrate grading of at least 95% total carbon.
(b) Demonstrate the robustness of the proposed flowsheet for the expected mill feed during the first few years of operation, and
(c) 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.

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

While there is a market for graphite concentrates grading as low as 80% total carbon, the price of a product increases with carbon grade. For a graphite flotation 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 has to 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 take into account 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 a Molo composite at Mintek in South Africa in 2012. This composite originated from 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 P_{50} 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 produced concentrate grades of 77.4% carbon with the dipersant 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 36. The second regrind was carried out 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 as a result of this secondary regrinding and cleaning circuit.

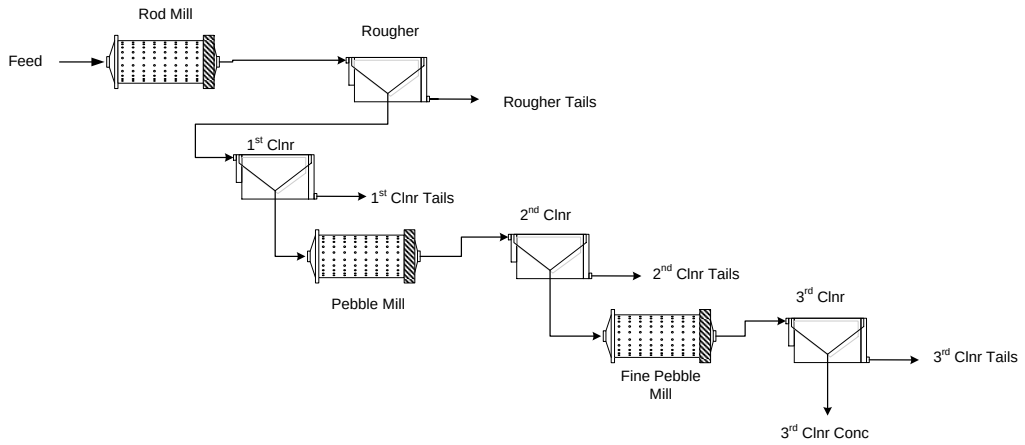


Figure 36: Flowsheet for Molo Mineralization – Mintek

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 Total Carbon and Graphitic Carbon Assay Methods – SGS

The following chemical analysis methods were employed for the SGS laboratory and pilot plant programs to determine total carbon and graphitic carbon content.

All assays were completed using control quality analysis, which falls between exploration quality and umpire/party quality with regards to QA/QC procedures.

Total carbon analysis was conducted by high temperature combustion followed by infrared detection. Samples with concentrations up to 30% total carbon were analysed with a LECO SC844 instrument, while samples containing higher concentrations of carbon were analysed using a LECO SC632. This instrument was specifically designed for high-grade samples and provides a better accuracy for graphite concentrates.

The estimated measurement uncertainty for the LECO SC844 instrument is 5.0 % relative or less for samples grading 1% total carbon or higher. The estimated measurement uncertainty for the LECO SC632 is 1.7% relative for concentrate grades of 90% to <95% total carbon and 1.4% relative for concentrate grades of 95% to 100% total carbon.

Graphitic carbon analysis was carried out with a three-step method. First the sample was roasted in an oven at 550°C for one hour to remove any organic carbon. The sample was then mixed with nitric acid and de-ionized water, digested and filtered to remove any carbonates. In the final step the residue was subjected to a total carbon analysis using a LECO infrared analyser.

While the estimated measurement uncertainty for the graphitic carbon method is affected by the sample matrix, it is significantly higher compared to the LECO SC632 instrument. Hence, graphitic carbon analysis is not conducted on graphite concentrates. Instead the total carbon analysis is carried out using the LECO SC632 analyser.

The total carbon analysis of the final concentrate was validated in approximately 20% of the flotation tests by proximate analysis, which is a three step process. The sample is first subjected at 105°C for at least one hour to determine the moisture content. The temperature is then increased to 500°C to determine the amount of volatiles. The temperature is maintained until a stable weight is obtained. In the final step the oven temperature is increased to 1,000°C and the graphite is combusted. The graphitic carbon concentration is determined by subtracting the percentage of moisture, volatiles and ash from 100%.

13.4 Flowsheet Development - SGS

A flowsheet development program was 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 was 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 P_{80} = 400 to 500 microns.

Four proceeding cleaner tests investigated the conditions of the primary cleaning circuit.

The flash and rougher concentrates were combined and subjected to a polishing grind using ceramic rods.

The polishing times were varied between 7 minutes and 30 minutes.

The polishing mill discharge was upgraded in three stages of cleaner flotation and the 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.

Taking into account the liberation properties of the different flake sizes, the intermediate concentrate was classified at 80 mesh (177 microns) and 150 mesh (106 microns) and 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 37 and the size fraction analysis results of the combined concentrate are presented in Table 16. The conditions of test F24 were selected as they constituted the best compromise between maximizing concentrate grades while minimizing graphite flake degradation. All size fractions yielded concentrate grades of at least 95.2% total carbon including the smallest size fraction of -400 mesh (-37 microns).

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

The carbon recovery into the combined concentrate was 83.4% in this open circuit test.

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

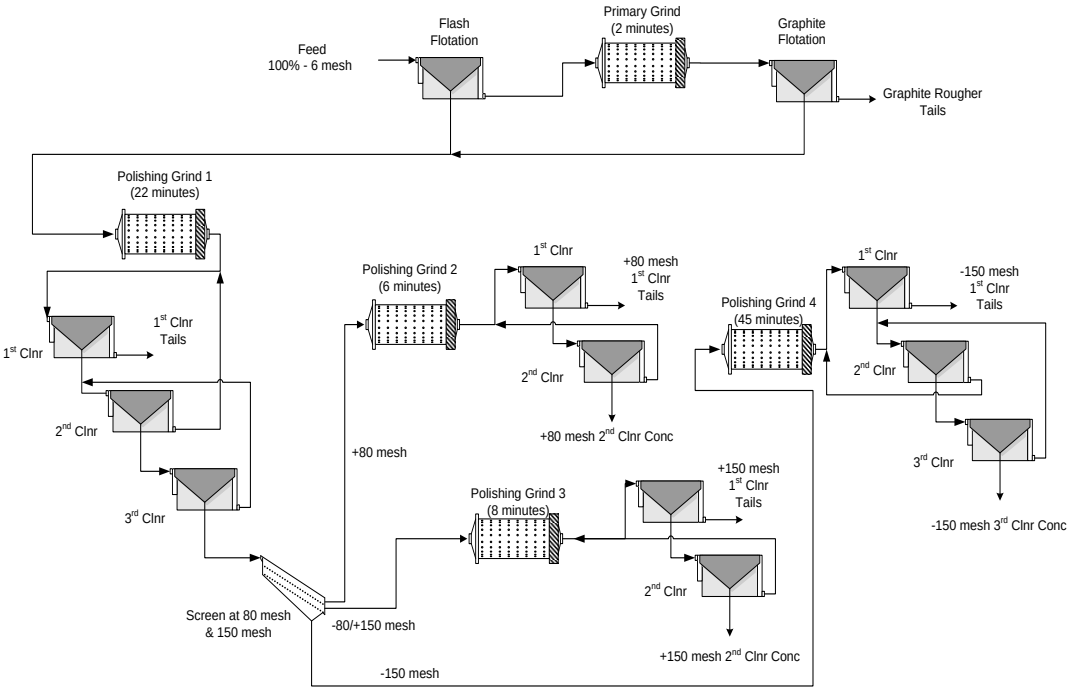


Figure 37: Proposed Molo Mineralization Flowsheet – SGS

Table 16: Size Fractions Analysis Results of Test F24

Product		Mass Distribution (%)	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

13.5 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 38. The first 80m starting from the West were classified as high-grade material and the following 80m as low-grade material. The trench sections were sub-divided into 2m intervals and approximately 2,500 kg of ore were extracted from each interval after removal of any soil and overburden.

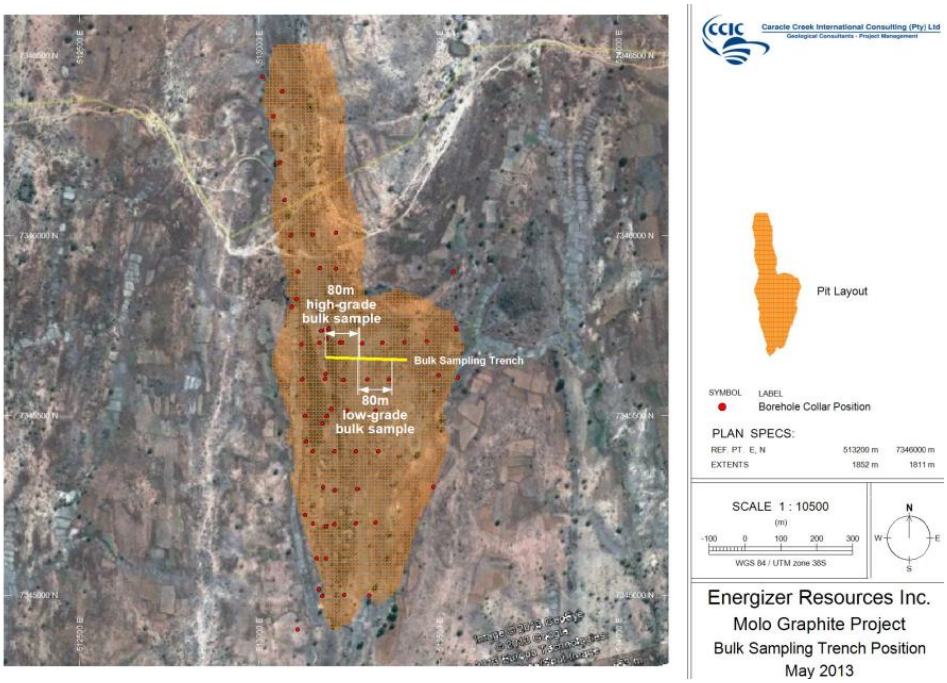


Figure 38: Map showing Position of Bulk Sampling Trench

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

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

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

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

[1] A x b from DWT

[2] A x b from SMC

A total of nineteen pilot plant runs, PP-01 to PP-19, were completed to process the approximately 200 tonnes of the Molo bulk composite using the flowsheet depicted in Figure 39 The only reagents that were employed were fuel oil number 2 (diesel collector) and methyl isobutyl carbinol (MIBC frother) at average dosages of 117 g/t and 195 g/t, respectively.

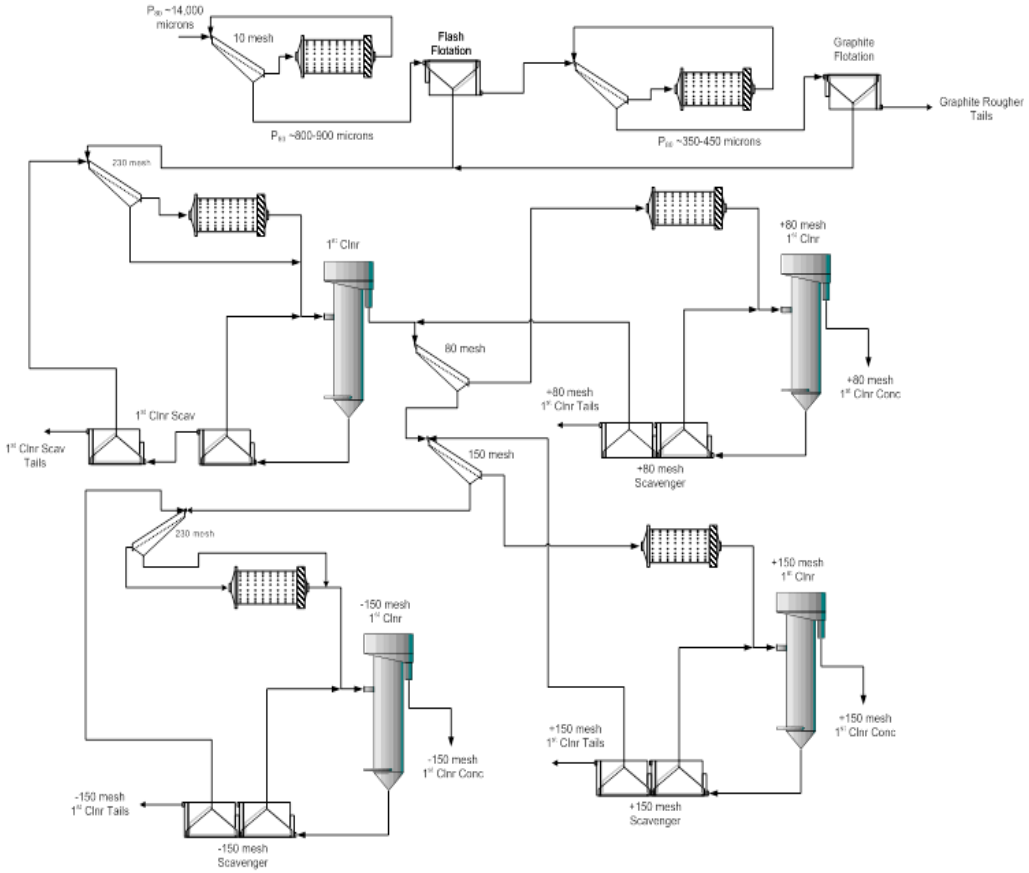


Figure 39: Molo Pilot Plant Flowsheet

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^{TM} 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 P_{80} = 835 microns and P_{80} = 443 microns.

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

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

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

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

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

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

Table 19: Average Combined Concentrate from Fifteen Pilot Plant Surveys

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

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

Table 20.

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

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

Table 20: Comparison of Laboratory and Pilot Plant Metallurgical Results

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

13.6 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 Energizer, 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 are capable of producing 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 employing 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 the majority of 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 in order 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 and concentrate grade.

13.6.1 Sample Selection

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

Table 21: Drill Hole Selection for Optimization and Variability Testing

Description	Drill Hole 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 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 22.

Table 22: Comminution Composites – SGS and Mintek

Lab	Description	Depth (m)	Drill Core Identification
SGS	Comminution Composite #1	14 to 28	Molo 16, 17, and 18
SGS	Comminution Composite #2	57 to 85	Molo 46
SGS	Comminution Composite #3	14 to 28	Molo 34 and 35
SGS	Comminution Composite #4	0 to 14	Molo 29, 30 and 32
Mintek	Grindmill Shallow Composite	0 to14	Molo 15, 18, 35, 36, and 39
Mintek	Grindmill Deep Composite	28 to 56	Molo 15, 29, 37, and 45

13.6.2 Comminution Testing

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

Table 23: Summary of Comminution Tests – SGS

Sample Name and Depth of the Drill Hole Intervals	Relative Density	JK Parameters		RWI (kWh/t)	BWI (kWh/t 500 um	BWI (kWh/t 212 um	AI (g)
		A x B	T_a				
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 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 24 and are consistent with the results obtained for the previous four composites.

Table 24: Summary of Comminution Test - Mintek 2014

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

13.6.3 *Optimization Flotation Program*

A series of six optimization composites were generated to evaluate the two new 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 has to be able to treat all ore within the pit layout, a decision was made to proceed with the majority of the optimization program using the more challenging shallow composite.

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

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

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

13.7 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 25 below.

Table 25: 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 40 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.

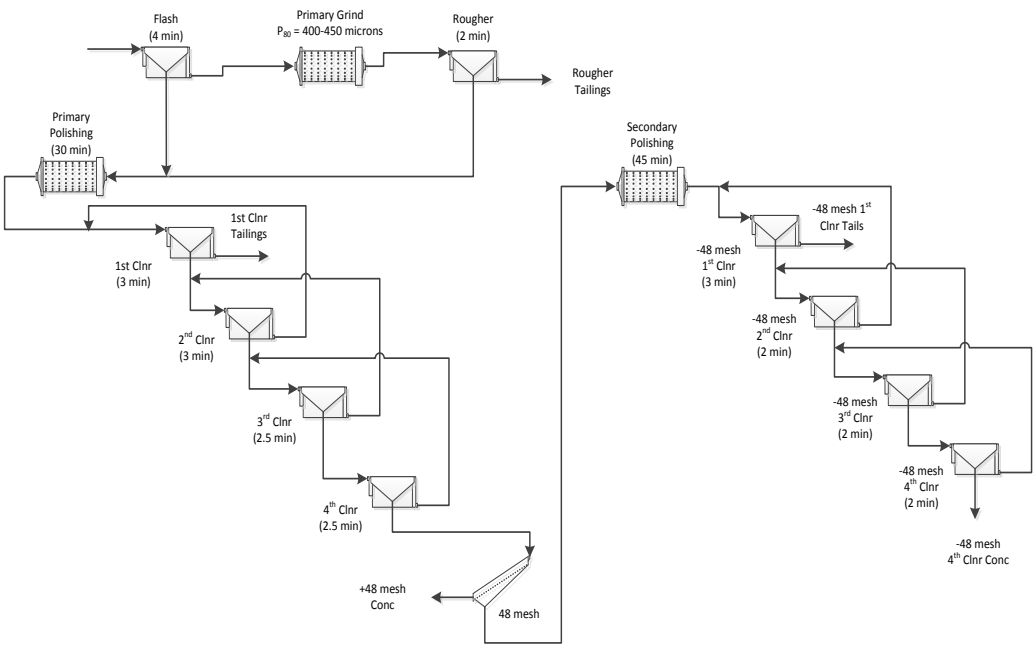


Figure 40: Optimized Flowsheet

Each of the twenty variability composites was subjected to an open circuit flotation test. Composites that failed to produce an acceptable concentrate grade of at least 94% total carbon were subjected to another flotation test using slightly modified conditions such as adjusted flotation times and reagent dosages, longer polishing times, or pH adjustment with lime in cases of a low pulp pH. However, these circuit adjustments failed to produce improved results in most cases.

The mass recovery into the various size fractions is summarized in Figure 41, 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 and 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 head 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.

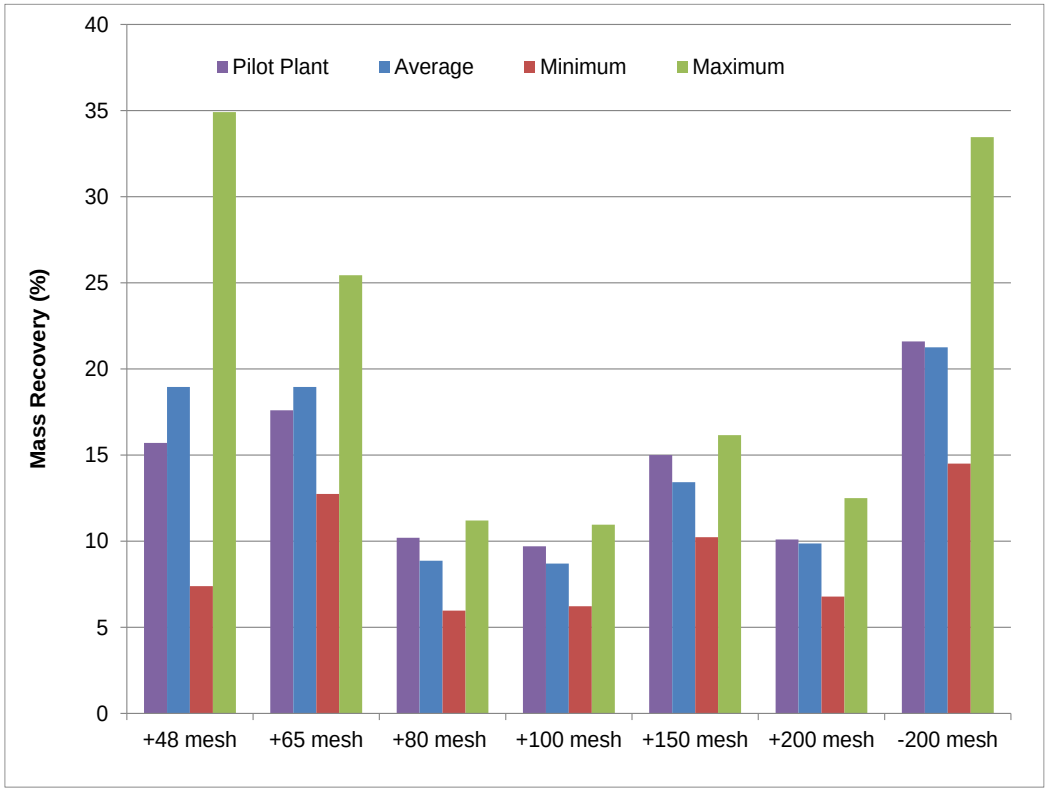


Figure 41: Range of Mass Recovery into Size Fractions (V1 to V32 and Pilot Plant)

The average, minimum, and maximum total carbon grades of tests V1 to V32 for each size fraction are depicted in Figure 42 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.

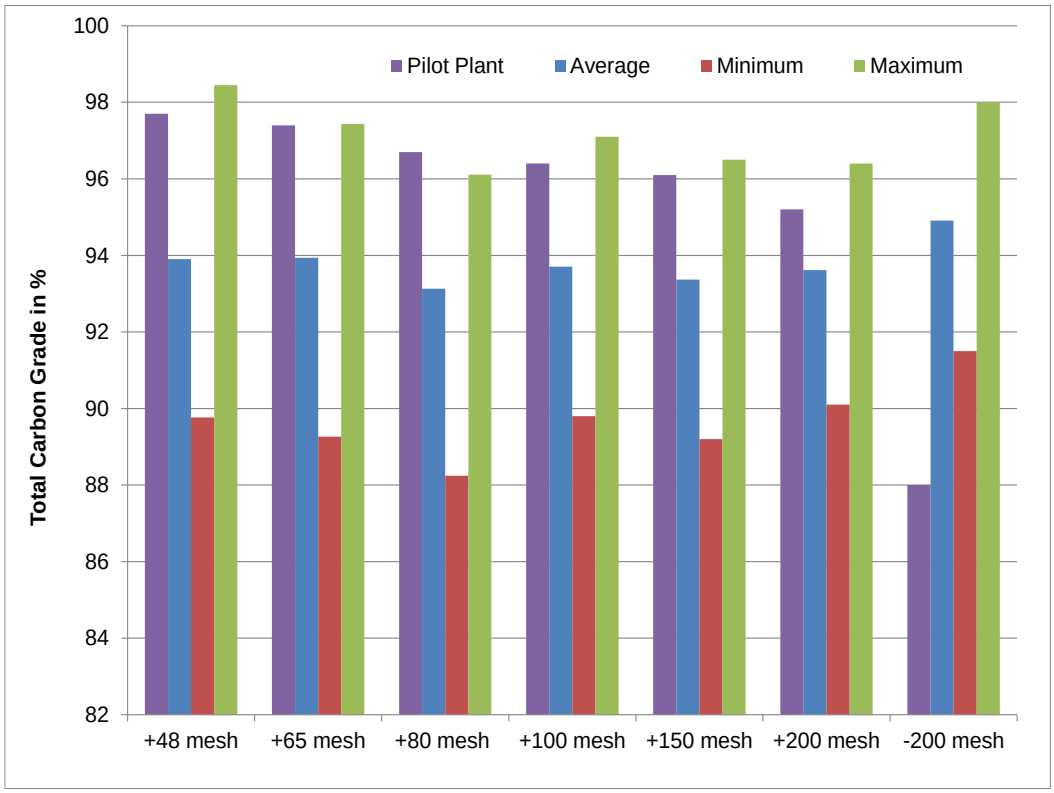


Figure 42: Range of Carbon Grades of Size Fractions (V1 to V32 and Pilot Plant)

The variability in the concentrate grade of the variability composites is further illustrated in Figure 43 which depicts the combined concentrate grade of each of the thirty two variability flotation tests.

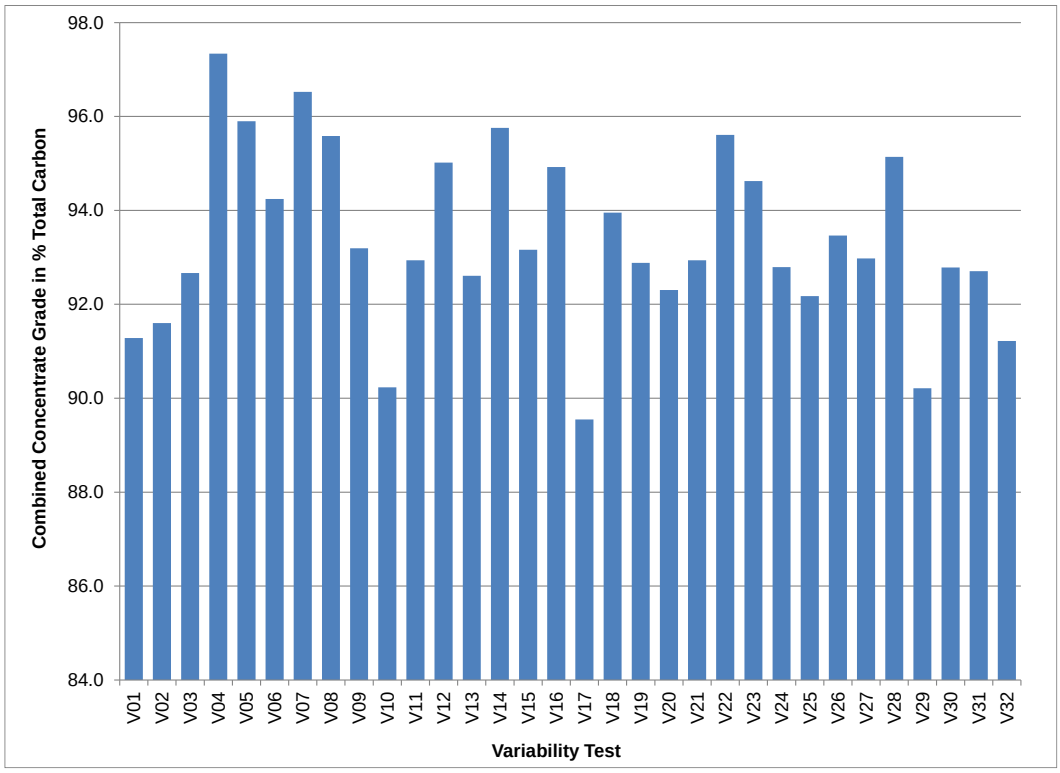


Figure 43: Combined Concentrate Grade of Tests on Variability Composites (V1 to V32)

The location and the metallurgical response of the drill hole intersections that were evaluated in the variability program within the 5 year pit layout are depicted in Figure 44. 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 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.

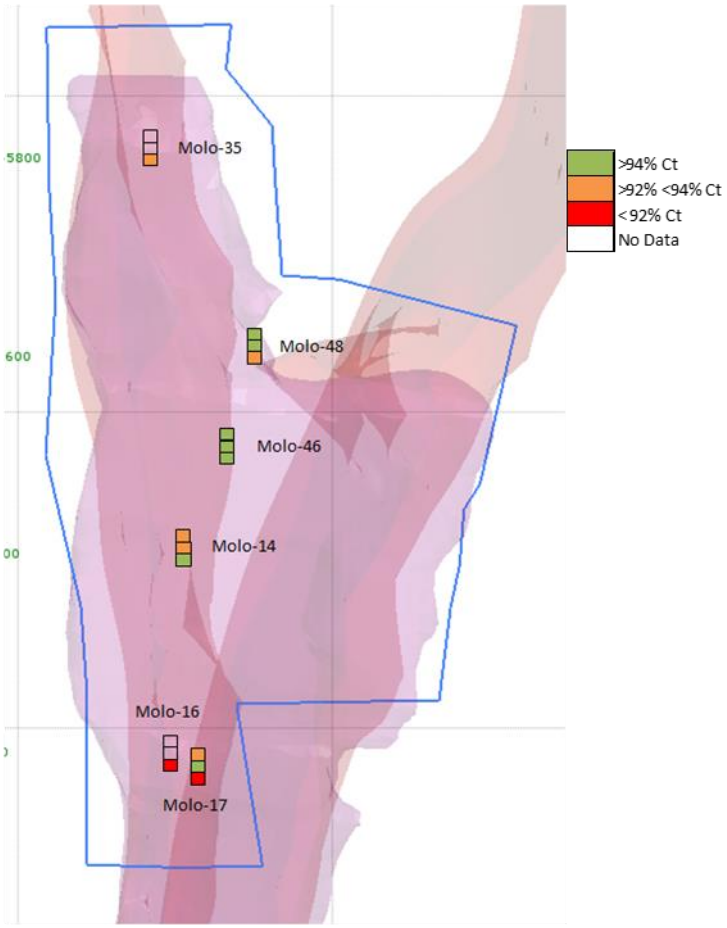


Figure 44: Location and Performance of Phase I Variability Drill Holes

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 of the 5 year pit layout. In discussions with the project geologist it was postulated that the majority 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 substantial variability was linked to this potential transition zone, or if it is encountered throughout the entire 5 year pit area. 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 variability is encountered throughout the entire 5 year pit area, upgrading strategies for the concentrate 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.

13.7.1 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 scrutiny.

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 45. The non-sulphide minerals indicated by the green arrows were interlayered with graphite (red arrows).

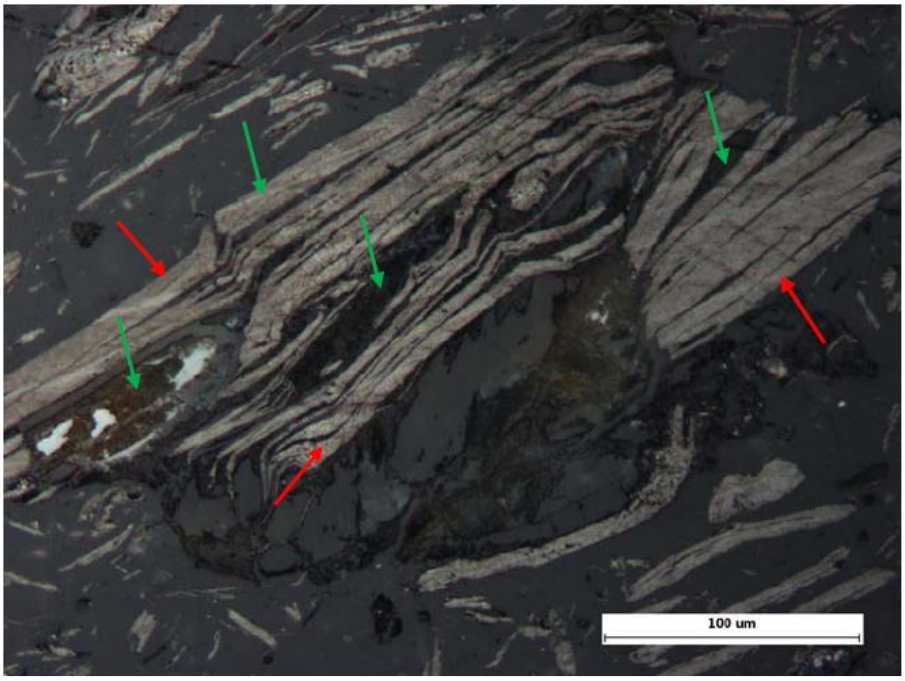


Figure 45: Intercalated Graphite

13.8 Variability Flotation – Phase II

The second phase of variability testing focused on evaluating drill holes in the eastern and western areas of the 5 year pit layout and on developing an understanding of the metallurgical response of the average mill feed within the first 5 years of operation.

13.8.1 Drill Hole Composites

A total of seven drill holes with three depth intervals each were subjected to open circuit cleaner tests to evaluate the metallurgical response of the Molo mineralization in the eastern and western areas of the 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 26.

Table 26: 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 46 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 within the 5 year pit design 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 ceramic grinding media.

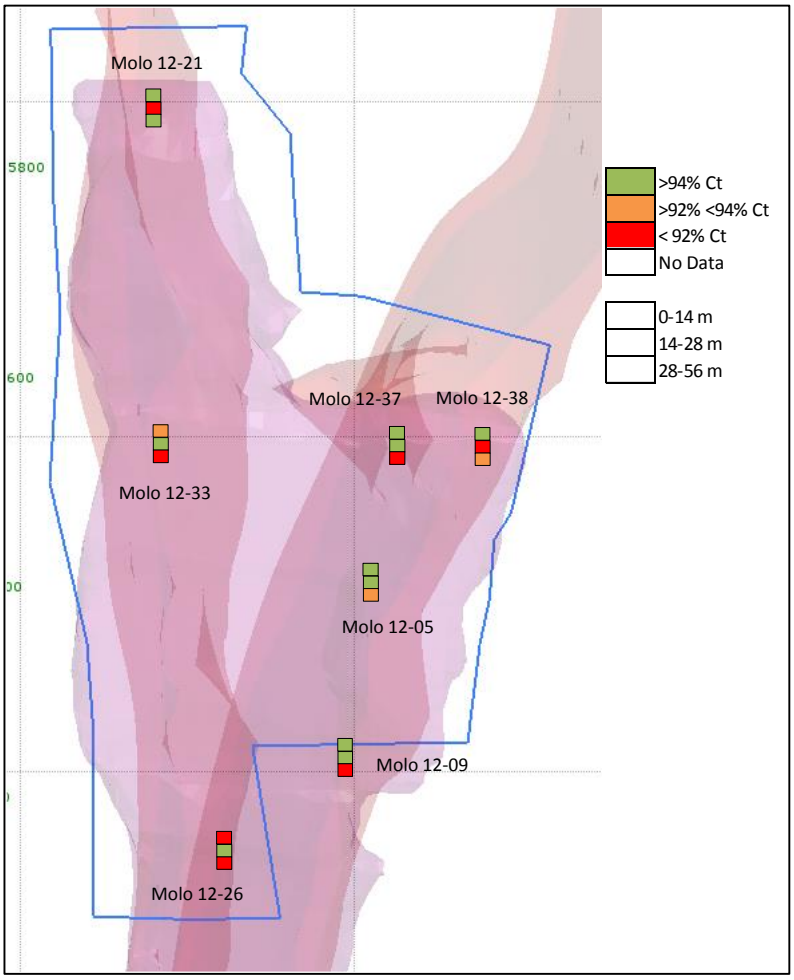


Figure 46: Location and Performance of Phase II Variability Drill Holes

13.8.2 Area Composites

In an attempt to develop a better understanding of the average metallurgical response of the Molo mineralization within the first 5 years of mining operation, area composites were generated, which included samples from all drill holes that were available.

For this purpose, the 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 14m, 14 to 28m, and 28 to 56m 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 27.

Table 27: Drill Holes Included in Area Composites

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

The average flake size distribution of the fifteen area composites is shown in Table 28 below. 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.

Table 28: Comparison of Flake Size Mass 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 47. 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 14m with a concentrate grade of 95.0% total carbon, while the worst performing composite was Area 4: 14 to 28m which produced a concentrate grading of only 89.4% total carbon.

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

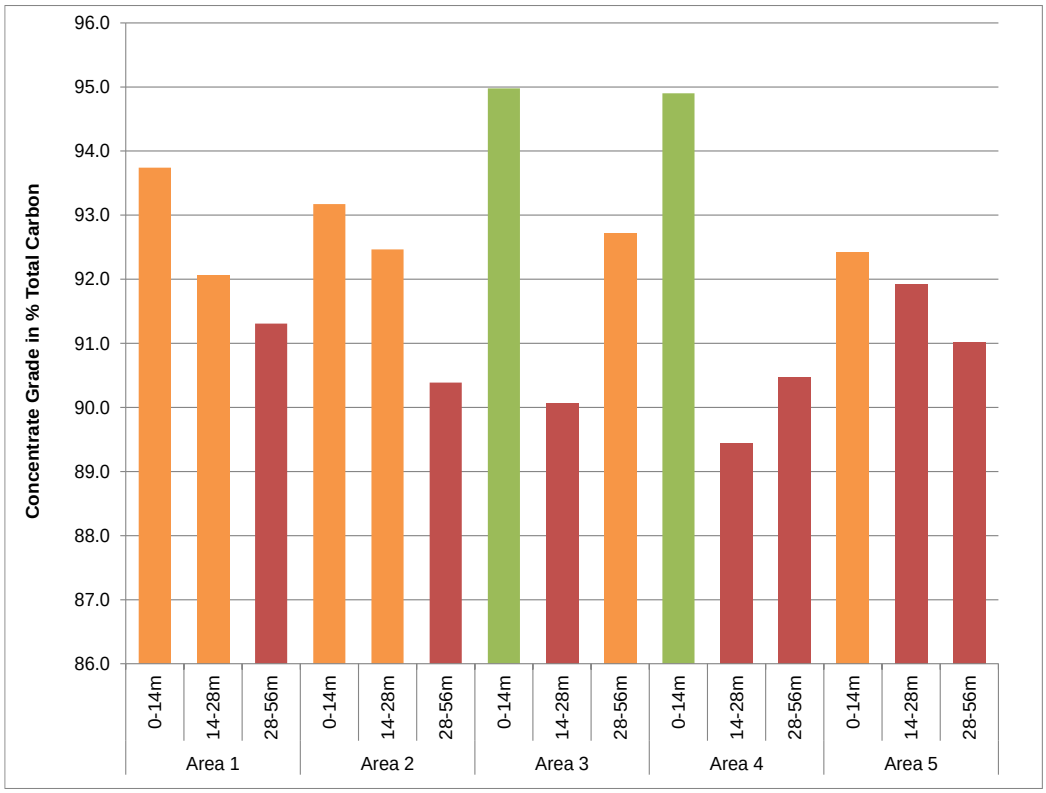


Figure 47: Combined Concentrate Grades of Twenty One Area Composites

The distribution of the five areas within the 5 year pit layout and the metallurgical response of the fifteen area composites are shown in Figure 48. Only the top 14m of the Area 3 and Area 4 composites produced good concentrate grades, which is consistent with the results of the individual drill holes that originated from that area and depth interval.

The conclusion from the Phase II variability program that further upgrading of the graphite flotation concentrate is required was confirmed for the area composites.

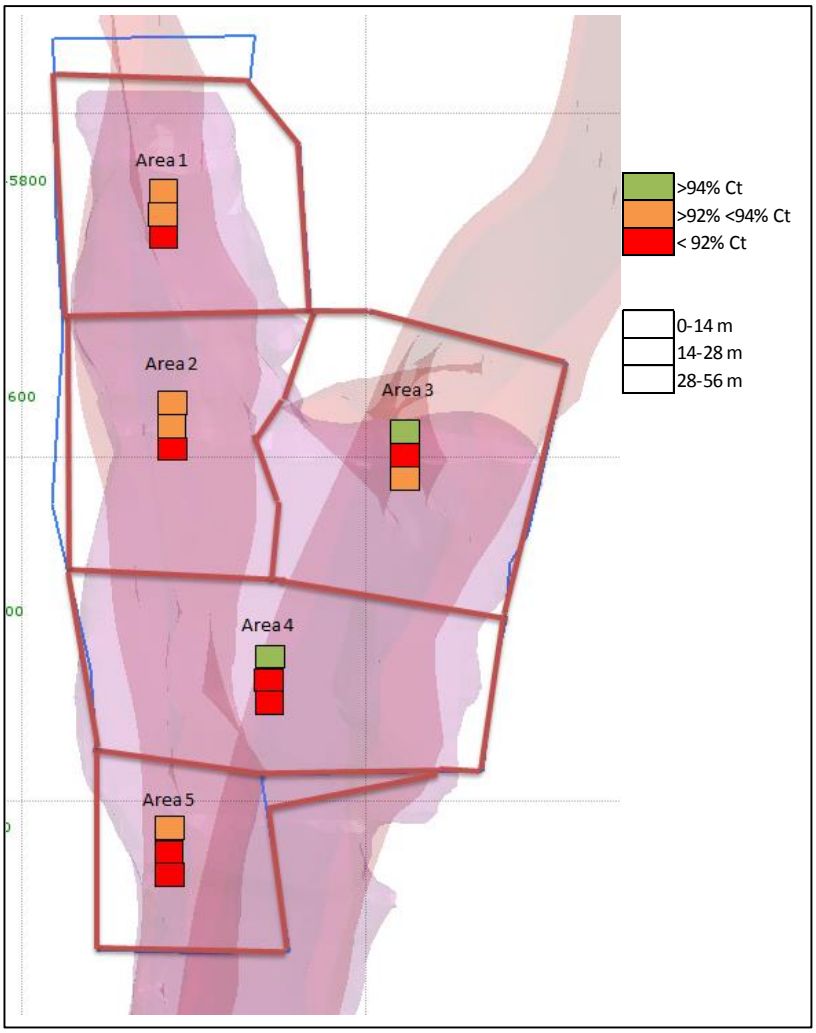


Figure 48: Location and Performance of Area Composites

The average open circuit carbon recovery for the fifteen area composite tests was 87.6% and ranged between 73.2% for the Area 1 is 14 to 28m composite and 93.6% for the Area 3 28 to 56m 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 of 90.5%.

13.9 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 typically achieve 15-25% higher prices compared to a concentrate grading less than 94% 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 the coarse 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 an attrition 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 flotation concentrate generated within the first 5 years of mining operation.

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 attrition mill. This is evidenced in Figure 49 and Figure 50 which depict the mass recovery into the size fractions of the final concentrates of tests conducted with the attrition mill and attrition scrubber, respectively. In order to quantify the 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 attrition 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 attrition mill reduced the mass of the +48 mesh and -48/+65 mesh concentrate by only 2.6% and 2.7%, respectively.

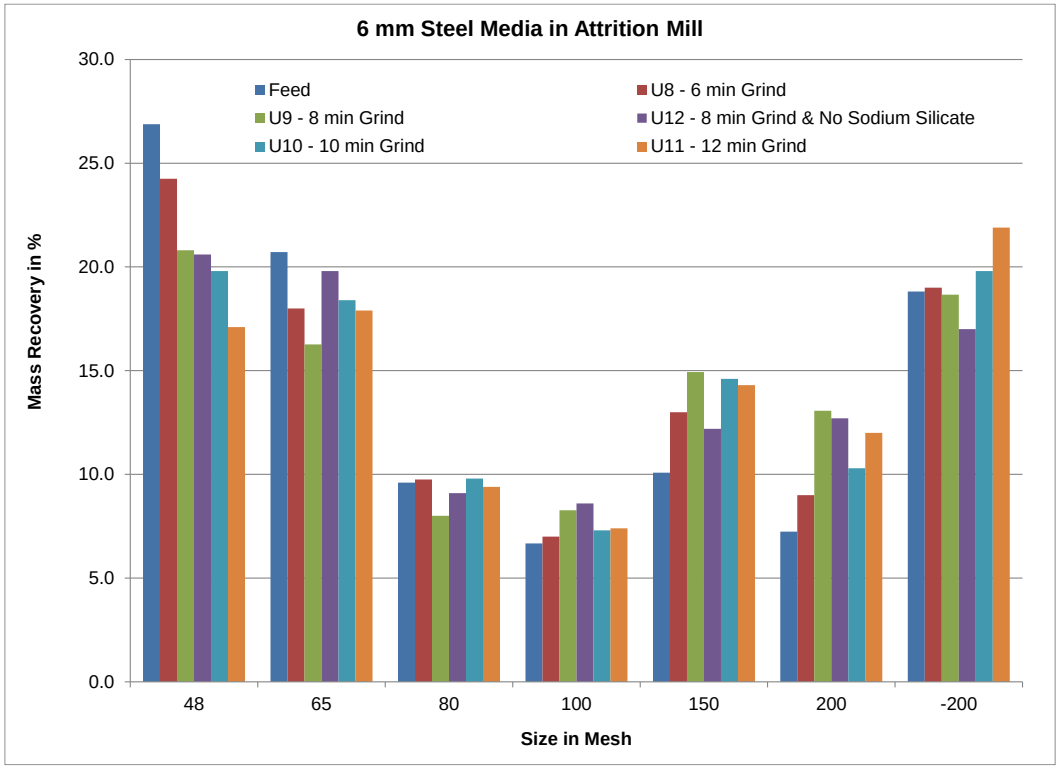


Figure 49: Attrition Mill Size Fraction Analysis – Mass Recovery

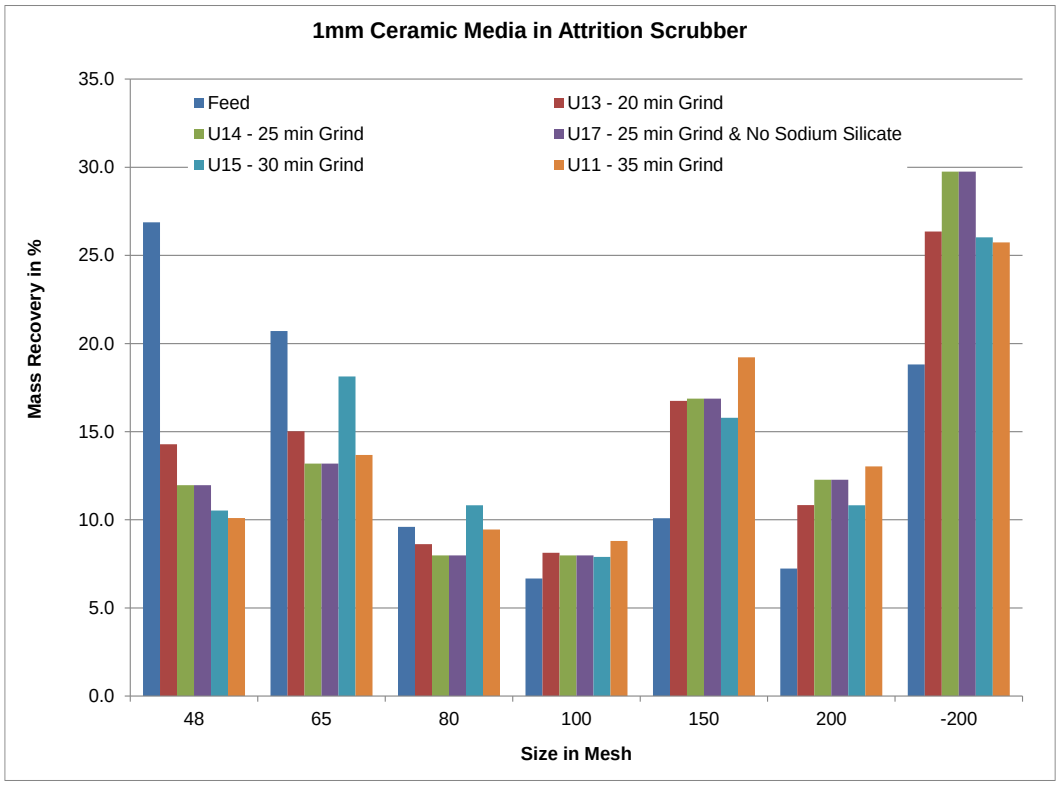


Figure 50: Attrition Scrubber Size Fraction Analysis – Mass Recovery

The total carbon grades into the size fractions of the combined concentrate for the five tests using the attrition mill are presented in Figure 51. The shortest grind time of 6 minutes produced concentrate grades of 96.9% to 98.1% total carbon. 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.

As the flake degradation increased at 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 attrition mill yielded at least 96.5% total carbon, which attests the robustness and repeatability of the upgrading approach.

A comparison of the results of tests U9 (40 kg/t concentrate of sodium silicate) 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.

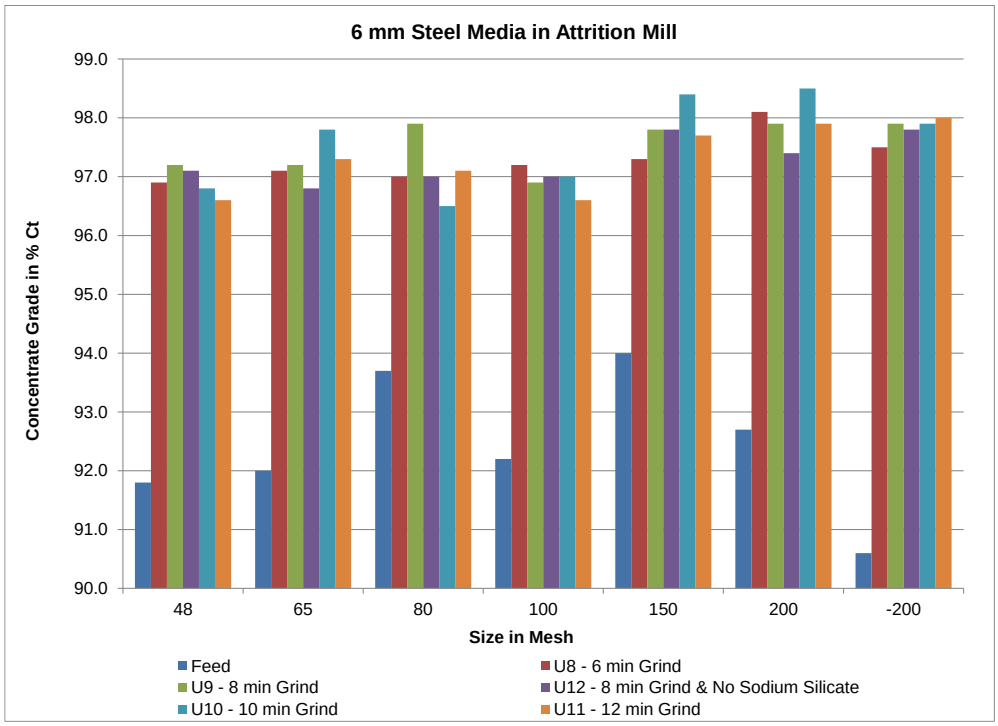


Figure 51: Concentrate Grades of Size Fractions – Attrition Mill

The complete Molo flowsheet with the attrition circuit is depicted in Figure 52.

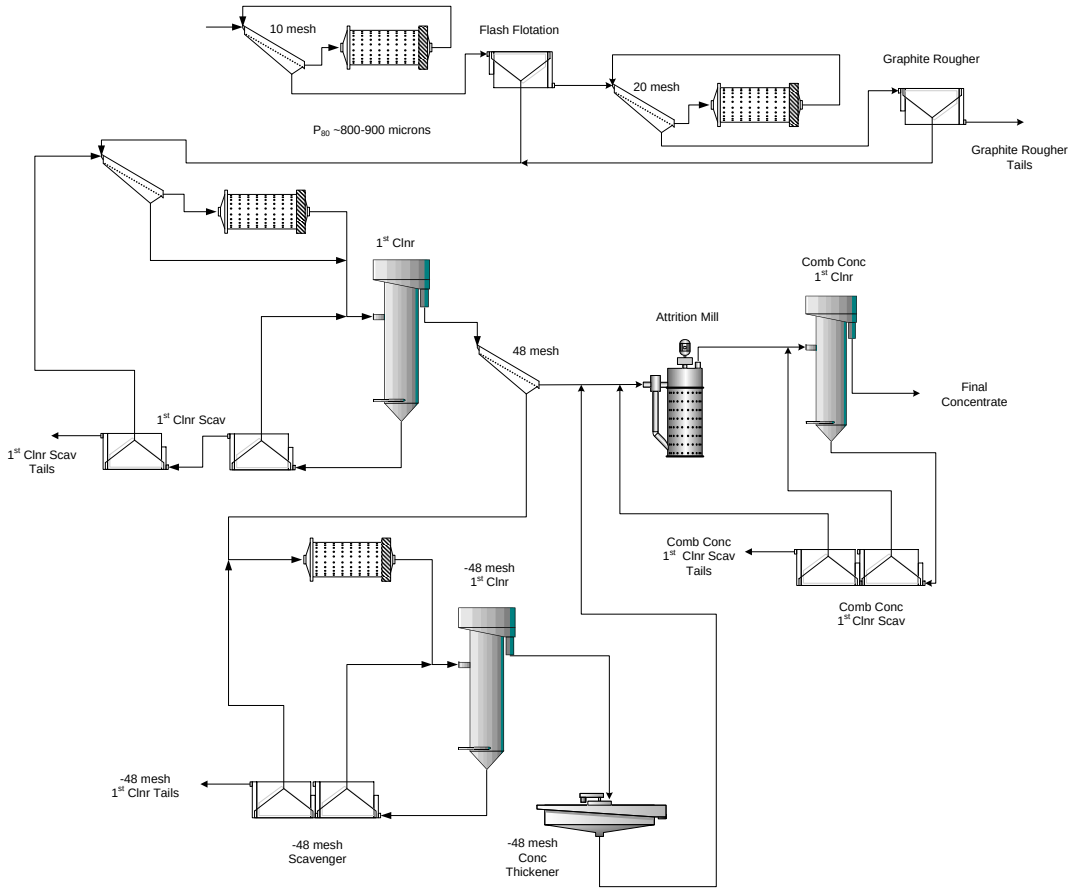


Figure 52: Molo Flowsheet including Attrition Circuit

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 five years of mining operation.

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

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

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

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

Product		Mass Distribution (%)	Grade (% Total Carbon)
Mesh	**Microns**		
+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**

13.9.1 Optical Mineralogy of Upgraded Concentrate

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

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 53 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 attrition mill, but that the thinner layers between graphite and NSG minerals as seen cannot be segregated.

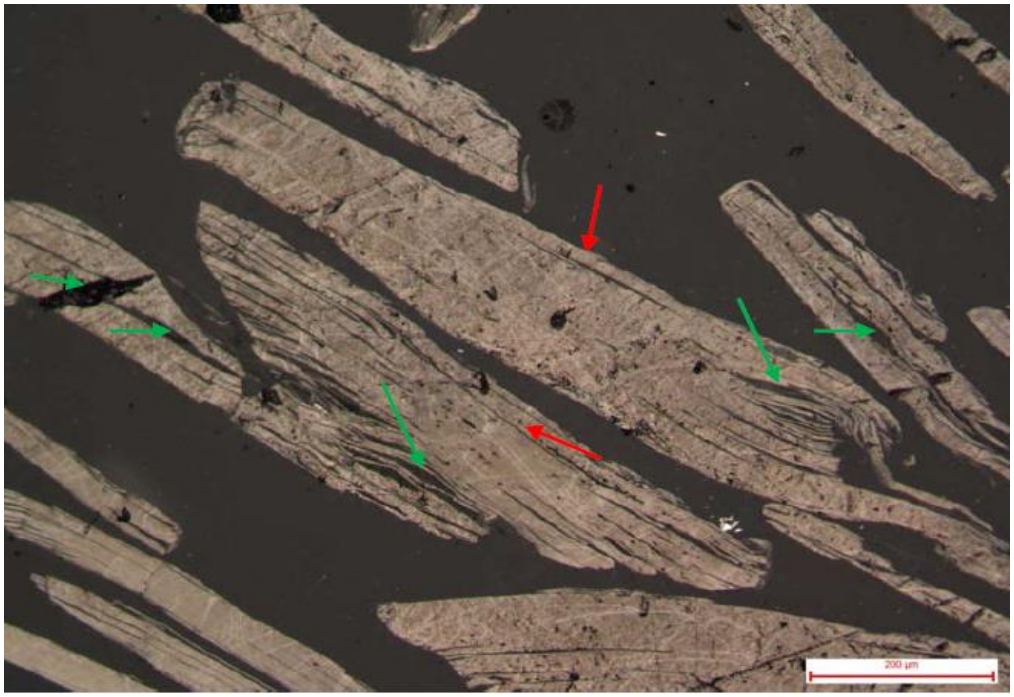


Figure 53: Optical Photomicrographs (PPRL) from the U-8 SFA 3rd Clnr Conc +48 Mesh

13.10 Variability and Risk Discussion

The metallurgical test programs have revealed that the Molo mineralization is quite variable with regards to the graphite concentrate grades as a result of the interlayering between graphite and non-sulphide gangue minerals. An upgrading circuit was designed to eliminate the majority of the coarse interlayering, thus resulting in a significant improvement of the concentrate grade from 92.1% to 97.1% total carbon. The upgrading circuit did not eliminate very intricate interlayering between graphite and non-sulphide gangue minerals. However, based on the degree of upgrading that was achieved on a combined concentrate representing the first five years of mining operation, complete elimination of interlayering is not required.

The entire process development, optimization, and variability testing was completed on drill core samples that originated from within the area of the first 5 years of mining operation and no metallurgical results are available for samples from areas that will be mined after the first 5 years. The biggest risk with regards to metallurgical response associated with ore that will be mined in years 6 onward is the degree and form of interlayering that may be encountered. The combined concentrate representing the first 5 years of mining operation was readily upgraded above the minimum required concentrate grade of 94% total carbon, which suggests that the process circuit will be able to process ore with a higher percentage of intricately interlayered graphite and still met minimum concentrate quality requirements.

The upgrading circuit has only been demonstrated on a laboratory scale to-date. The evaluation of alternative grinding media and larger scale tests including vendor testing will help to reduce the process risk associated with this part of the circuit. At the same time, additional testing has an upside potential with regards to final concentrate grade and/or flake size distribution if other attrition media proves to be more suitable in eliminating the interlayering.

As with any metallurgical test program, the results and conclusions are limited to the drill core and/or trench samples that were evaluated in the laboratory and pilot scale metallurgical programs. The variability programs incorporated all drill holes that were available within the critical first 5 years of operation. In the case of the area composites a total of thirty-one drill holes were included in the fifteen area composites, which provides a very good indication of the expected average plant performance during that period of time. In addition, forty-one individual drill hole intervals were subjected to variability testing. These samples were selected to represent the expected variance of the first five years of mill feed with respect to feed grade, spatial and vertical distribution, and mineral domain.

Deleterious elements and permissible concentrations are process specific and differ from end user to end user. In addition to impurities, other physical and chemical properties affect the attractiveness of a given concentrate for end users. End users typically request representative samples of graphite concentrates and carry out their own in-house and proprietary analysis on the samples.

13.11 Additional Testing

The physical properties of graphite are very different to most other commodities as a result 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.

13.11.1 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 one quantified a solids loading rate of 0.25 t/m^2/h for the concentrate thickener yielding an underflow density of 36% solids at a flocculant dosage of 5 ppm Magnafloc® 919. The overflow contained 230 ppm solids. The tailings required the addition of a coagulant to achieve satisfactory overflow clarity of less than 100 ppm. The reagent regime consisted of 5 ppm Magnafloc® 1011 and 500 ppm Magnafloc® 370. An underflow density of 50% solids with an overflow containing 70 ppm solids was achieved at a solids loading rate of 0.75 t/m^2/h.

Vendor two quantified the solids loading rate for the concentrate thickener at 0.64 t/m^2/h at a flocculant SNF 905 VHM dosage of 20 g/t. The solids loading rate for the graphite tails was 0.46 t/m^2/h at a flocculant SNF 934 VHM dosage of 140 g/t. The concentrate and tailings thickener underflow solids concentration at 2 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 one 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 2 failed to generate an acceptable graphite thickener tails overflow clarity. Two reagent suppliers, supplier 1 and supplier 2, were contracted to carry out a more comprehensive reagent screening to evaluate a reagent regime requiring lower dosages.

Supplier one 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 two did not develop a reagent regime that achieved dosages lower than the ones recommended by equipment Vendor one.

13.11.2 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/m^2/h. Filtration tests on the tailings produced filter a cake moisture content between 12.7% and 17.9% w/w at filtration rates of 92 to 218 kgDs/m^2/h.

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

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

13.11.3 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.

13.11.4 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.

14 MINERAL RESOURCE ESTIMATES

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 Energizer 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 54). Elevations within the area of study range from 570 mamsl to 543 mamsl, with an average gradient of 2.0º. The high grade domain on the western side creates a ridge up to 2.0 m in elevation.

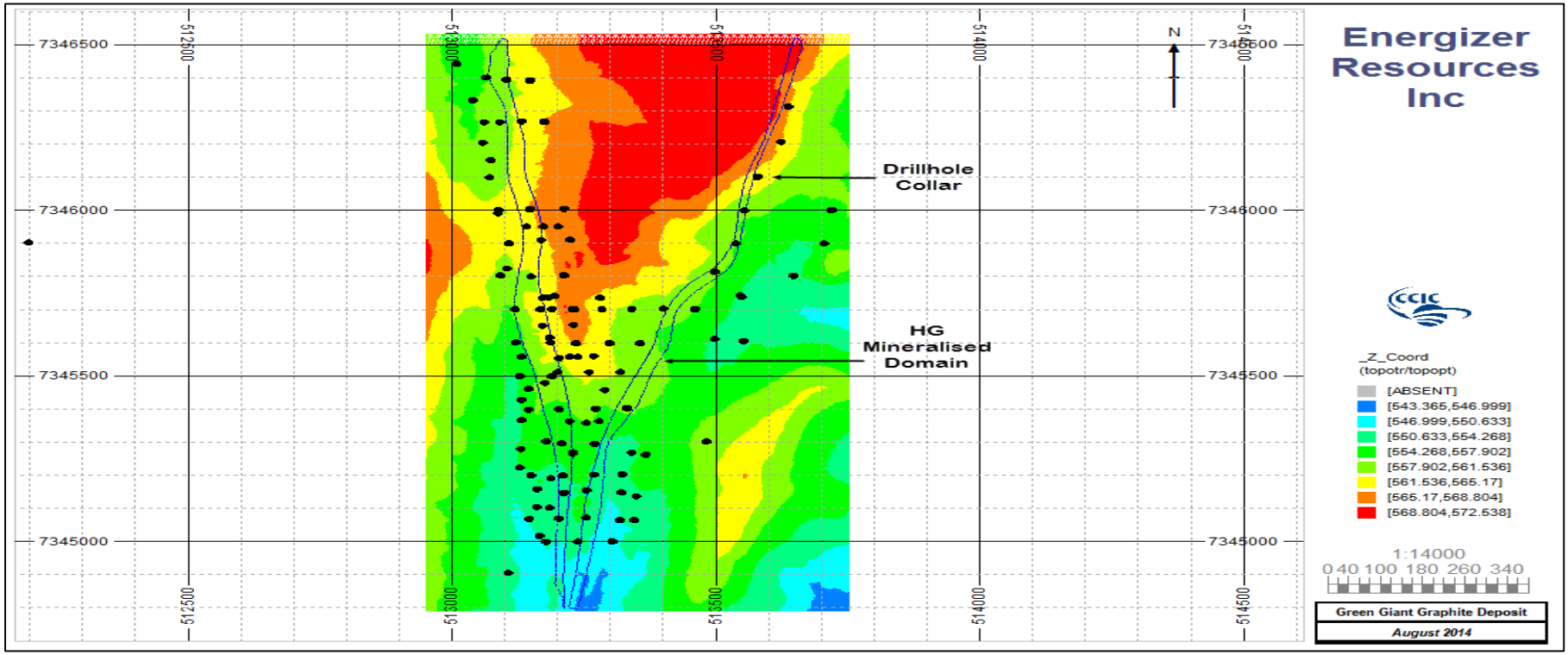


Figure 54: Topographic Contours. Elevation is in mamsl

14.1.2 Drill Holes

This Mineral Resource estimate is based on 80 drill holes (total 11,660m) and 35 trenches (total 8,492m) drilled by Energizer. Drill spacing varies from 100m * 100 m in poorly informed areas to 50 m * 50 m in well informed areas. Figure 31 below 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 Energizer in a Microsoft Access format. Logging codes used for lithological modelling are summarised in Table 30.

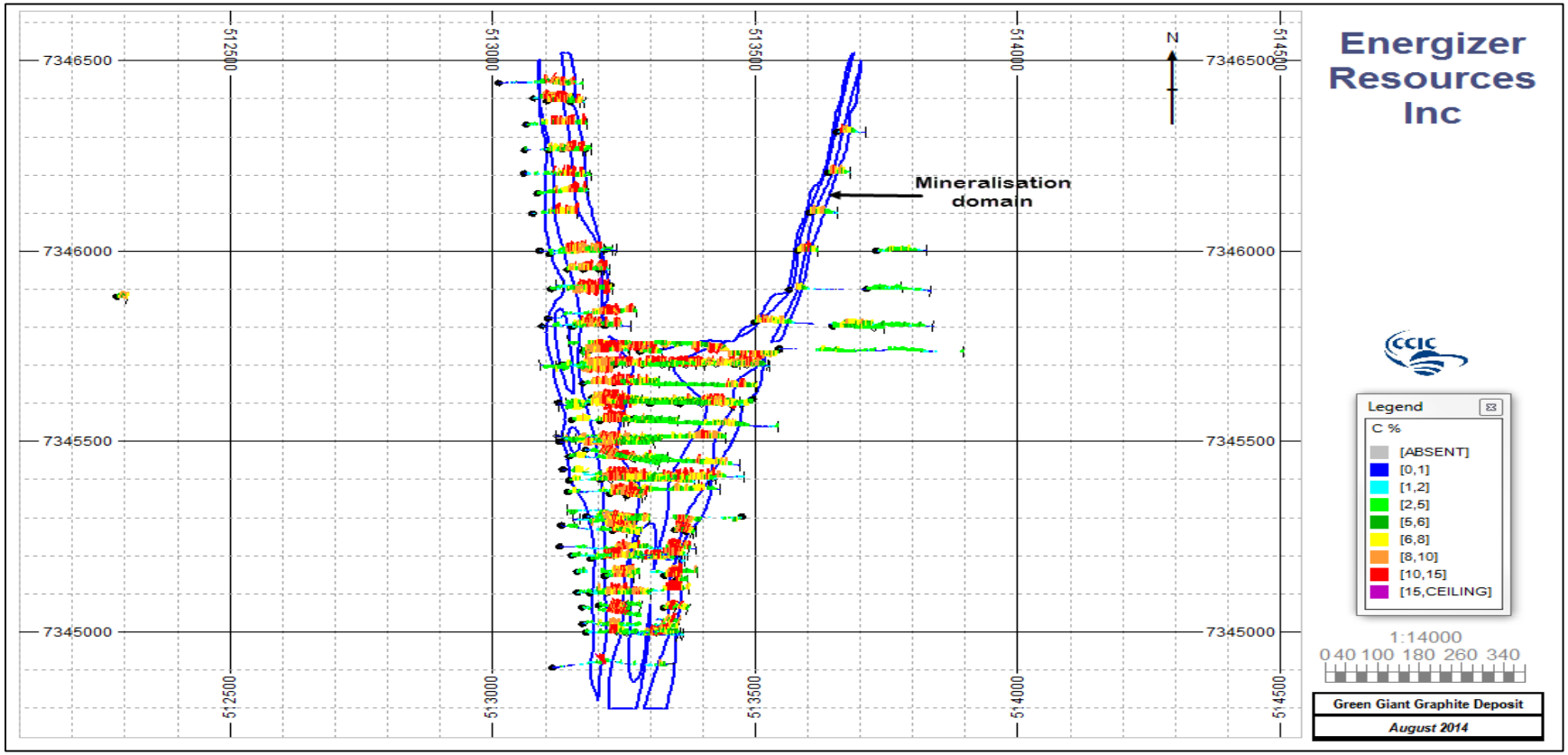


Figure 55: Drill Hole Posting Plan Coloured on C (%) Grades

Table 30: Summary of Lithological Codes used

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

Spatial and statistical comparisons of % C between drill holes and trenches are presented in Figure 56 and Figure 57. 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.

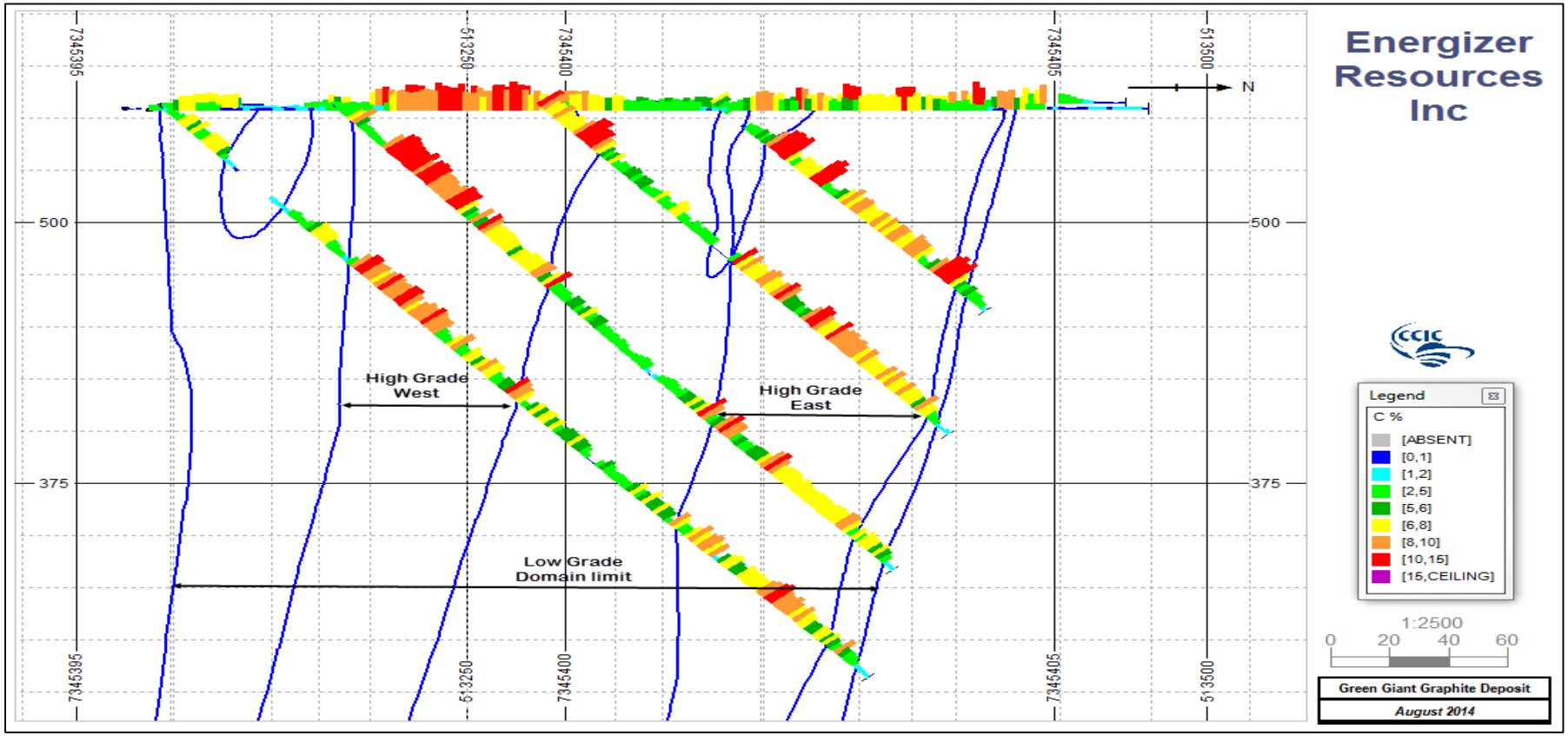
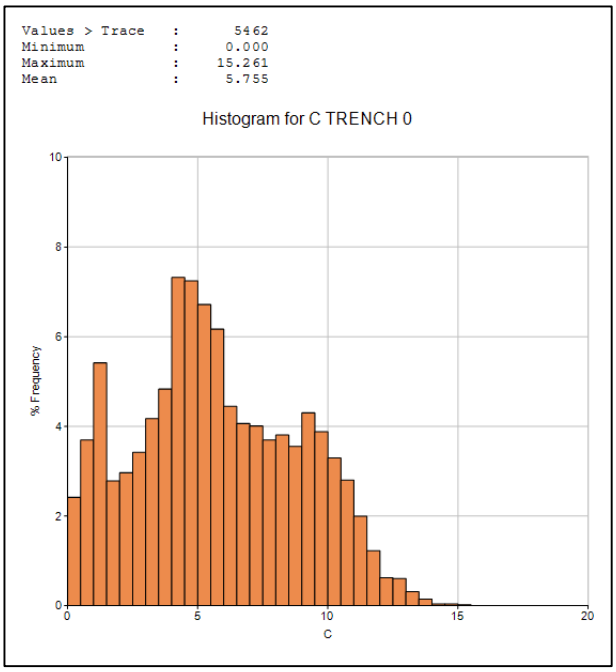


Figure 56: Cross Section showing the Grade Distribution (C%) in the Drill Holes and Trenches



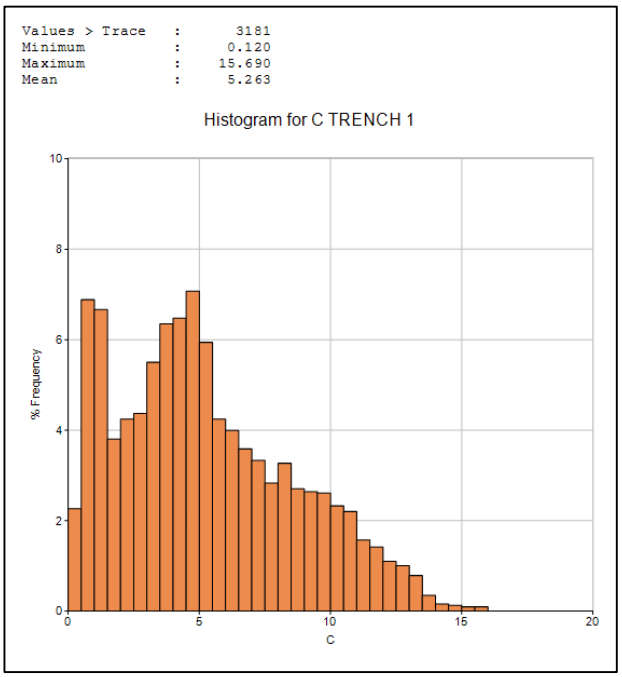


Figure 57: Statistical Comparisons between Diamond Drill Hole (DD) and Trench (TH) Samples

14.1.3 Relative Density

A total of 226 RD measurements are contained in the Molo database, 179 of which were taken from the GpGn. See Figure 58.The average RD for all 226 readings is

2.39 t/tm^3. RD values within the GpGn range from 1.59 t/tm^3 to 2.95 t/tm^3, with an average of 2.35 t/tm^3.

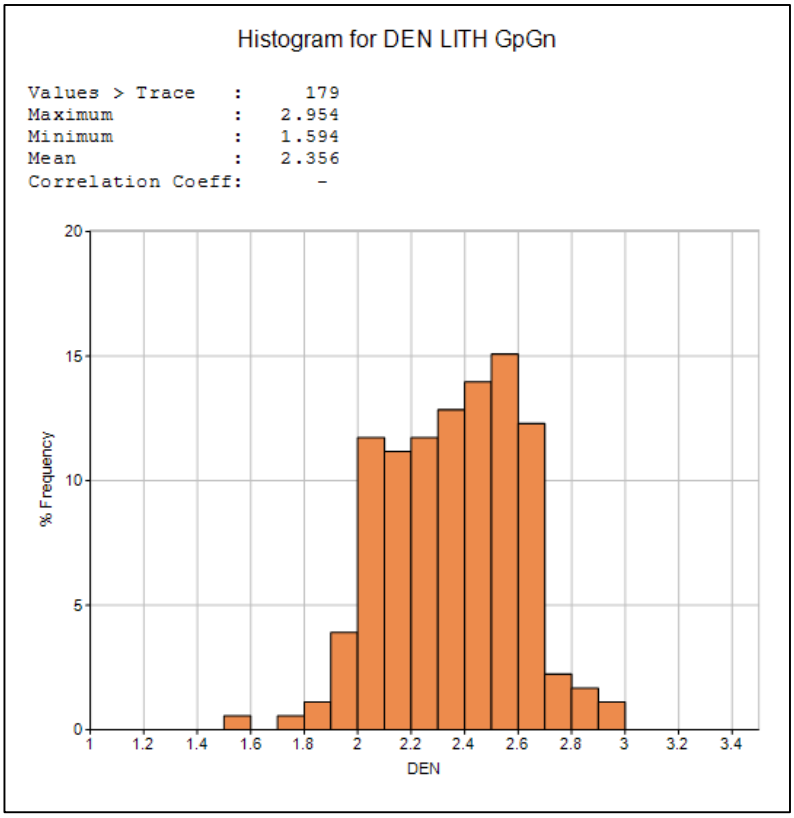


Figure 58: Histogram of Relative Density Readings within the Graphitic Gneiss

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 Figure 59 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.

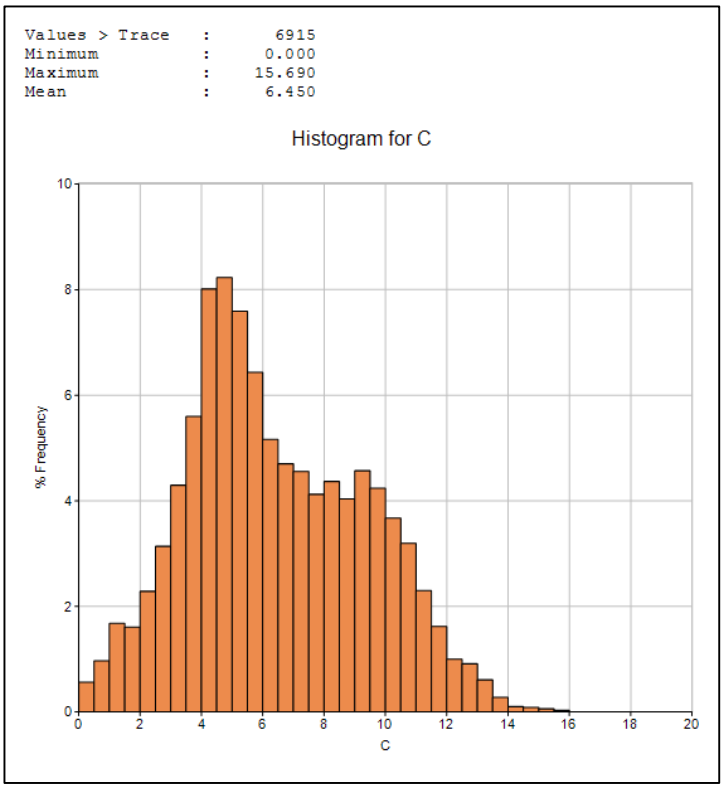


Figure 59: Histogram of C Distribution

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 sub-divided 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 60 and Figure 61 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 60.0m, where only the “high grade west” domain is developed, to more than 260.0m in the central portions. “High grade” domains are generally 60.0m thick, thinning to the south.

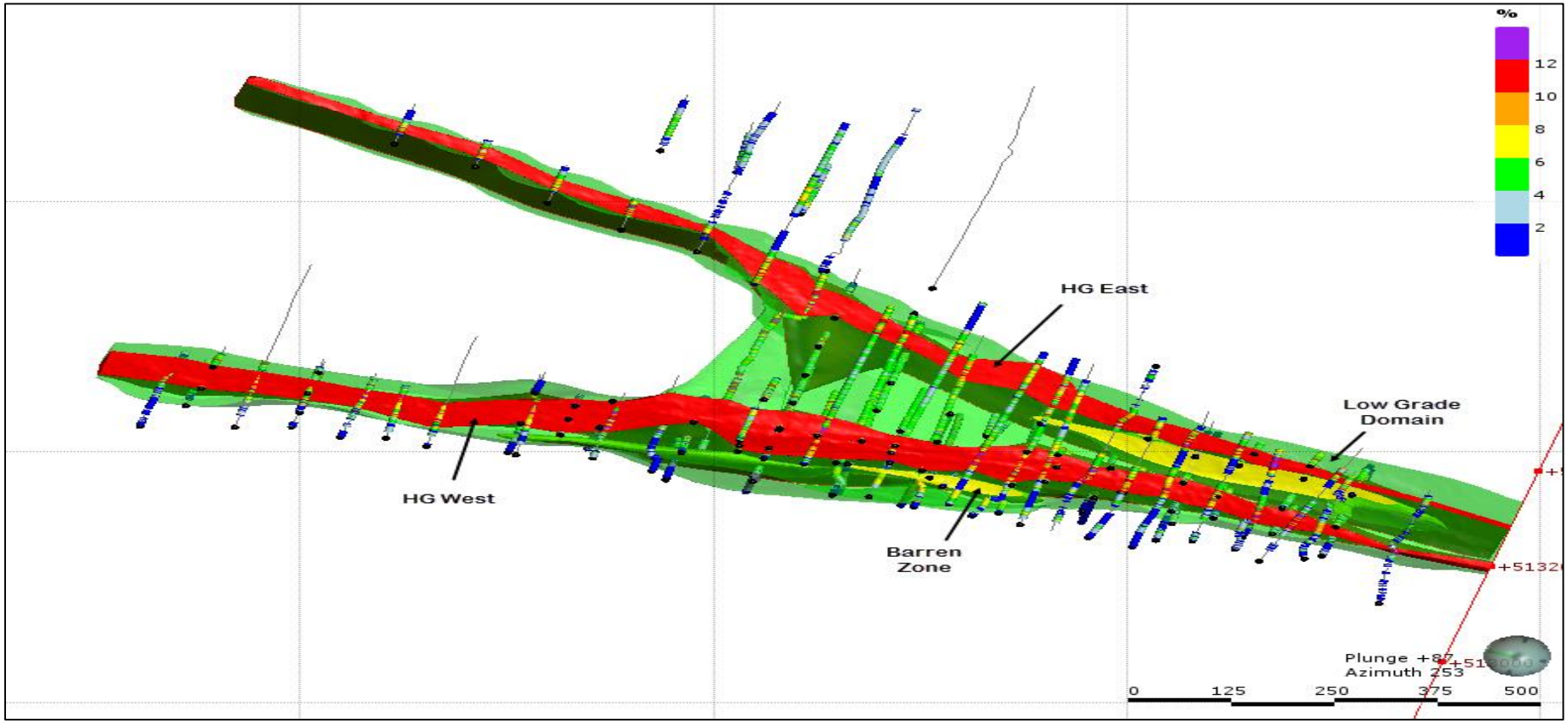


Figure 60: Isometric View showing "Low" and "High" Grade Domains

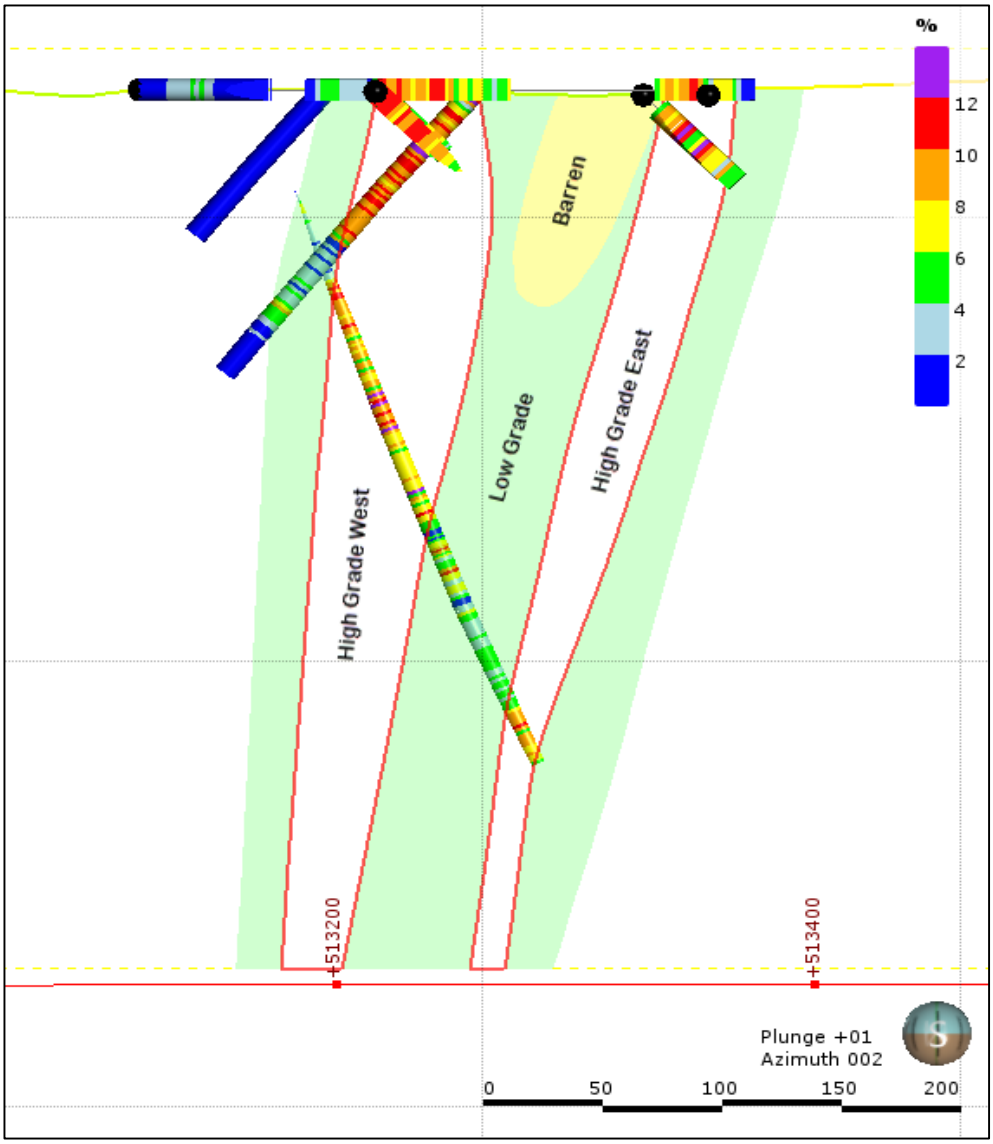
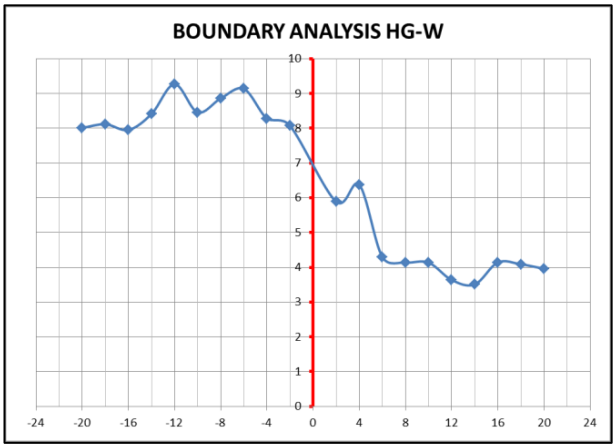


Figure 61: West East Section showing the "Low" and "High" Grade Domains

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



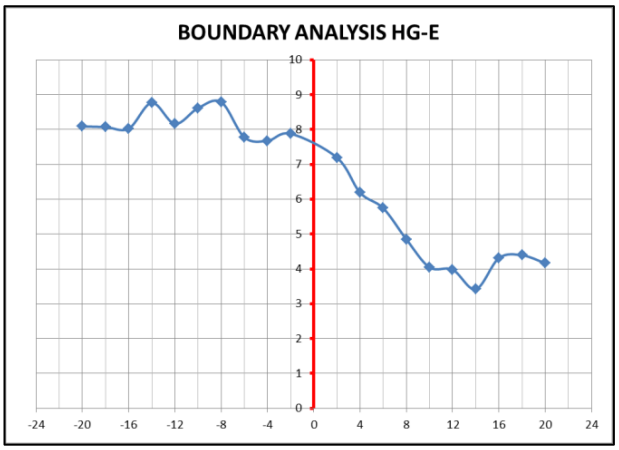


Figure 62: "High" and "Low" Grade Boundary Analysis

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

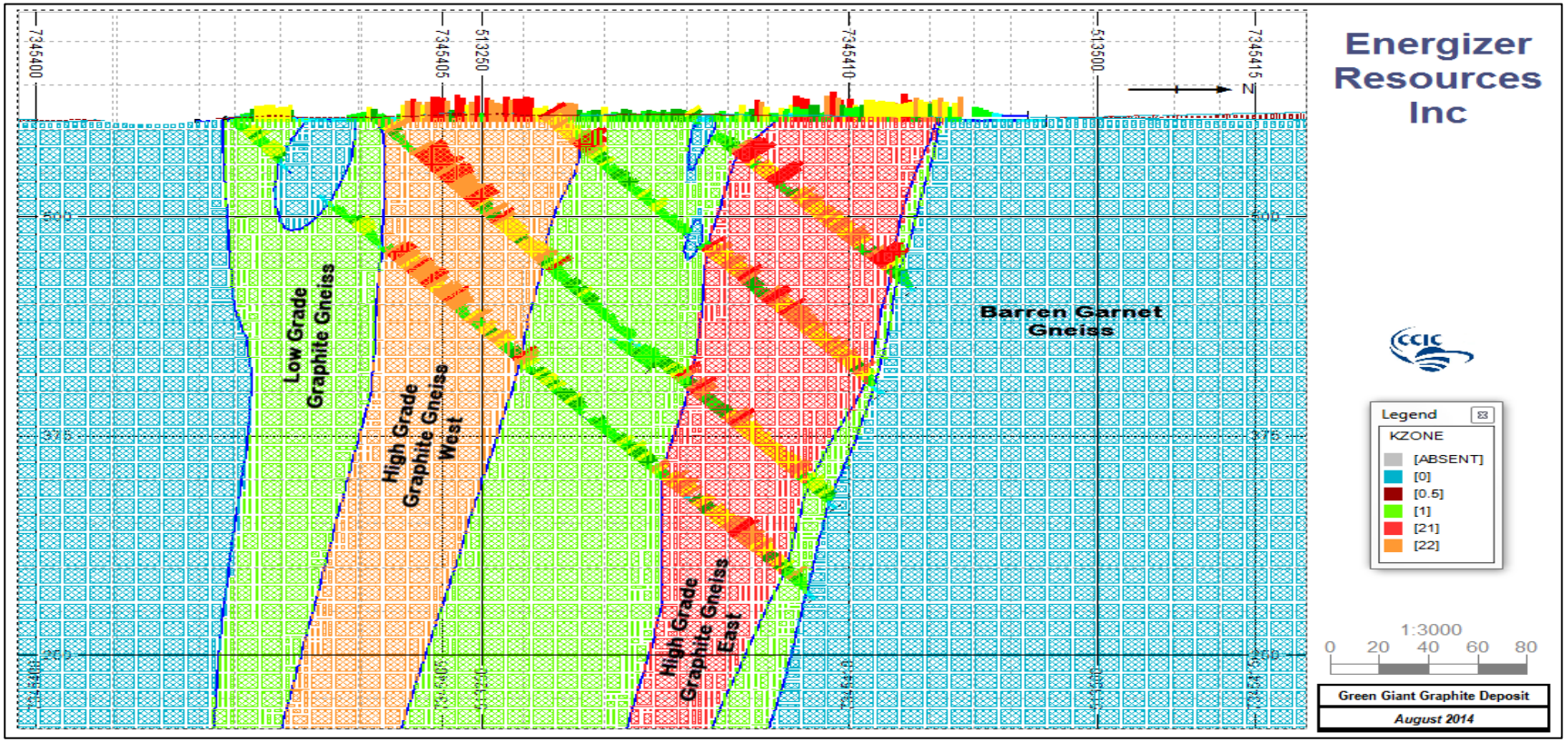


Figure 63: Cross-section showing the KZONE flagging in the block model

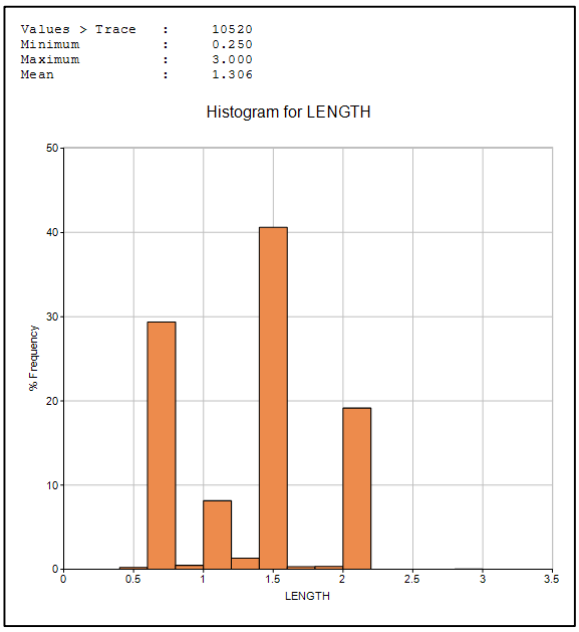
A summary of the KZONE flagging undertaken in Datamine is provided below in Table 31.

Table 31: Summary of KZONE Flagging in Datamine

Zone Description	KZone Value	Code Name
Garnet gneiss barren zone	0	WST
Overburden	0.5	OVB
Low Grade mineralised zone	1	LG
High Grade mineralised zone East	21	HG -E
High Grade mineralised zone West	22	HG -W

14.3 Compositing

The predominant sampling interval was either 0.5m, 1.0m, 1.5m, or 2.0m, and hence a composite length of 2.0m was used to include all samples, (Figure 64). Compositing used the KZone to ensure that samples were composited within the different domains. Any samples less than 0.5m after compositing were excluded from the geostatistical analysis and estimations.



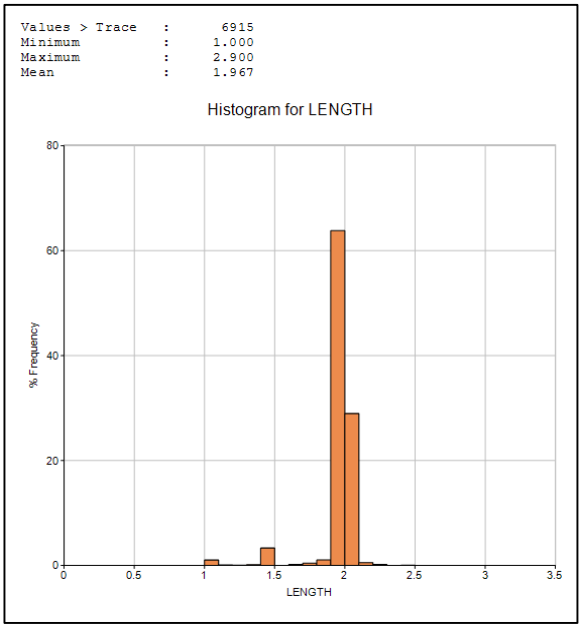


Figure 64: Histogram of Sample Length Prior to and after Compositing

14.3.1 Composited Statistics

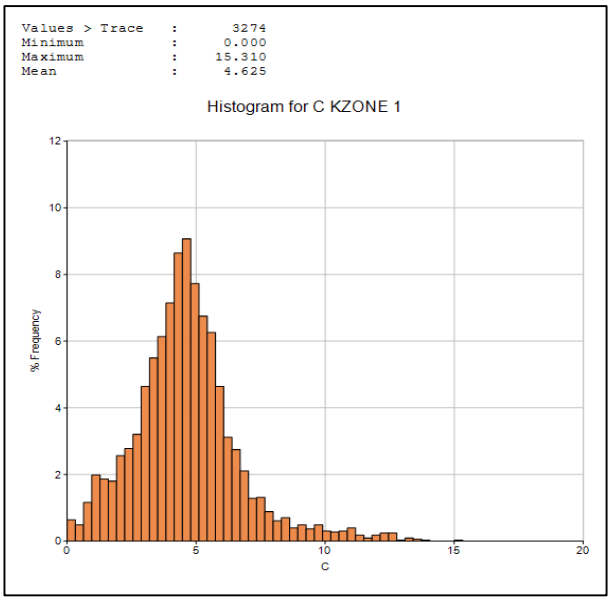
The sampling protocol for core intervals deemed to be barren with respect to graphite mineralisation is that they were not to be submitted for analyses. All un-

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 32 below.

Table 32: 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 65, Figure 66 and Figure 67. The sample distributions for both the “high grade” domains exhibit very similar statistical characteristics.



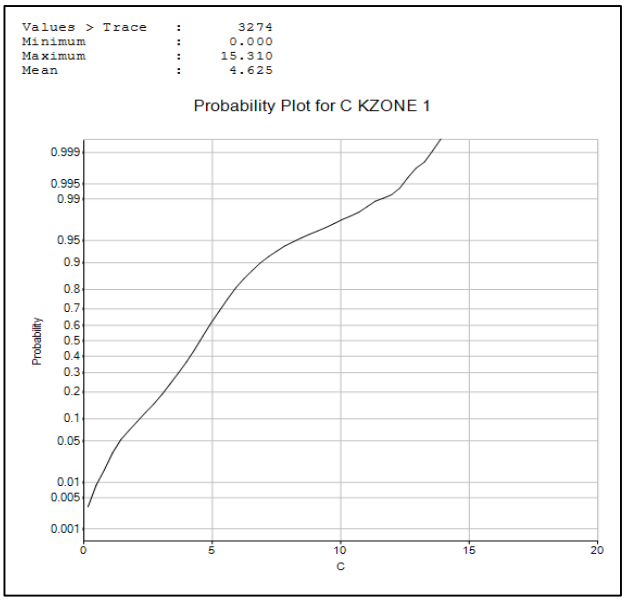
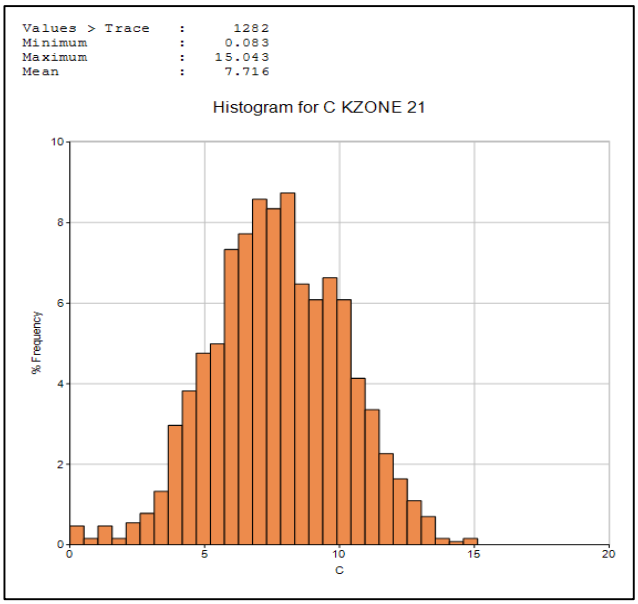


Figure 65: Sample Distributions for the "Low Grade" Domain



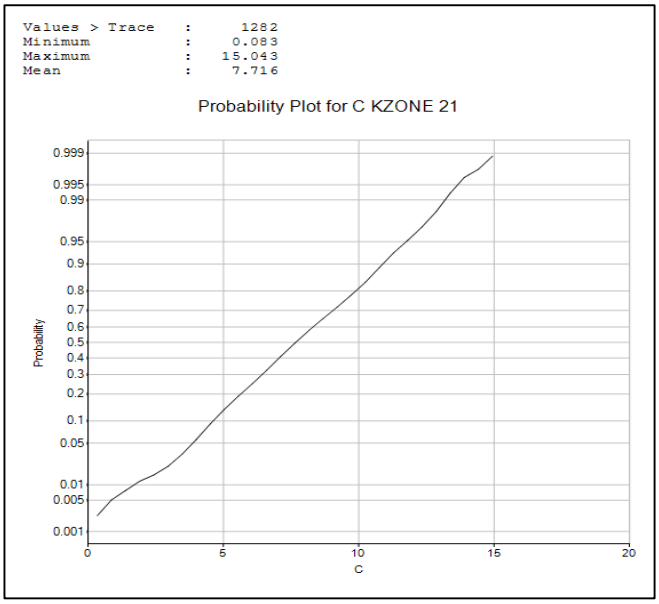
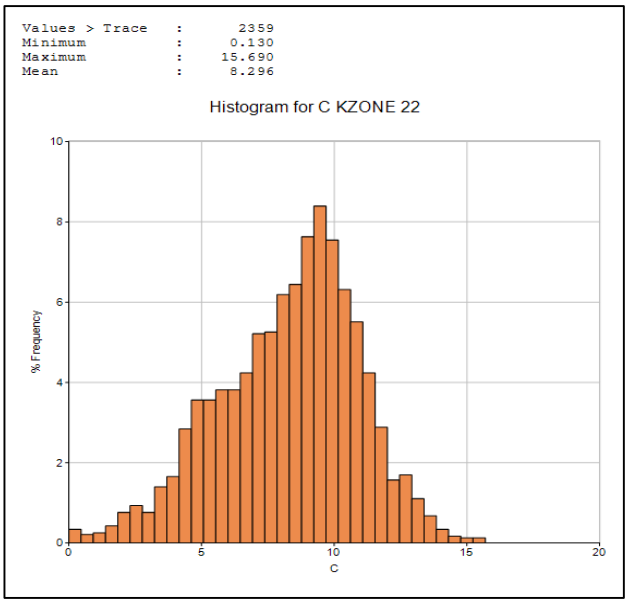


Figure 66: Sample Distributions for the "High Grade East" Domain



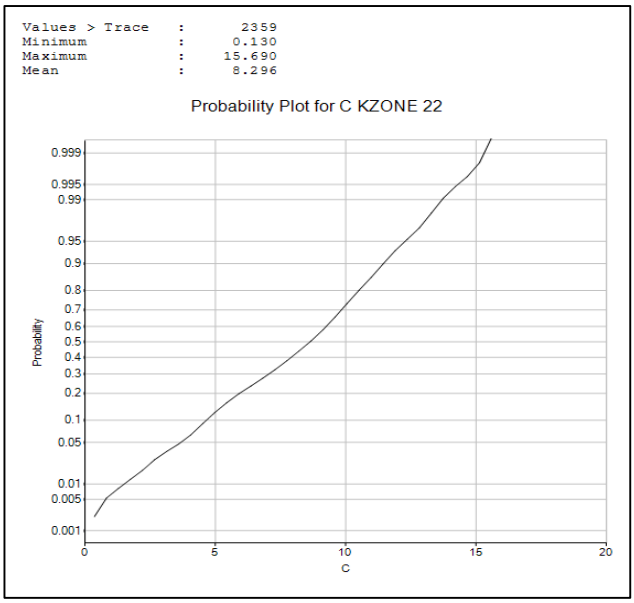
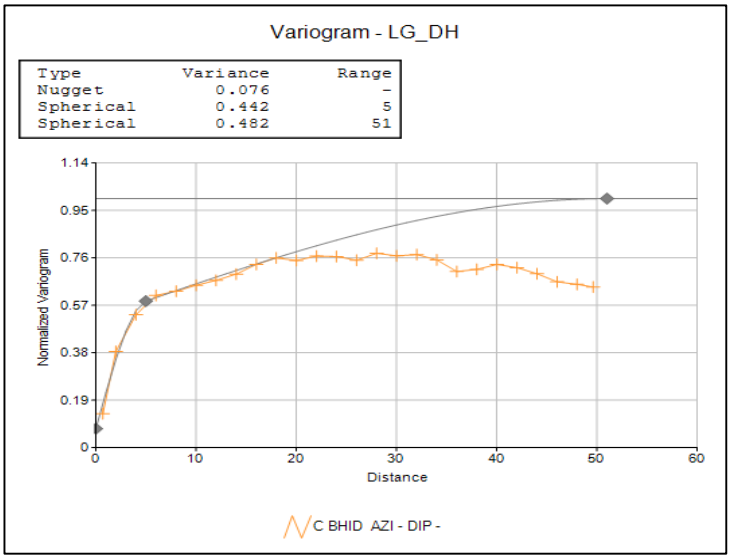


Figure 67: Sample Distributions for the "High Grade West" Domain

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 68, Figure 69 and Figure 70 to nugget for

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



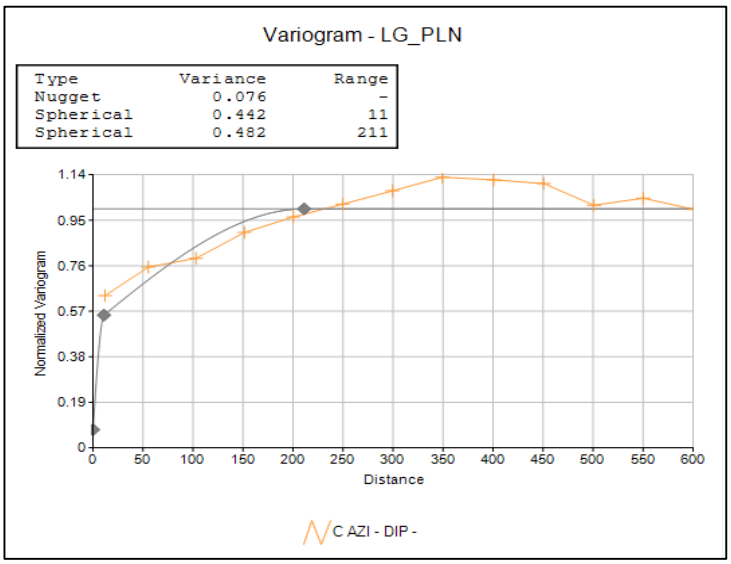
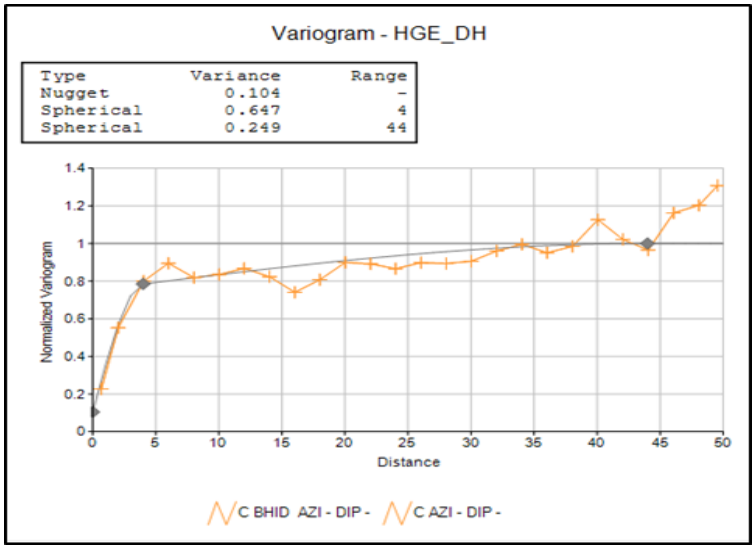


Figure 68: Variogram Model for “Low Grade” Domain – Down Hole and Isotropic



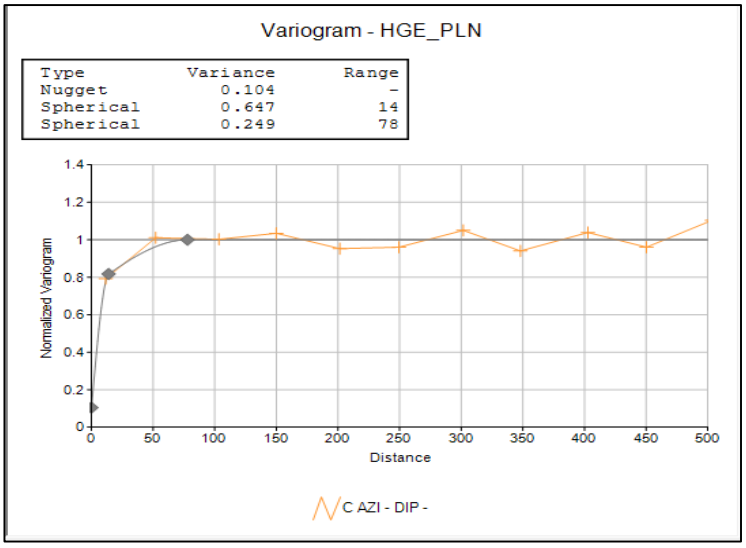
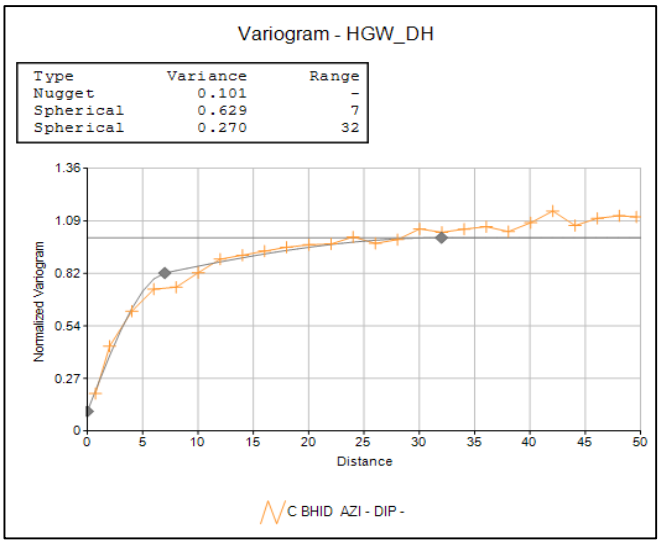


Figure 69: Variogram Model for the "High Grade" East Domain – Down Hole and Isotropic



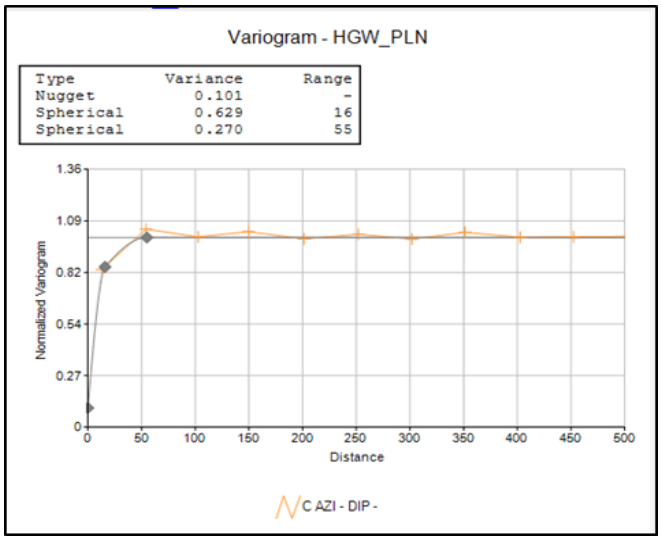


Figure 70: Variogram Model for the "High Grade" West Domain – Down Hole and Isotropic

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 33 below. All samples that were greater than the top capping value were re-set to the top capping value.

Table 33: Summary of Top Capping Values

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

14.6 Grade Estimation

14.6.1 Method

The method of estimations for % C was Ordinary Kriging. Estimations were undertaken using the Estima process in Datamine. Parameters for estimations (i.e. Parent Block size, Search distances and the number of samples to be used for an estimate) 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 71 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

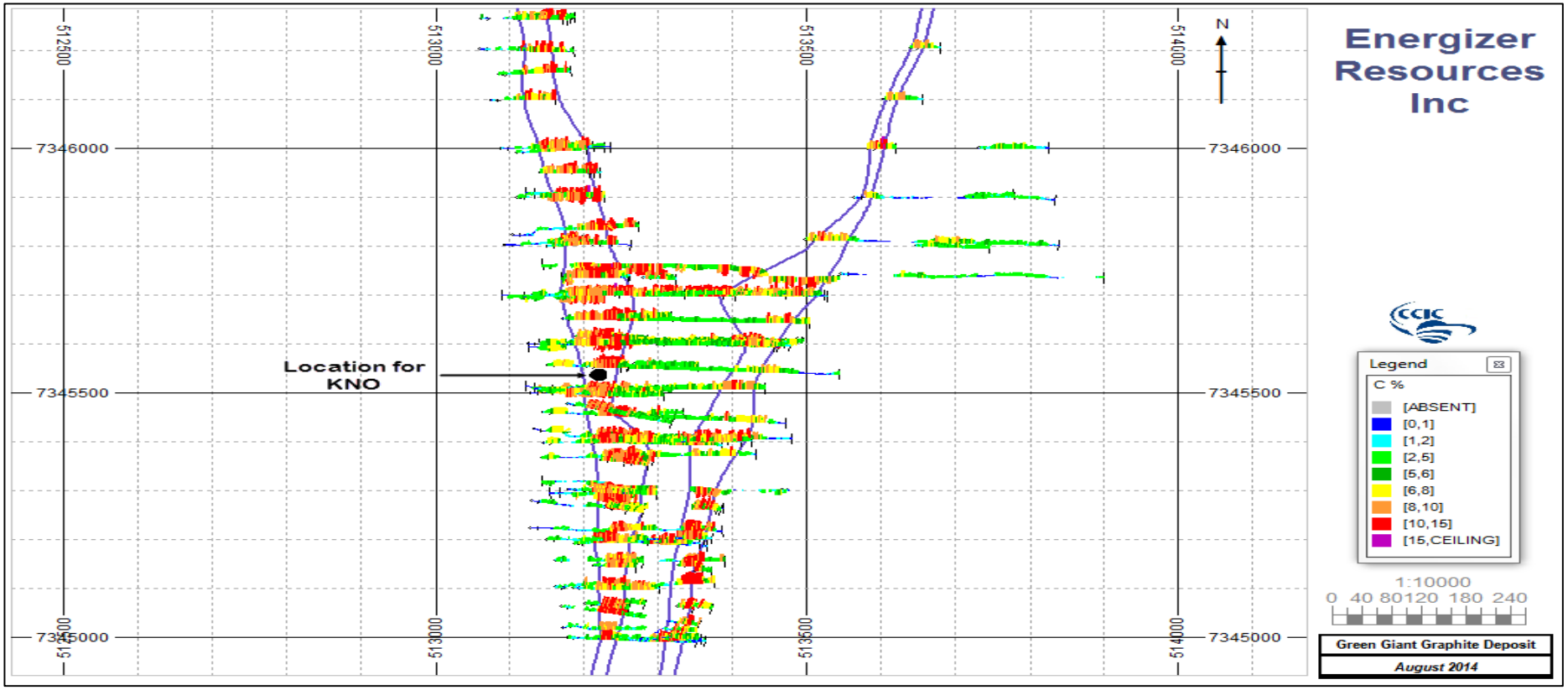


Figure 71: Plan showing the Location for the Krige Optimisations

During parent cell size optimisation, the approach taken was to use the variogram ranges as a guide to set the search distances, and not to apply any minimum and maximum number of samples to be used. A default parent cell size of 10m x 10m x 10m was selected using the borehole spacing as a guide. Whilst all parameters remain constant, the parent cell size in the Y direction was set to 5m and then incrementally increased for 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 ("PCNEGWTH") are recorded for every estimation. These outputs are then plotted against the incremental cell sizes to determine the optimum cell size in the Y direction. The same process is then repeated for the cell sizes in the X direction. Figure 72 compares the plot of the incremental parent cell sizes against the KVAR for the Y direction. The optimum parent cell size, taking practical mining constraints into consideration, was set to 40m x 10m x 10m in the X, Y and Z directions respectively.

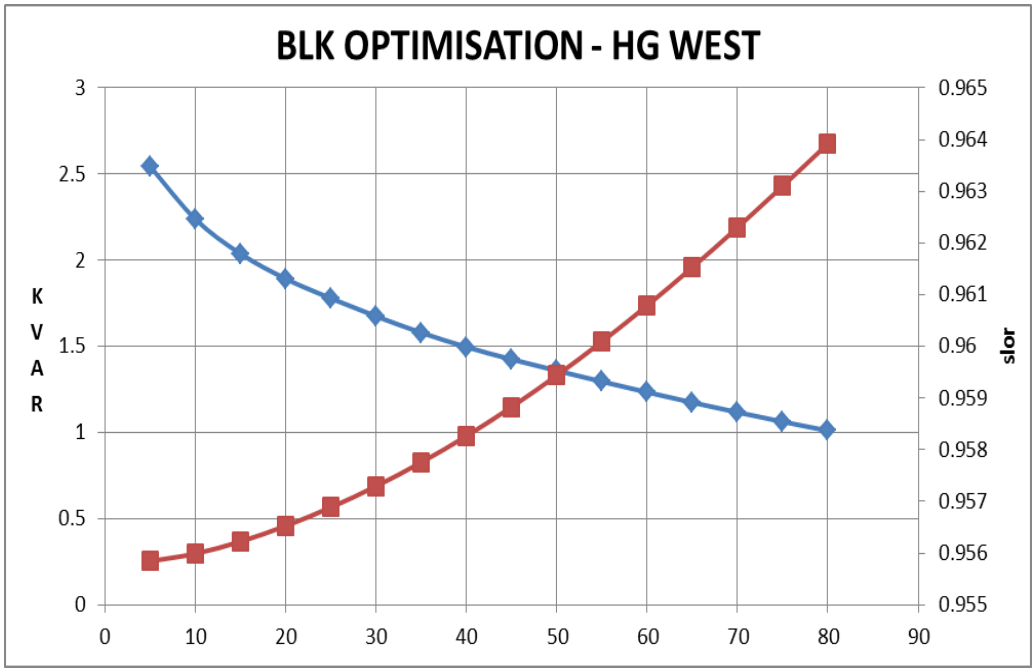
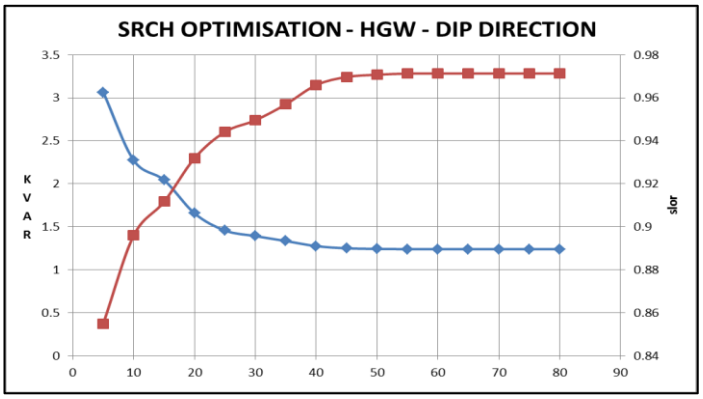


Figure 72: Parent Cell Size Optimisations

During search optimisations, the parent cell size was set to 10m x 40m x 10m and, as described above, search distance in the X direction was incrementally increased. The results of the search optimisation are plotted in Figure 73, which compares both KVAR and SLOR against search distances. Attention is also given the number of samples per estimate and the percentage of samples with negative Kriging weights. The optimum search distance, taking borehole spacing into account, was set as 70m x 40m x10m 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.



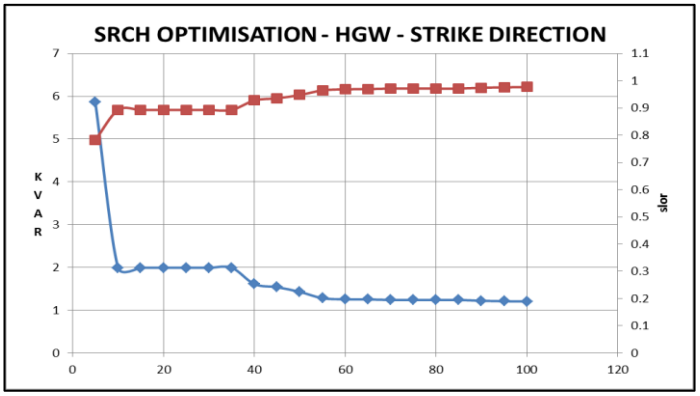


Figure 73: Search Distance Optimisations

14.6.3 Model Construction and Parameters

A block model was constructed using a parent cell size of 10m x 40m x10m 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 74 “Waste” domains have been assigned and estimated for indicative purposes only, and have not been reported in the mineral resource statement.

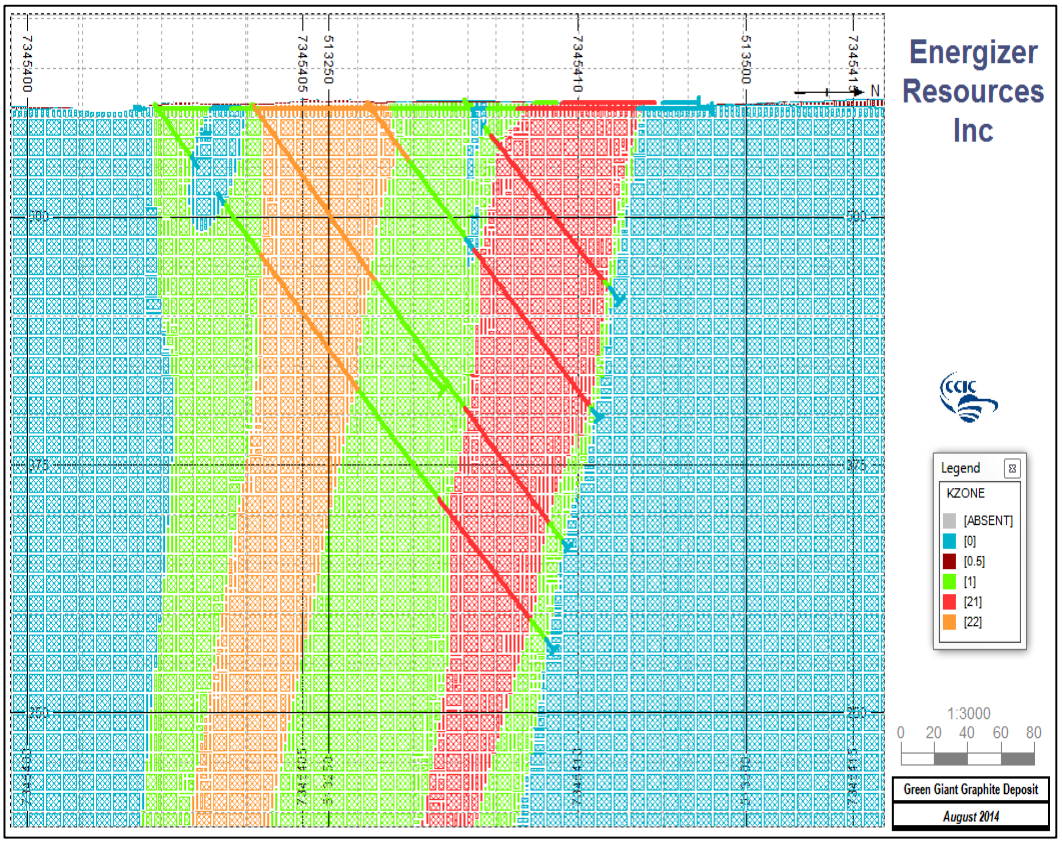


Figure 74: Section Showing Block Model Coding

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 original search ellipse in all three directions (x,y,z). 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 34. The mean of the samples is weighted by length, while the model estimates are weighted by volume. The means between sample and model estimates compares favourably.

Table 34: Statistical Comparisons between the Sample and Model Estimates

	Sample Composites				**Model Estimates**		
Kzone	LG	HG-E	HG-W		LG	HG-E	HG-W
Field	C	C	C		C	C	C
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 x 40m x 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 low and high grade domains are illustrated in Figure 75, Figure 76 and Figure 77. The mean of the estimates are smoother and less variable than that of the samples, while preserving the grade trends of the samples.

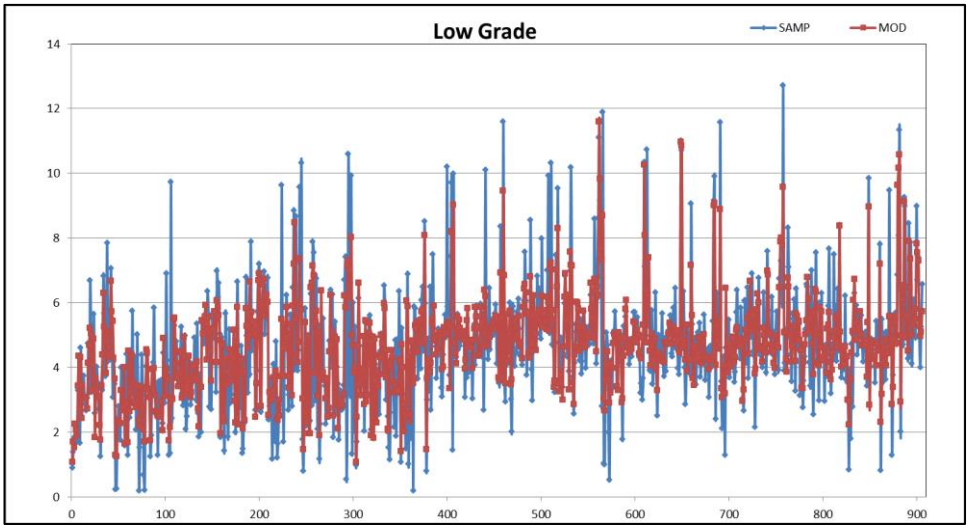


Figure 75: Trend Analysis Plot for % C, "Low" Grade Domain

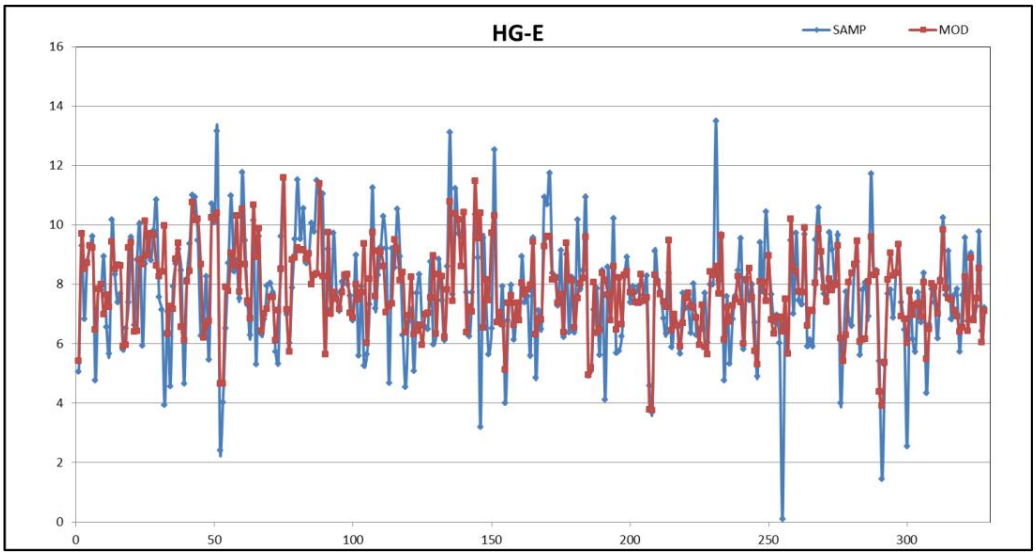


Figure 76: Trend Analysis Plot for % C, "High" Grade East

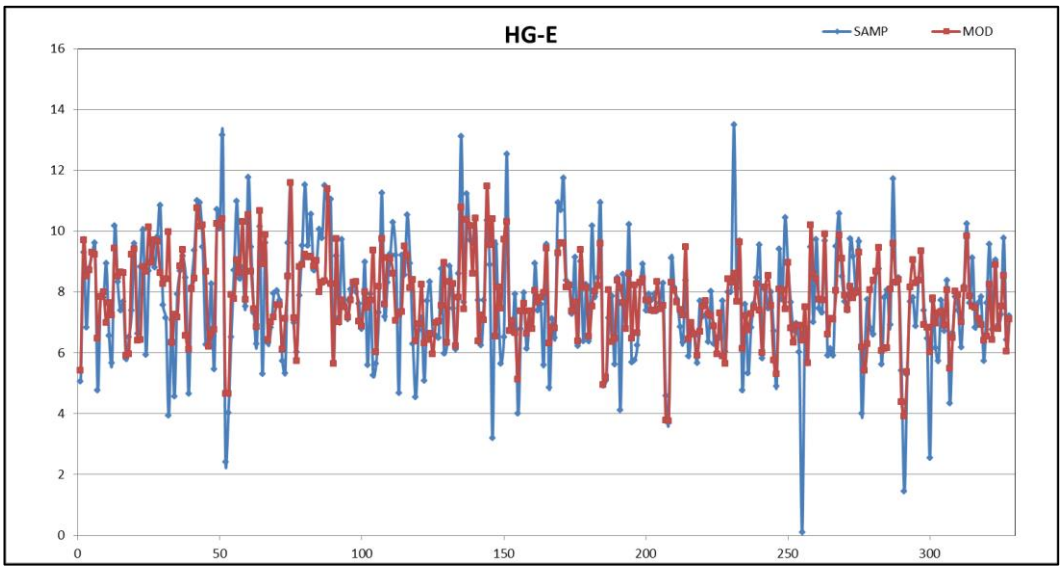


Figure 77: Trend Analysis Plot for % C, "High" Grade West

14.8 Grade Distribution Plots

Figure 78 below 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.

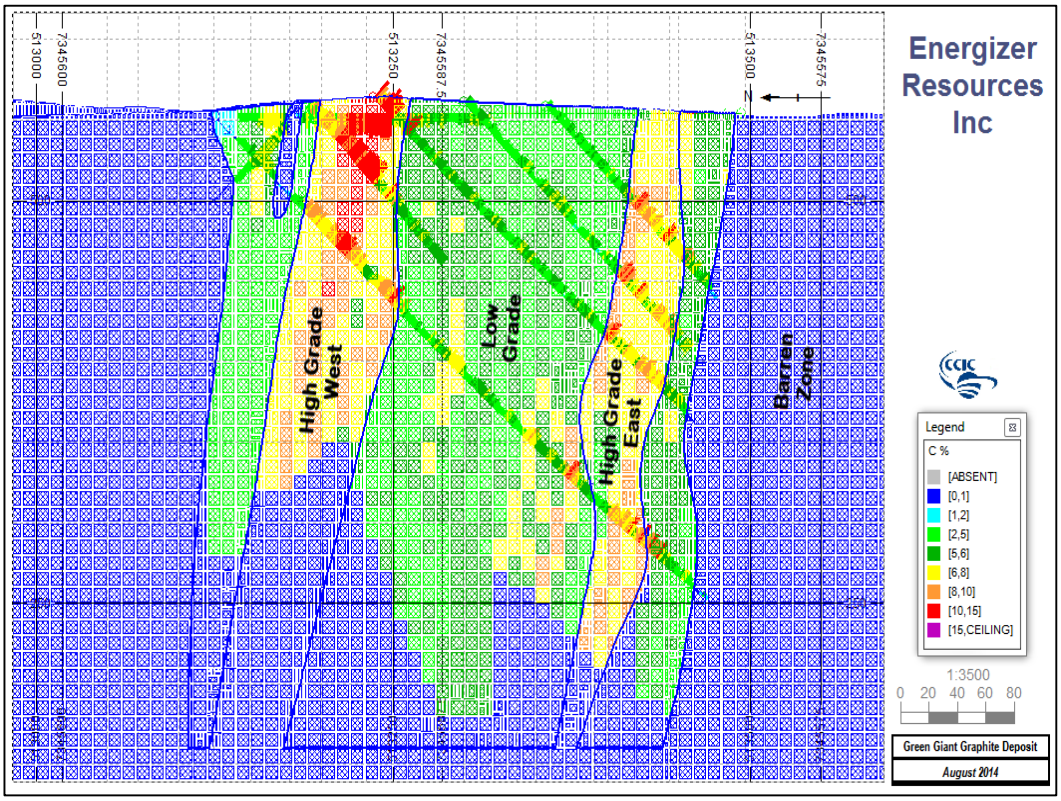


Figure 78: Section showing C (%) Grade Distributions in the Model

The grade tonnage curves for the “low” and “high” grade domains are presented in Figure 79 and Figure 80. 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.

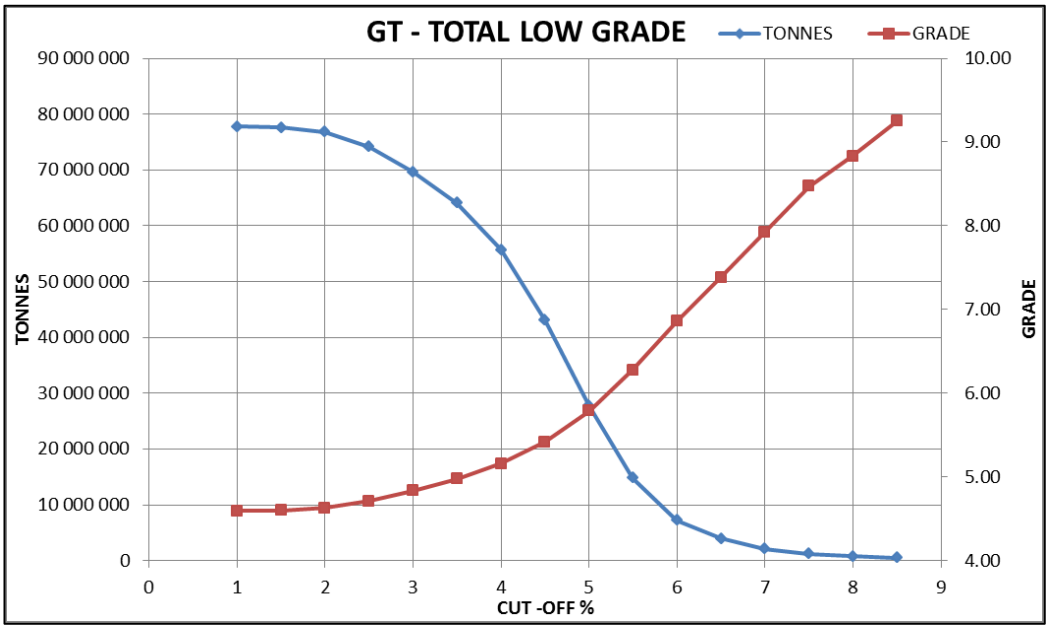


Figure 79: Grade Tonnage Curve for the "Low" Grade Domain

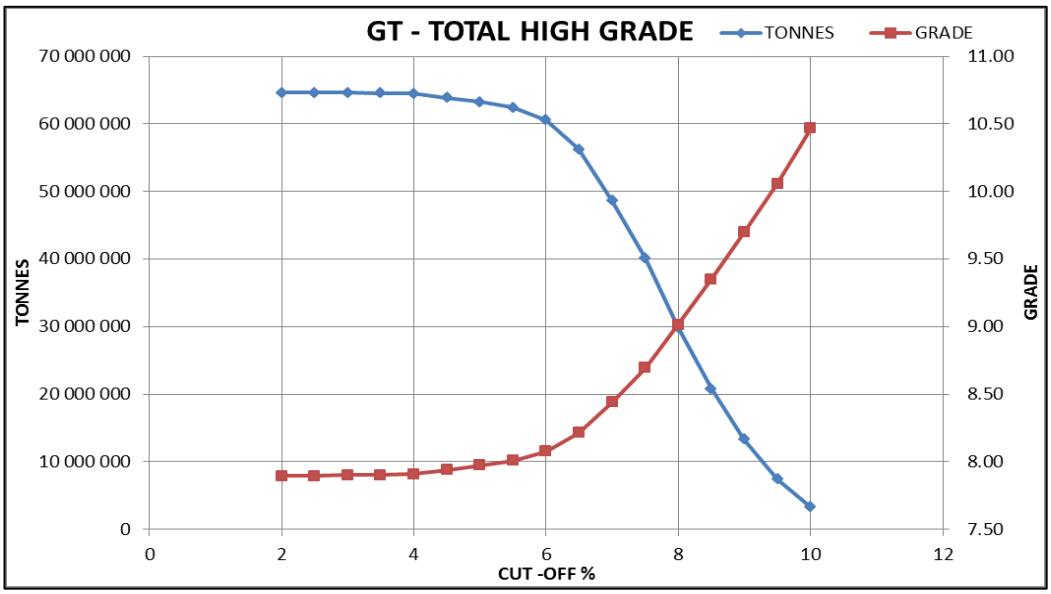


Figure 80: Grade Tonnage Curve for the "High" Grade Domains

14.9 Resource Classification

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

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

Drill hole spacing varies from 100m by 100m in some areas, to 50m by 50m in other areas especially within the first five years of mining footprint. The well informed areas provide adequate geological confidence to place the resources into the Measured Category. Hence, the resource classification methodology Figure 81 and Figure 82) was based on the following criteria:

- Areas with drill spacing less than 50m by 50m, was considered for measured resources
- Areas with drilling of 50m along dip and 10m along strike, was considered for indicated resources
- Areas within the 100m by 100m drill spacing, was considered for inferred resources

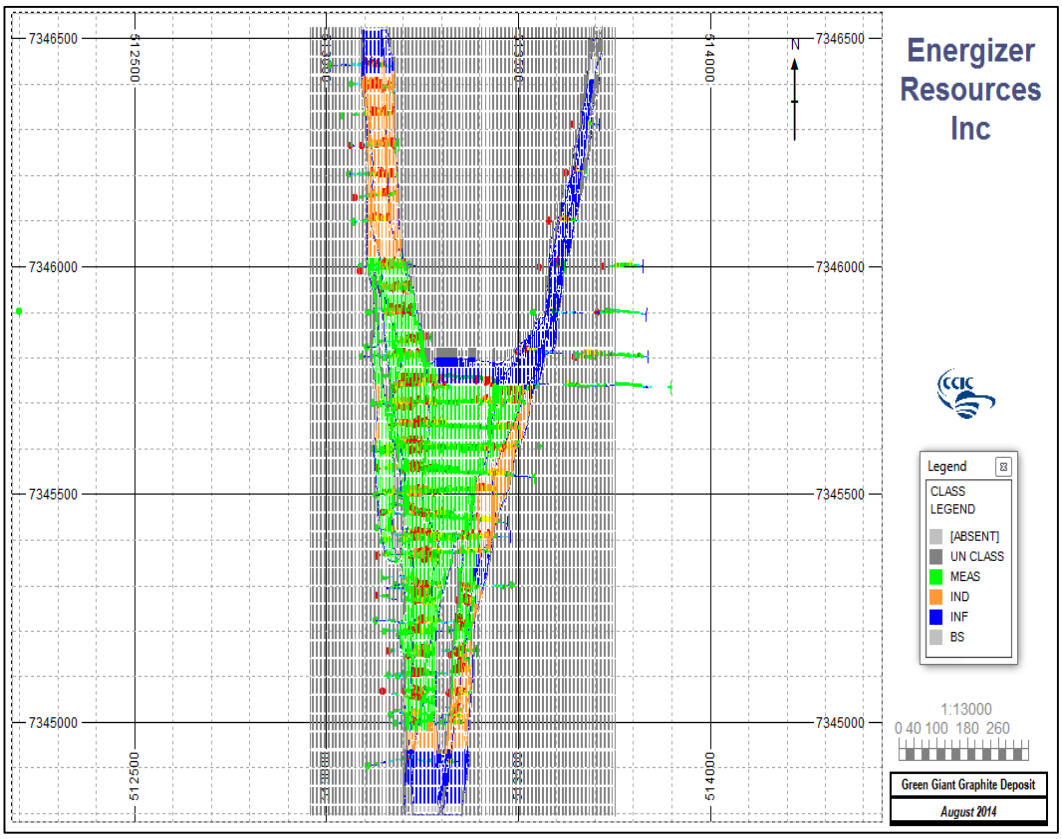


Figure 81: Plan showing Mineral Resource Classification

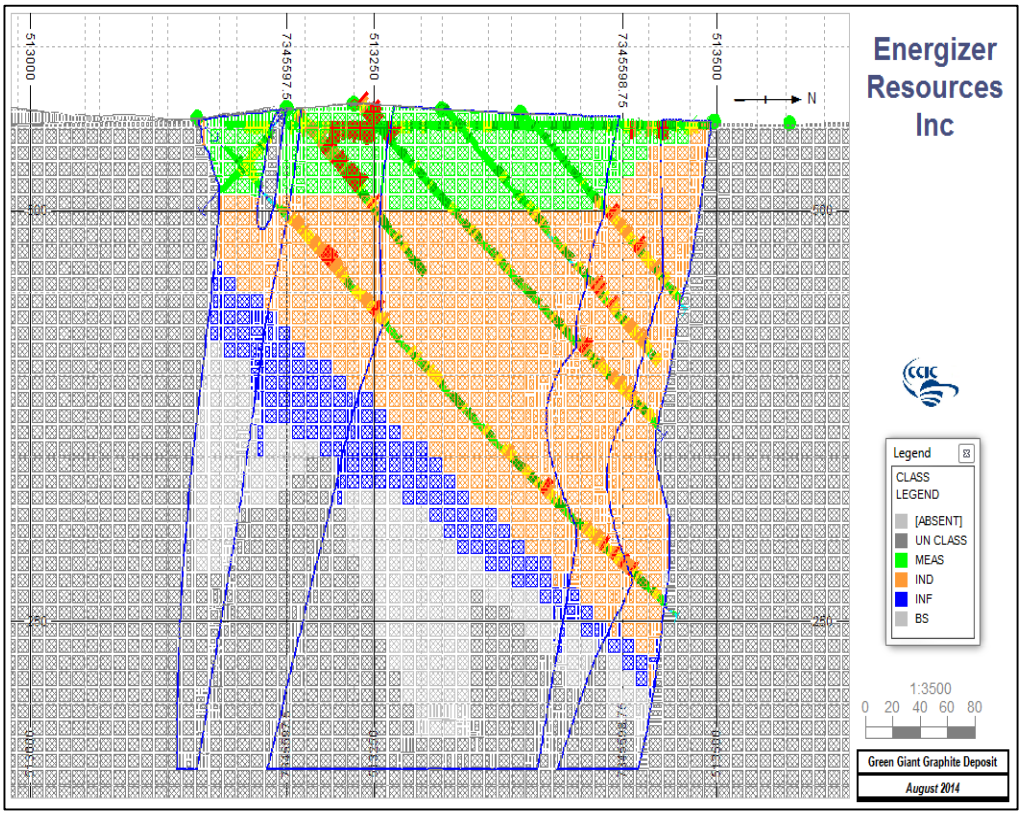


Figure 82: Section showing Mineral Resource Classification

14.10 Resource Tabulation

The current mineral resource estimate for Molo is summarised Table 35. A RD of 2.36 t/m^3 was assigned to the mineralised domains. The mineral resources are classified as Measured, Indicated and Inferred categories according to the CIM Definition Standards. 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. The reason for the increase in tonnage is due to additional exploratory drilling on the north eastern limb of mineralisation. 23.62 Mt, grading at 6.32% C have been upgraded by infill drilling into the Measure Resource category.

Table 35: Mineral Resource Statement for Energizer Resources Molo Graphite Deposit – August 2014

Classification	Material Type	Tonnes - Mt	Grade - C%	Graphite - T
Measured	"Low Grade"	13 048 373	4.64	605 082
Measured	"High Grade"	10 573 137	8.40	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

Mt = million tonnes; C% = carbon percentage; Graphite – T = Tonnes of graphite.

Notes:

1 Mineral Resources are classified according to the CIM definitions.

2 "low grade" Resources are stated at a cut-off grade of 2 % C.

3 "high grade" Resources are stated at a cut-off grade of 4 % C.

4 Eastern and Western high grade assays are capped at 15 % C.

5 A RD of 2.36 t/m^3 was used for the tonnage estimation.

15 MINERAL RESERVE ESTIMATES

15.1 Mineral Resources

15.1.1 Mineral Resources Statements

The Molo resource model is defined by 10m (N-S) by 10m (E-W) by 10m (vertical) blocks that are either waste or ore, each with global densities of 2.65 and 2.36 respectively. The ore blocks are flagged as high grade, or low grade. The low grade mineral resources are reported at a grade cut-off of 2% and the high grade resources are reported at a 4% cut-off. The resource tonnes and grades are reported for the Measured, Indicated and Inferred categories at the above stated cut-off grades.

The geologically updated (Datamine) resource model was issued by Caracle Creek International Consulting Coal's "CCIC" on the 21st of August 2014.

Table 36: CCIC Resource Summary

CCIC				
Classification	**Material Type**	**Tonnes – t**	**Grade – C%**	**Graphite – t**
Measured	Low grade	13,048,373	4.64	605,082
Measured	High grade	10,573,137	8.40	887,835
Measured Total		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
Indicated Total		76,745,953	6.25	4,796,341
Inferred	Low grade	24,233,267	4.46	1,080,677
Inferred	High grade	16,681,453	7.7	1,285,039
Inferred Total		40,914,721	5.78	2,365,716
2% grade cut-off on "Low Grade"				
4% grade cut-off on "High Grade"				

The DRA Mining Division interrogated and validated the model within the Geovia Surpac 6 mining software package. By using the resource model's density, grade and block dimensions, the tonnes are re-calculated and compared to the reported resource estimation tonnes from CCIC.

Table 37: DRAM Resource Estimation

DRAM				
Classification	**Material Type**	**Tonnes - t**	**Grade - C%**	**Graphite - t**
Measured	Low Grade	13,048,373	4.64	605,082
Measured	High Grade	10,573,137	8.40	887,835
Measured – Total		23,621,509	6.32	1,492,916
Indicated	Low Grade	39,539,401	4.73	1,871,075
Indicated	High Grade	37,206,549	7.86	2,925,266
Indicated - Total		76,745,950	6.25	4,796,341
Inferred	Low Grade	24,233,266	4.46	1,080,677
Inferred	High Grade	16,681,453	7.70	1,285,039
Inferred - Total		40,914,719	5.78	2,365,716
2% grade cut-off on “Low Grade”				
4% grade cut-off on “High Grade”				

*Rounding error account for the slight total discrepancies.

The CCIC and the DRAM resource statements have been reconciled as a way of validating the resource model and the comparison is illustrated in the following table:

Table 38: CCIC vs. DRA (Mining) Resource Comparison

CCIC					DRA (Mining)					CCIC vs DRA (Mining) % Difference		
Classification	**Material Type**	**Tonnes – t**	**Grade – C%**	**Graphite – t**	**Classification**	**Material Type**	**Tonnes – t**	**Grade – C%**	**Graphite – t**	**Tons**	**Grade**	**Graphite**
Measured	Low grade	13,048,373	4.64	605,082	Measured	Low grade	13,048,373	4.64	605,082	0.00%	-0.06%	0.00%
Measured	High grade	10,573,137	8.40	887,835	Measured	High grade	10,573,137	8.40	887,835	0.00%	-0.03%	0.00%
Measured Total		23,621,510	6.32	1,492,916	Measured Total		23,621,510	6.32	1,492,916	0.00%	0.00%	0.00%
Indicated	Low grade	39,539,403	4.73	1,871,075	Indicated	Low grade	39,539,403	4.73	1,871,075	0.00%	0.05%	0.00%
Indicated	High grade	37,206,550	7.86	2,925,266	Indicated	High grade	37,206,550	7.86	2,925,266	0.00%	0.03%	0.00%
Indicated Total		76,745,953	6.25	4,796,341	Indicated Total		76,745,953	6.25	4,796,341	0.00%	-0.01%	0.00%
Inferred	Low grade	24,233,267	4.46	1,080,677	Inferred	Low grade	24,233,267	4.46	1,080,677	0.00%	-0.01%	0.00%
Inferred	High grade	16,681,453	7.7	1,285,039	Inferred	High grade	16,681,453	7.7	1,285,039	0.00%	0.04%	0.00%
Inferred Total		40,914,721	5.78	2,365,716	Inferred Total		40,914,721	5.78	2,365,716	0.00%	0.04%	0.00%
2% grade cut-off on Low Grade”							2% grade cut-off on Low Grade					
4% grade cut-off on High Grade”							4% grade cut-off on High Grade					

15.1.2 Resource Grade Tonnes Analysis

The validated resource model has been interrogated by charting the ore tonnes against the corresponding average grade at various cut-off grades. This process assists the mine design and production scheduling activities by aligning them to practical production targets at given grades.

The high grade and low grade tonnes at various grade cut-off's are illustrated in Table 39 below.

Table 39: Resource Tonnes per Cut-off Category

Grade Category	High Grade	Low Grade	High and Low Grade
0>C<=0.02			
0.02>C<=0.03		7 145 132	7 145 132
0.03>C<-0.04		13 952 699	13 952 699
0.04>C<=0.05	1 227 176	27 966 031	29 193 206
0.05>C<=0.06	2 687 216	20 545 565	23 232 781
0.06>C<=0.07	11 906 165	5 086 991	16 993 156
0.07>C<=0.08	18 809 089	1 312 775	20 121 864
0.08>C<=0.09	16 518 814	556 858	17 075 672
0.09>C<=0.10	10 012 366	170 104	10 182 471
0.10>C<=0.11	2 939 386	77 013	3 016 399
0.11>C<=0.12	359 962	7 873	367 835
0.12>C<=0.13	964	-	964
Total	**64 461 138**	**76 821 040**	**141 282 178***

*Rounding errors have resulted in the sight discrepancy in total tonnes.

Additionally, the ore tonnes including the corresponding average grade are reported at various grade cut-offs in grade-tonnes tables and charts.

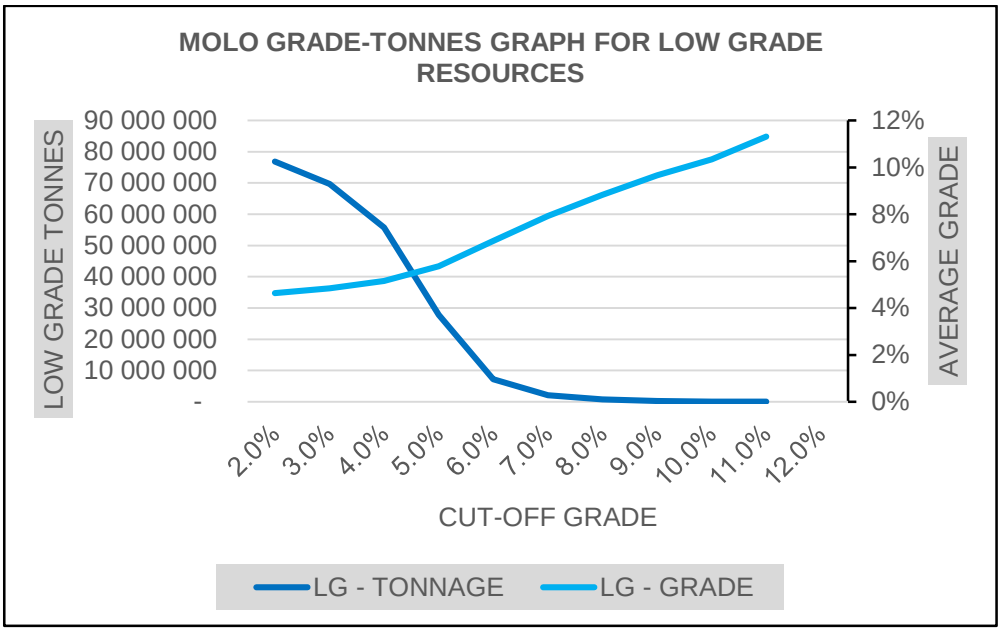


Figure 83: Low Grade - Grade Tonnes Chart

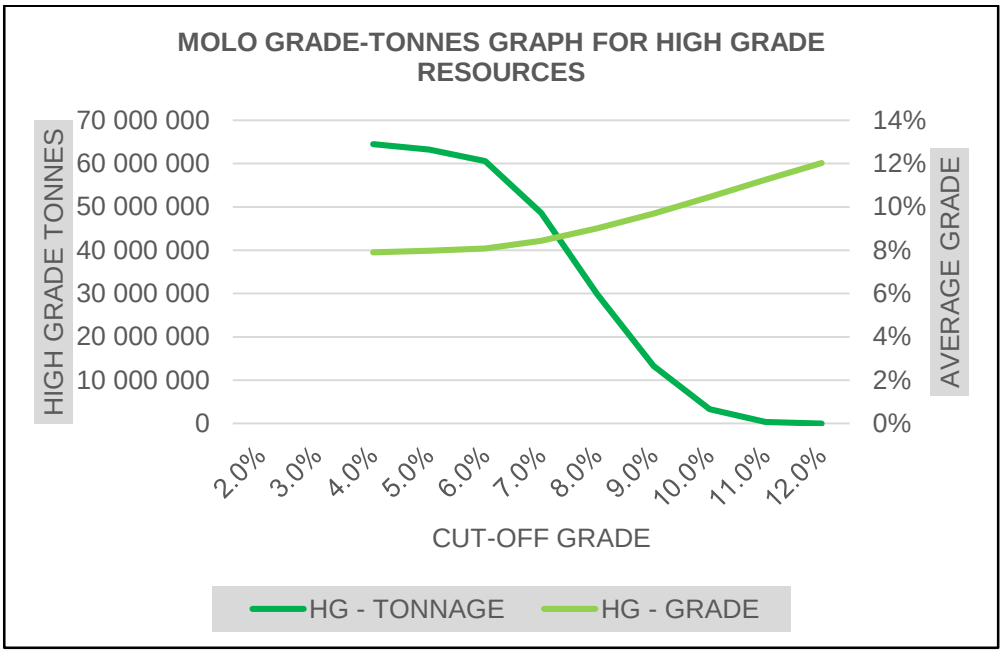


Figure 84: High Grade - Grade Tonnes Chart

15.2 Mineral Reserves

The mineral reserves for the Molo 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 DRA Projects / Energizer 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;

15.2.1 Whittle Optimisation Shell

15.2.2 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.

15.2.2.1 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 (30t) 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.

15.2.2.2 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.

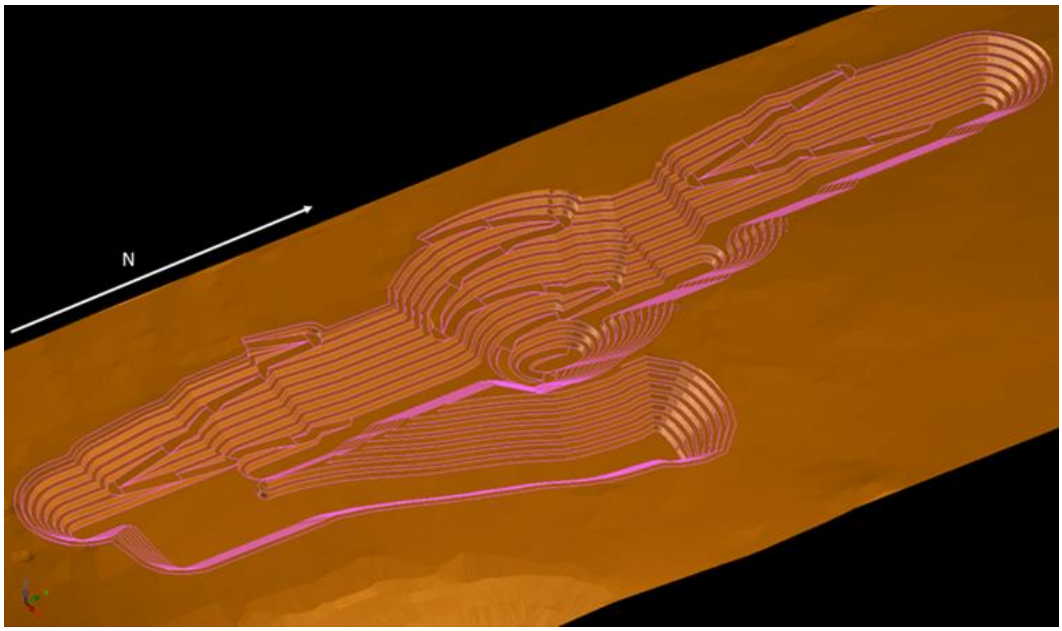


Figure 85: Molo Final Pit Design

15.2.3 Mining Modifying Factors

15.2.3.1 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.

15.2.3.2 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%.

15.2.4 *Measured & 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 reserves 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.

15.2.5 *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 40: 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%				

Table 41: Molo Planned Materials Movement

Year	Period Number	High Grade Tonnes	Low Grade Tonnes	Total Rom Tonnes	Reject Tonnes	Waste Tonnes (Includes Any Inferred)	Topsoil Tonnes	Feed To Plant Grade	Rom Graphite Tons	High Grade Tonnes Grade	Low Grade Tonnes Grade	Reject Tonnes Grade (@ 4.5% Cut-Off)
2017	1	654 017	208 478	862 495	211 588	62 549	77 861	0.072	62 176	0.077	0.056	0.032
2018	2	746 448	116 047	862 495	180 731	51 862	13 543	0.076	65 458	0.080	0.052	0.032
2019	3	801 923	60 572	862 495	131 565	81 444	16 759	0.081	69 630	0.082	0.067	0.033
2020	4	603 710	258 785	862 495	206 494	299 508	937	0.070	60 737	0.080	0.049	0.031
2021	5	603 710	258 785	862 495	148 095	297 878	2 917	0.070	60 327	0.079	0.049	0.028
2022	6	603 710	258 785	862 495	411 255	1 008 667	3 670	0.070	60 311	0.078	0.052	0.029
2023	7	603 710	258 785	862 495	467 691	1 013 833	14 087	0.069	59649	0.076	0.053	0.030
2024	8	605 364	259 494	864 858	219 478	720 423	49 744	0.070	60 598	0.077	0.053	0.033
2025	9	603 710	258 785	862 495	261 369	404 927	944	0.070	60 095	0.077	0.053	0.030
2026	10	603 710	258 785	862 495	212 839	301 901	2 197	0.070	60 641	0.078	0.052	0.031
2027	11	603 710	258 785	862 495	199 225	286 033	35 074	0.069	59 397	0.076	0.052	0.035
2028	12	605 364	259 494	864 858	122 329	254 983	-	0.069	59 277	0.076	0.052	0.034
2029	13	603 710	258 785	862 495	104 173	421 808	4 778	0.069	59 224	0.076	0.052	0.035

Year	Period Number	High Grade Tonnes	Low Grade Tonnes	Total Rom Tonnes	Reject Tonnes	Waste Tonnes (Includes Any Inferred)	Topsoil Tonnes	Feed To Plant Grade	Rom Graphite Tons	High Grade Tonnes Grade	Low Grade Tonnes Grade	Reject Tonnes Grade (@ 4.5% Cut-Off)
2030	14	603 710	258 785	862 495	278 539	495 795	31 586	0.069	59 433	0.076	0.052	0.031
2031	15	603 710	258 785	862 495	384 539	719 332	-	0.070	60 120	0.077	0.052	0.031
2032	16	605 364	259 494	864 858	172 473	522 222	782	0.069	60 005	0.077	0.052	0.033
2033	17	603 710	258 785	862 495	321 273	500 416	-	0.070	60 122	0.077	0.052	0.029
2034	18	603 710	258 785	862 495	313 803	372 577	-	0.070	60 064	0.077	0.052	0.029
2035	19	603 710	258 785	862 495	208 600	485 400	1 474	0.069	59 221	0.076	0.052	0.031
2036	20	605 364	259 494	864 858	164 668	415 326	11 097	0.069	59 390	0.076	0.052	0.034
2037	21	603 710	258 785	862 495	127 084	775 289	-	0.069	59 255	0.076	0.052	0.034
2038	22	603 710	258 785	862 495	134 351	801 045	-	0.069	59 413	0.076	0.052	0.034
2039	23	603 710	258 785	862 495	122 349	515 523	-	0.069	59 942	0.077	0.052	0.036
2040	24	605 364	259 494	864 858	-	-	-	0.069	60 106	0.077	0.052	
2041	25	603 710	258 785	862 495	-	-	-	0.069	59 942	0.077	0.052	
2042	26	603 710	258 785	862 495	-	-	-	0.069	59 942	0.077	0.052	
TOTAL		16 095 988	6 340 697	22 436 685	5 104 511	10 808 740	267 452	0.0702	1 574 456			

15.3 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 30t articulated dump trucks, a 6 m^3 excavator and a 8 m^3 Front End Loader.

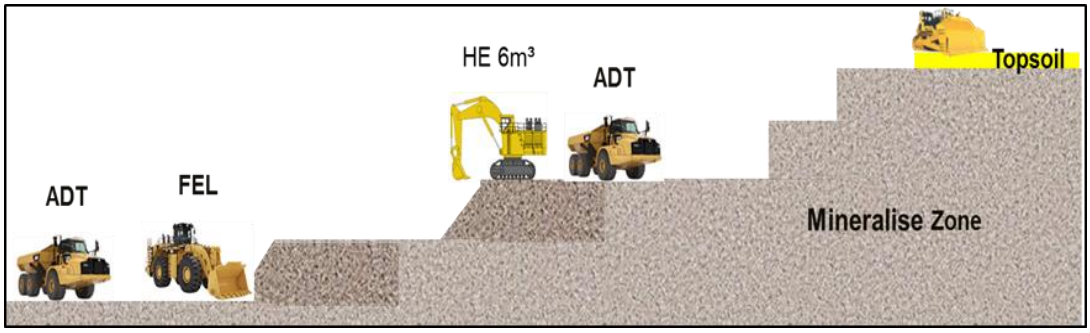


Figure 86: Molo Open Pit Mining Bench Layout

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 Mineral Reserves

16.1.1 Background

The mineral reserves for the Molo 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 DRA Projects / Energizer

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 Measured and Indicated resources *only* in order to report the Molo reserves. These economic and technical parameters are explained and illustrated in the ensuing Whittle optimisation section of this report.

The Molo resource model prepared by CCIC is comprised of 10 meter (east-west) by 10 meter (north-south) and 10 meter (vertical) blocks for both ore and waste. See section 16 for more details.

16.1.2 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 a number of 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 42: 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	$4.78	USD
Mining re-handle cost	$0.36	USD
Processing		
Plant recovery	87	%
Mining re-handle cost	$0.36	USD
Processing unit cost	$17.78, or $ 19.67	USD
G & A cost	$ 6.02	USD
Mill Throughput	862	Ktpa
Product/Selling		
Product price	$1,340.04, or $1,317.70	USD
Royalty	0%	%
Selling cost	$371.00	USD

16.1.3 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 46º could be adopted although the slope angles in the fresh and hard rock material could be made steeper, up to 48º.

16.1.4 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.

16.1.5 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%.

16.1.6 Plant Recovery & Costs

The plant recovery of 87% 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 either USD$1340.04, or USD$1317.70 depending on the processing plant recovery of 90% and 82% respectively. The lower plant recovery attempts to maximise the low grade ore which is expected to produce larger flake sized product.

16.2 Whittle Scenarios

Four Whittle scenarios were considered in order to select the best optimisation approach regarding the following parameters;

Mining cost	Owner mining cost versus Leased equipment
Processing cost	87.8% plant recovery versus 82% recovery
Plant recovery	87.8% maximising high grade ore and 82% maximising low grade ore

Table 43: Whittle Input Parameters per Scenario

	Scenario 1	**Scenario 2**	**Scenario 3**	**Scenario 4**
	Owner @ 90% Plant Recovery	**Owner @ 82% Plant Recovery**	**Leased Equipment @ 90% Plant Recovery**	**Leased Equipment @ 82% Plant Recovery**
Whittle Input Parameters				
Mining Cost	4.13	4.13	4.78	4.78
Mining Recovery	95.00%	95.00%	95.00%	95.00%
Dilution	3.00%	3.00%	3.00%	3.00%
Processing Cost	19.67	17.78	19.67	17.78
Ore Handling Cost	0.25	0.25	0.36	0.36
G&A Cost	6.02	6.02	6.02	6.02
Total Process Cost	25.94	24.05	26.05	24.16
Plant Recovery	87.8%	82%	87.8%	82%
Selling Cost	371	371	371	371

16.3 Whittle Results

The Whittle optimal results for each scenario are tabulated below and scenario 1 was selected as the go-forward case for the design of the final pit.

Table 44: Whittle Results Summary per Scenario

	Scenario 1	**Scenario 2**	**Scenario 3**	**Scenario 4**
	Owner @ 90% Plant Recovery	**Owner @ 82% Plant Recovery**	**Leased Equipment @ 90% Plant Recovery**	**Leased Equipment @ 82% Plant Recovery**
Ore	16 343 888	17 807 612	15 794 049	17 956 875
Waste	8 932 324	9 559 842	8 142 408	9 606 400
Total	25 276 212	27 367 454	23 936 457	27 563 275
Strip Ratio	0.55	0.54	0.52	0.53
Graphite Feed To Plant	1 291 116	1 389 530	1 248 534	1 399 371
Graphite Grade To Plant	7.9	7.8	7.9	7.8
Product Recovery Of Plant	1 133 600	1 139 414	1 096 213	1 147 485
Pit Value After Capital NPV @ 10%	237 224 779	233 863 654	226 199 099	219 561 730
LOM	18.95	20.65	18.31	20.82

The Whittle optimal pit shell for scenario 1 is 1.45 km long (N-S), 443m wide (east-west) and 119m deep. The optimisation runs in Whittle do not include the minimum mining width parameter and therefore, inclusion of this parameter in the final pit design is likely to add more waste to the ultimate results.

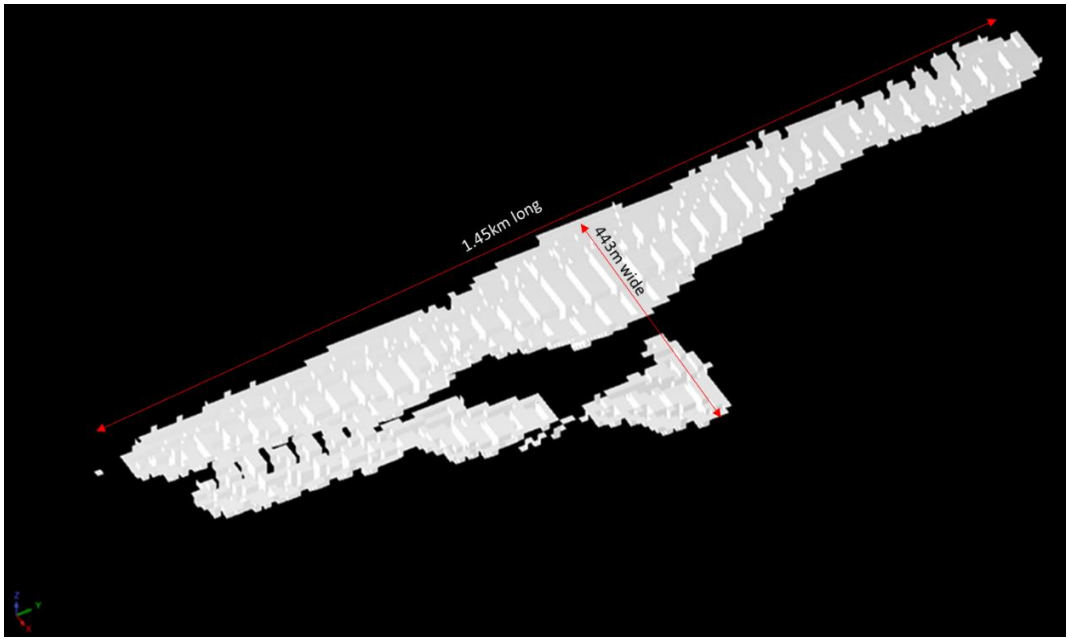


Figure 87: Whittle Optimal Pit Shell

16.4 Open Pit Final Design

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

16.4.1 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 (30t) articulated haul trucks at a width of 3m 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 180º turn with a radius of 15m.

16.4.2 Bench Design

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

16.4.3 Pushback

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.

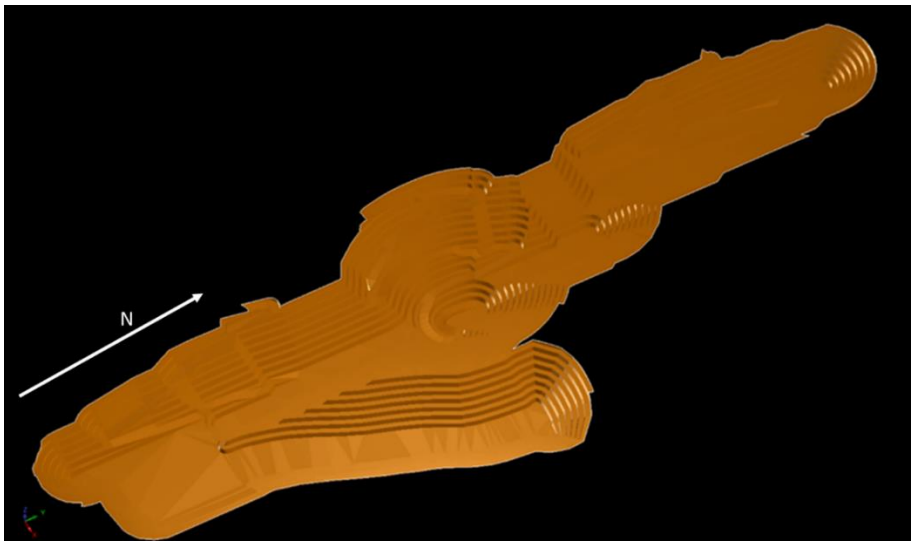


Figure 88: Final Design Pit

16.5 Mine Planning

The final pit mine plan is aimed at maximising the feed grade whilst minimising the strip ratio in the first 5 years of the production. A pushback has thus been designed and scheduled for year 5. Mining operations are planned to commence in the central and wider portion of the pit, later progressing to the narrow areas of the pit in the north and the south portions of the pit.

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 70% to 30% and the average head grade is 7%. 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 87% and 90 % of the graphite. Depending on the ROM feed rate, feed grade and process recovery efficiency in the processing plant, graphite products from 52 ktpa to 62 ktpa can be produced.

16.5.1 Production Schedule

The final pit production schedule is based on a run of mine production target of 862, 500t per annum. To sustain this production rate an average waste to ROM stripping ratio of 0.81 waste tonnes per 1 ROM tonne will be required over the LOM. The material movement in the production schedule is comprised of the high grade ore stockpile, low grade ore stockpile, reject ore stockpile and waste dumps.

The 30 year production schedule is depicted in Table 45 below:

Table 45: Molo Production Schedule for the LOM

Year	Period Number	High Grade Tonnes	Low Grade Tonnes	Total Rom Tonnes	Reject Tonnes	Waste Tonnes (Includes Any Inferred)	Topsoil Tonnes	Feed To Plant Grade	Rom Graphite Tons	High Grade Tonnes Grade	Low Grade Tonnes Grade	Reject Tonnes Grade (@ 3.0% Cut-Off)
2017	1	654 017	208 478	862 495	211 588	62 549	77 861	0.072	62 176	0.077	0.056	0.032
2018	2	746 448	116 047	862 495	180 731	51 862	13 543	0.076	65 458	0.080	0.052	0.032
2019	3	801 923	60 572	862 495	131 565	81 444	16 759	0.081	69 630	0.082	0.067	0.033
2020	4	603 710	258 785	862 495	206 494	299 508	937	0.070	60 737	0.080	0.049	0.031
2021	5	603 710	258 785	862 495	148 095	297 878	2 917	0.070	60 327	0.079	0.049	0.028
2022	6	603 710	258 785	862 495	411 255	1 008 667	3 670	0.070	60 311	0.078	0.052	0.029
2023	7	603 710	258 785	862 495	467 691	1 013 833	14 087	0.069	59 649	0.076	0.053	0.030
2024	8	605 364	259 494	864 858	219 478	720 423	49 744	0.070	60 589	0.077	0.053	0.033
2025	9	603 710	258 785	862 495	261 369	404 927	944	0.070	60 095	0.077	0.053	0.030
2026	10	603 710	258 785	862 495	212 839	301 901	2 197	0.070	60 641	0.078	0.052	0.031
2027	11	603 710	258 785	862 495	199 225	286 033	35 074	0.069	59 397	0.076	0.052	0.035
2028	12	605 364	259 494	864 858	122 329	254 983	-	0.069	59 277	0.076	0.052	0.034
2029	13	603 710	258 785	862 495	104 173	421 808	4 778	0.069	59 224	0.076	0.052	0.035
2030	14	603 710	258 785	862 495	278 539	495 795	31 586	0.069	59 433	0.076	0.052	0.031

Year	Period Number	High Grade Tonnes	Low Grade Tonnes	Total Rom Tonnes	Reject Tonnes	Waste Tonnes (Includes Any Inferred)	Topsoil Tonnes	Feed To Plant Grade	Rom Graphite Tons	High Grade Tonnes Grade	Low Grade Tonnes Grade	Reject Tonnes Grade (@ 3.0% Cut-Off)
2031	15	603 710	258 785	862 495	384 539	719 332	-	0.070	60 120	0.077	0.052	0.031
2032	16	605 364	259 494	864 858	172 473	522 222	782	0.069	60 005	0.077	0.052	0.033
2033	17	603 710	258 785	862 495	321 273	500 416	-	0.070	60 122	0.077	0.052	0.029
2034	18	603 710	258 785	862 495	313 803	372 577	-	0.070	60 054	0.077	0.052	0.029
2035	19	603 710	258 785	862 495	208 600	485 400	1 474	0.069	59 221	0.076	0.052	0.031
2036	20	605 364	259 494	864 858	164 668	415 326	11 097	0.069	59 390	0.076	0.052	0.034
2037	21	603 710	258 785	862 495	127 084	775 289	-	0.069	59 255	0.076	0.052	0.034
2038	22	603 710	258 785	862 495	134 351	801 045	-	0.069	59 413	0.076	0.052	0.034
2039	23	603 710	258 785	862 495	122 349	515 523	-	0.069	59 942	0.077	0.052	0.036
2040	24	605 364	259 494	864 858	-	-	-	0.069	60 106	0.077	0.052	
2041	25	603 710	258 785	862 495	-	-	-	0.069	59 942	0.077	0.052	
2042	26	603 710	258 785	862 495	-	-	-	0.069	59 942	0.077	0.052	
TOTAL		**16 095 988**	**6 340 697**	**22 436 685**	**5 104 511**	**10 808 740**	**267 452**	**0.0702**	**1 574 456**			

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.

16.5.2 Production Schedule Results

The following graphs and figures summarise the production schedule results:

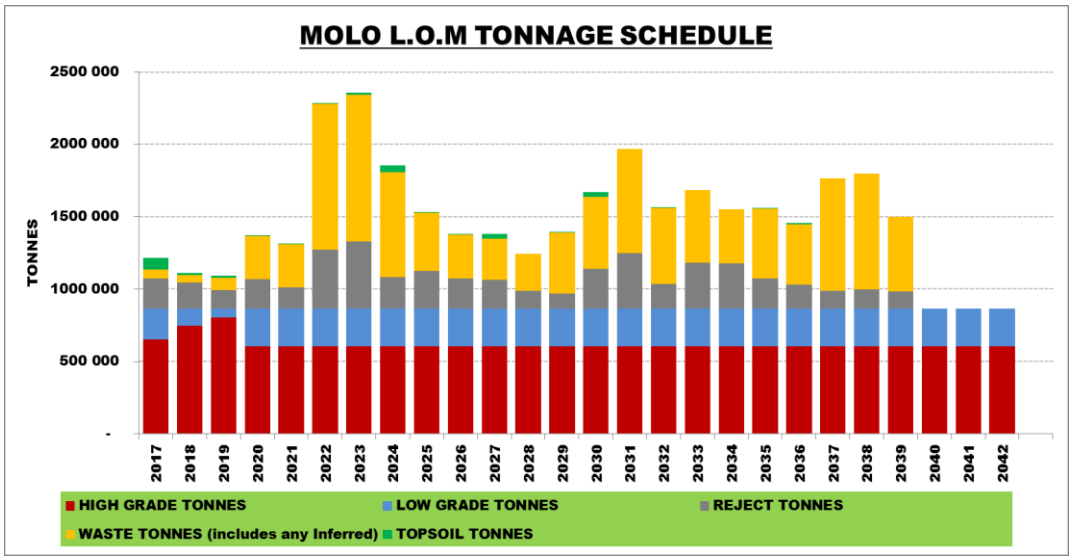


Figure 89: Molo Annual Production Schedule for the LOM

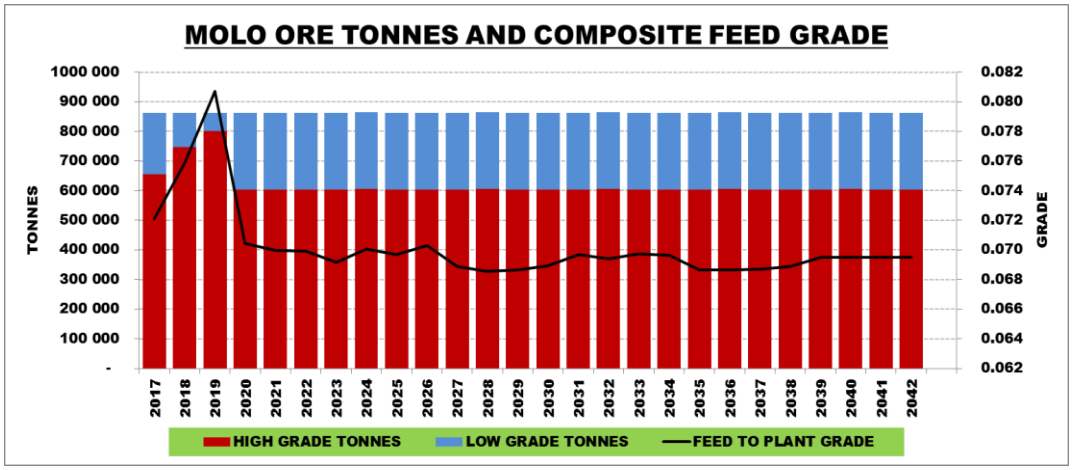


Figure 90: Molo Mill Feed Schedule for the LOM

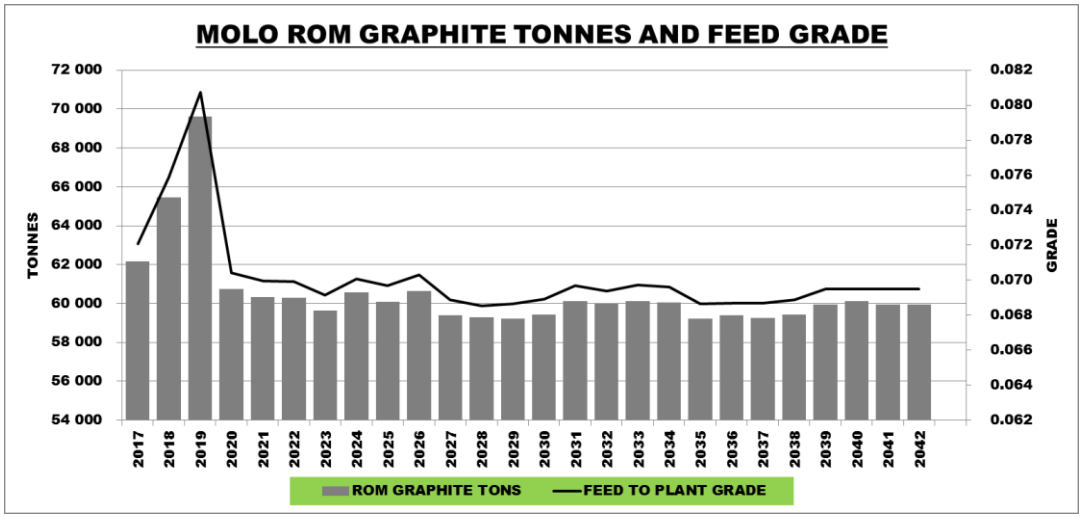


Figure 91: Molo Graphite Content in Feed to Mill

16.6 Mining Operations

16.6.1 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 46: Mining Production and Support Equipment

Production and Support Equipment	No. of Units
Production Equipment	
Excavator – 5 m^3	1
Diesel FEL 6 m^3	1
30 Tonne Renault Tipper Truck	6
Dozer	1
20 000l Water Truck	1
Support Equipment	
Truck Mounted Drill Rig	1
Large Motor Grader	1

Compactor (Vibratory)	1
Diesel Bowser (5,000 litre)	1
Pump (50 m^3/hr 100m head)	2
Skid Steer Loader	1
Backhoe Loader	1
Mechanical / Electrician's Truck (7t)	1
Flatbed Truck (7t)	1
LDV Supervision	2
Crew Vehicle (Double Cab)	2
Admin Vehicle (Double Cab / SUV)	5
Minibus (12 Seats)	2
Total	**32**

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 in excess of 3 Mtpa.

One truck mounted drill rig was selected to drill the designed 110 to 130 mm blast holes and will be capable of drilling, (8 hours per day 5 day a week) approximately 41,600 metres per annum at a penetration rate of 20 metres per hour.

The initial haulage distances for the articulated dump trucks are expected to be approximately 1.0 kilometre, from the open pit-pit to the ROM tip, or the ROM stockpile areas that feed the process plant.

16.6.2 Labour

The number of Expats to be used in the Molo Project has been limited to as few as possible. This is due to the mining being relatively simple, and the productivity levels required being lower than normal.

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

Table 47: Mining Labour Compliment Fixed Cost per Annum

Energizer Staff List: Mining			
Personnel Complement Breakdown	**Responsibility**	**Grade**	**No.**
Management			
Mine Manager	Management	D4	1
Administrator	Management	B4	1
Management			2
Safety & Training			
Safety Officer	Safety & Training	C4	0
Training Officer	Safety & Training	C4	0
Safety & Training (Include in Overheads)			0
Technical Services			
Tech Manager	Technical Services	D3	0
Surveyor	Technical Services	C4	1
Geologist	Technical Services	D2	1
Mine Planner	Technical Services	C4	1
Grade/Surveyor Ass	Technical Services	A3	2
Grade Controller	Technical Services	B3	1
Technical Services			6
Mining			
Drill & Blast Foreman	Mining	C4	1
Mine Overseer	Mining	D2	1
Miner Blaster	Mining	A2	1
Production Foreman	Mining	C4	2
Blasting Assistants	Mining	A2	5
Pumping / Cleaning	Mining	A2	9
Excavator Operator	Mining	B3	2
FEL Operator	Mining	B3	2
ADT Operator	Mining	B3	6
Drill Operator	Mining	B3	2
Dozer Operator	Mining	B3	1
Truck W/E Operator	Mining	B3	2
Backhoe Operator	Mining	B3	1
Comp / Grader Operator	Mining	B3	2
Relief Operator	Mining	B3	5

Energizer Staff List: Mining			
Personnel Complement Breakdown	**Responsibility**	**Grade**	**No.**
Mining			42
Engineering			
Engineering Planner	Engineering	C4	1
Engineering GES	Engineering	D1	1
Diesel Mechanic	Engineering	C2	6
Electrician	Engineering	C2	0
Auto Electrician	Engineering	C2	2
Boilermaker	Engineering	C1	2
Diesel Assistant	Engineering	A3	6
Elect Assistant	Engineering	A3	0
Auto Elect Assistant	Engineering	A3	1
Boiler Assistant	Engineering	A3	2
Engineering			21
Total Mining Staff			**71**

16.6.3 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 = 80 to 120 MPa

The drill and blast products assumed are:

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

The drill and blast parameters:

- Hole diameter = 102 to 150 mm
- Bench height = 8 m

- Stemming length = 1.5 m
- Burden = 2.9m
- Spacing =3.0 m
- Spacing offset =1.5 m
- Sub drill = 0.5 m
- Explosive density = 0.9 g/cm^3
- Powder factor = 0.68
- Charge mass = 30 kg to 35 kg per hole

16.6.4 Mine Infrastructure

Provision has been made at the plant site for a mining office which will accommodate 22 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 and canteen will be shared facilities 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
- Canteen (shared)
- Change House including dressing room, laundry and boiler room (shared)
- Ablution facilities

16.6.5 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.

An average consumption of 300 tonnes of explosives per annum is expected.

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 Project 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 87.8% C(t) to final concentrate, at the required grade, from an average plant feed head grade of 7.04% C(t). This will produce an estimated average of 53 kilotonnes per annum (ktpa) of final concentrate over the LOM, within a range of 48 ktpa and 61 ktpa, depending on feed head grade fluctuations and at the design throughput tonnage of 862.5 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 Design Criteria

As noted above, the process design is based on an annual production capacity of 862.5 kilotonnes of plant feed material at a nominal head grade of 7.04% C(t). The results of various test work programs conducted during the course of the study form the basis of the process design criteria. The list below indicates the sources of information used to compile the design criteria.

Information source codes

- A Assumed, based on current information
- B Previously reported
- C As instructed or determined by the Client/Client Representative
- D1 Selected, based on test work results
- D2 Selected, based on design experience
- D3 Calculated, based on other inputs
- D4 Selected, based on OEM recommendations

Table 48 below presents the mineralized plant feed material characteristics and expected metallurgical performance.

Table 48: Plant Mineralized Material Characteristics

Criteria	Units	Value		Source/Responsibility
		Expected/Avg	Design	
Solids Density	$t.m^{-3}$	2.2 - 2.4	2.4	C
Bulk Density (broken rock)	$t.m^{-3}$		1.65	D2
Plant Feed Head Grade	% C(t)	6.8 -8.7	7.04	D1
Plant Treatment Capacity	ktpa		862.5	C
Recovery to Final Concentrate	%C(t)		87.8	C
Final Concentrate Grade	%C(t)		97.3	C
Final Concentrate Production	ktpa	53	61	D3

Table 49 below indicates the plant's nominal throughputs and operating schedule, and forms the basis of the major equipment sizing.

Table 49: Plant Design Operating Schedule

Criteria	Units	Value		Source/Responsibility
		Expected/Avg	Design	
Mineralized ROM Material Delivered to Plant	ktpa		862.5	C
Delivery Schedule				
Days per Week	days	5	5	D2
Shifts per Day	shifts	1	1	D2
Hours per Shift	h		10	D2
Delivered per Shift	dmt		3317	D2

Criteria	Units	Value		Source/Responsibility
		Expected/Avg	Design	
Delivery Rate	$dmt.h^{-1}$		332	D2
Crusher Plant Operating Schedule				
Days per Week	days	7	7	C
Shifts per Day	shifts	3	3	C
Hours per Shift	h	8	8	D2
Utilization	%	≥ 68	68	D2
Operating Hours per Annum	h		5957	D3
Crusher Circuit Throughput	$dmt.h^{-1}$		145	D3
Milling and Flotation Operating Schedule				
Days per Annum	days	365	365	C
Hours per Day	h	24	24	C
Utilization	%	92	92	D2
Operating Hours per Annum	h	8059	8059	D3
Circuit feed rate	$dmt.h^{-1}$	107	107	D3

Table 50 below presents the plant concentrate production basis of design.

Table 50: Concentrate Production

Criteria	Units	Value		Source/Responsibility
		Expected/Avg	Design	
Concentrate Filtration, Drying and Bagging Production Schedule				
Days per Annum	days	365	365	C
Hours per Day	h	24	24	C
Utilization	%	80 - 90	92	D2
Operating Hours per Annum	h	≥ 7000	8000	D3
Circuit Feed Rate	dmt.[h1]	6.58	7.57	D3
Filter Type and Model		2.4m × 10m vacuum belt	D1	
Dryer Type and Model		Rotary kiln	D1	
Dryer Feed Moisture	%	≤ 25	25	D4
Final Concentrate Moisture	%	≤ 1	≤ 1	C
Sifter Screen Aperture Sizes	mesh	50, 80, and 100 (2 × parallel, identical streams)	C/D1	
Final Concentrate Size Class Splits				
+ 50mesh	%	23.6	18 - 30	C
- 50 + 80mesh	%	22.7	17 - 28	
- 80 + 100mesh	%	6.9	5 - 9	
- 100mesh	%	46.8	35 - 59	

17.2 Process Block Flow Diagram

The crushing circuit design comprises three-stage crushing and screening. The crushed product passes through a surge bin from where it is fed to the milling circuit via a coarse-flake classification screen, which produces an under-size fraction for flash flotation and an oversize for primary milling.

Primary milling is followed by rougher flotation which, along with flash flotation, recovers the graphite concentrate from the main stream. Dedicated cleaning / re-cleaning attritioning and flotation circuits upgrade the concentrate to the final specification. Tailings from the flotation circuit are deposited on a dedicated final TSF.

Thickened concentrate from the flotation plant is dewatered by filtration and drying, bagged in discrete, pre-determined size classes and shipped as the final product as required.

Figure 92, Figure 93, Figure 94 and Figure 95 illustrate the basic, overall process flow sheet.

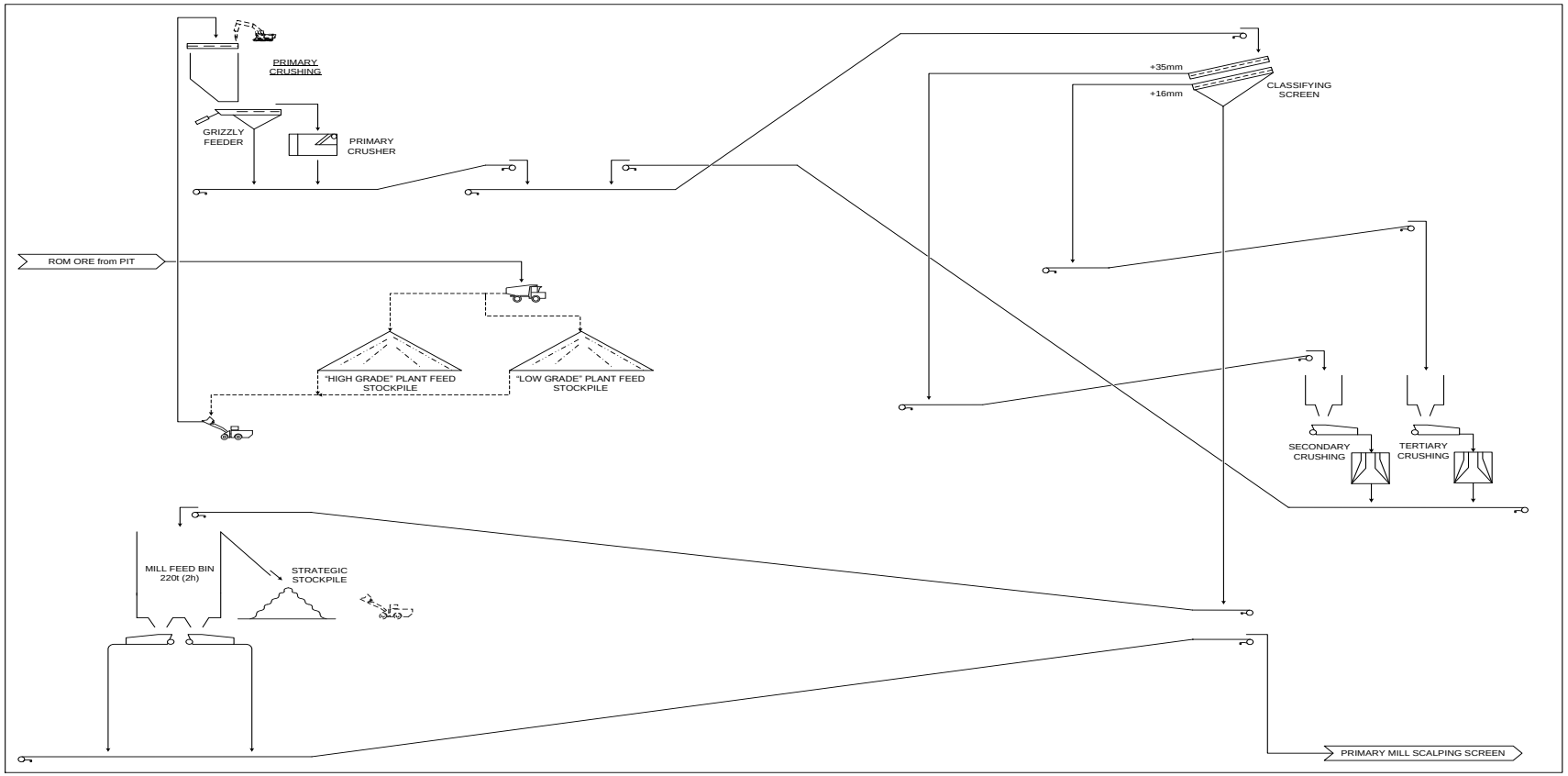


Figure 92: Crushing and Screening

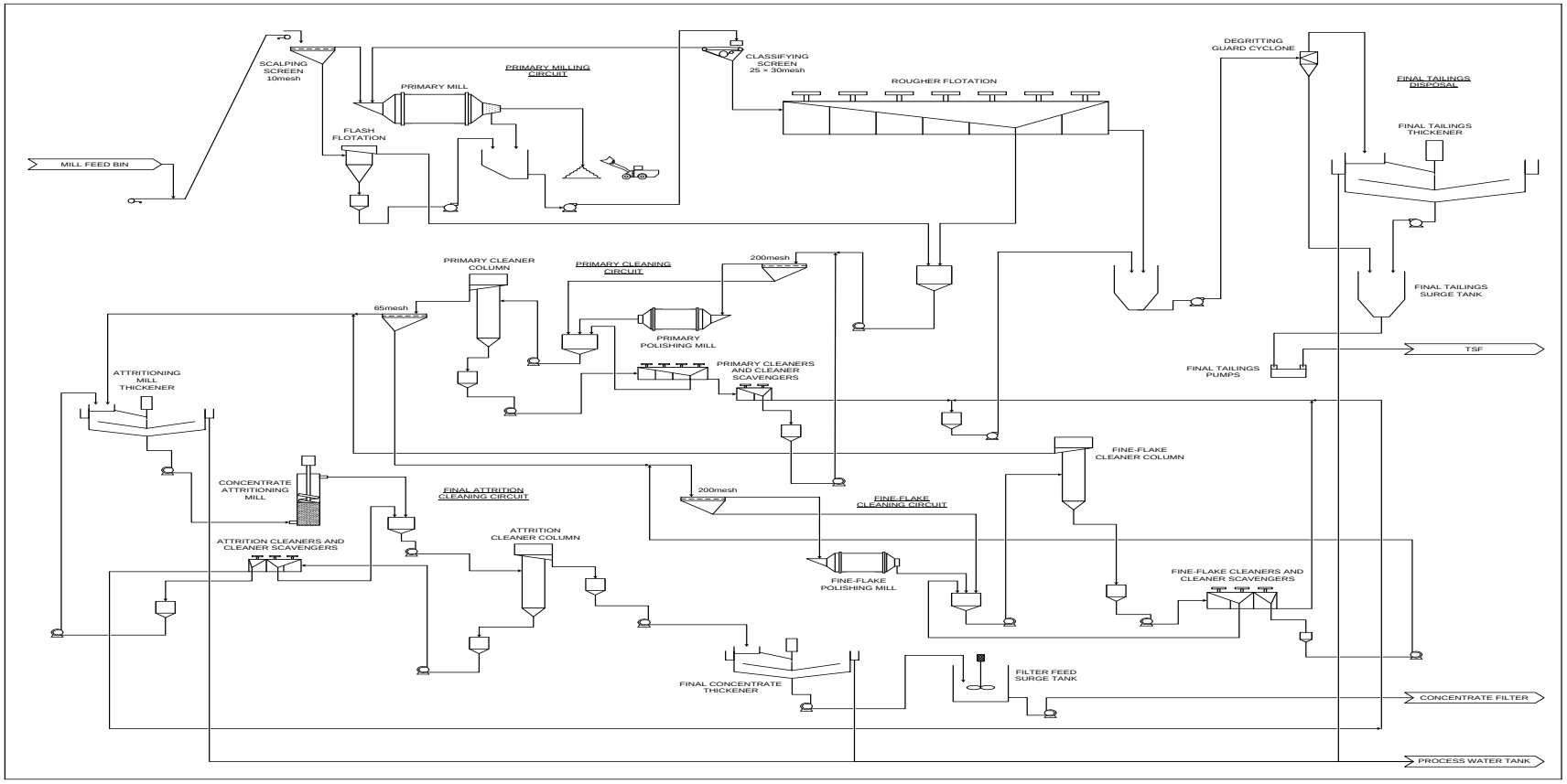


Figure 93: Milling and Flotation

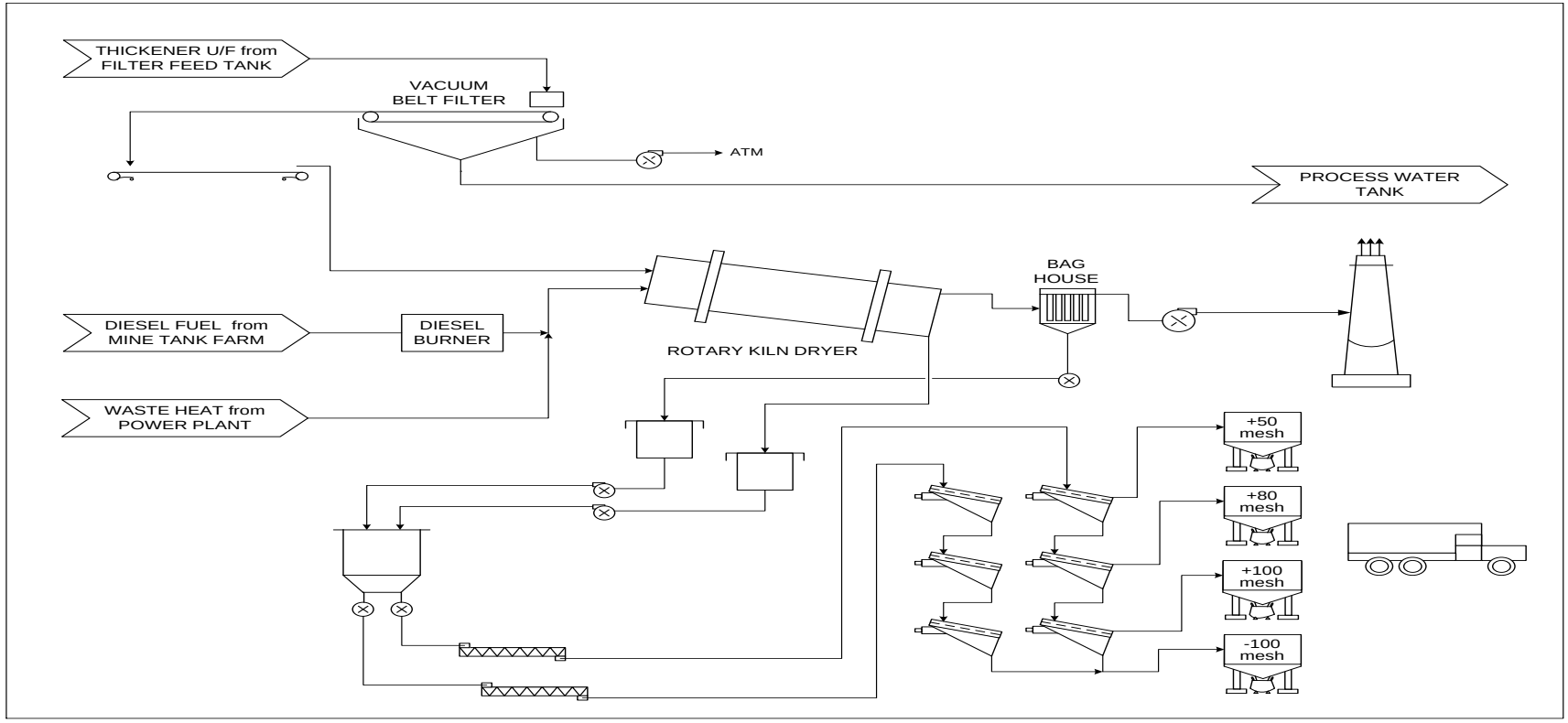


Figure 94: Final Product Handling

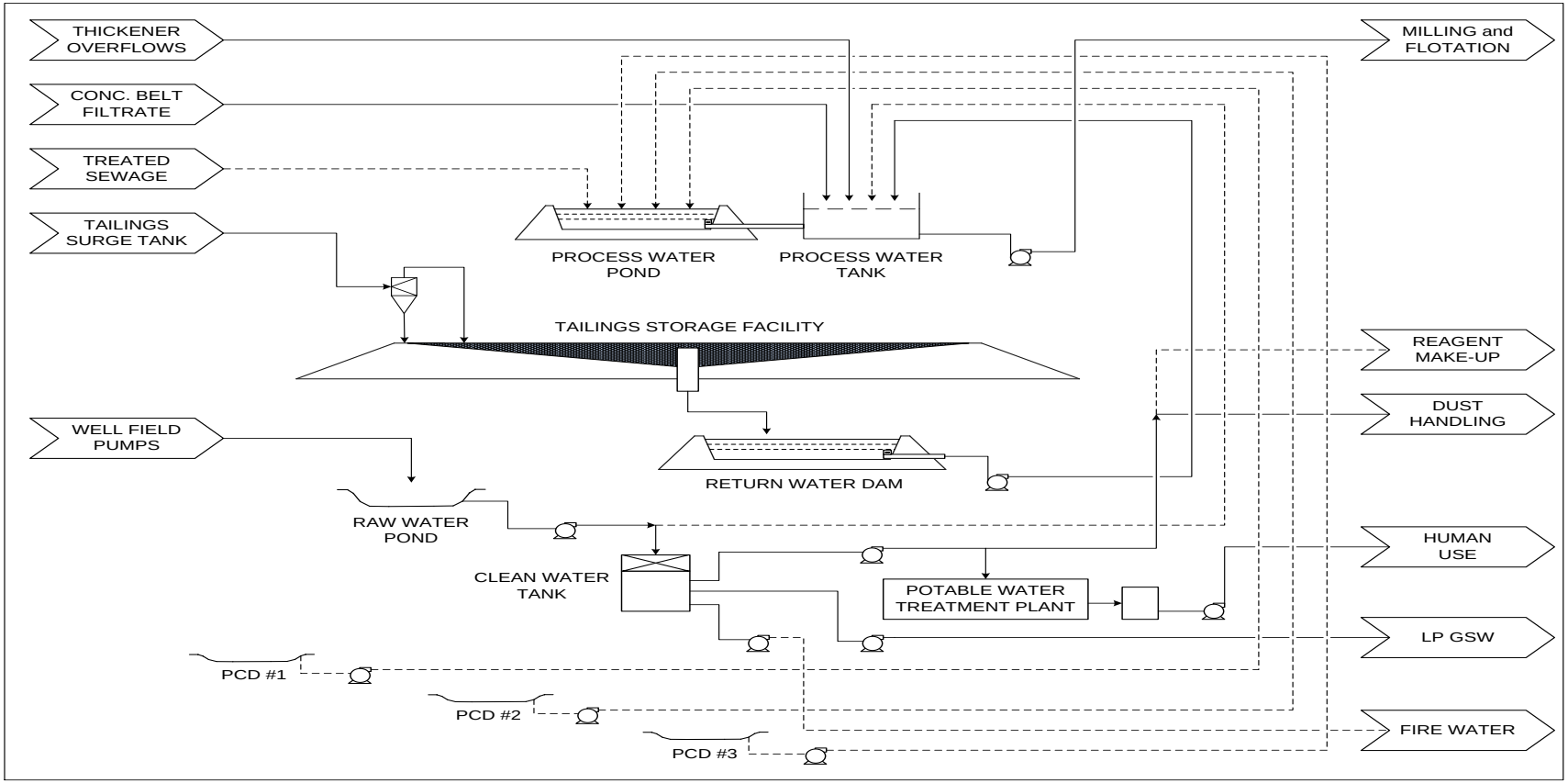


Figure 95: Plant Water Reticulation

17.3 Process Description

17.3.1 Crushing and Sizing

The three stage crushing circuit is designed to operate at ±68% utilization, equating to a feed capacity of approximately 145 dmt.h^{-1}. Mineralized ROM material is delivered to a ROM bin feeding the crushing and screening section of the plant. A static tramp screen removes any oversize material. ROM material is crushed to a product size P_{80} of approximately 13 mm via the three-stage crushing process. Primary crushing is accomplished by means of a jaw crusher operating in concert with a vibrating grizzly feeder, whilst the secondary and tertiary crushers are cone crushers in closed-circuit with a dedicated, double deck classifying screen.

17.3.2 Primary Milling and Flash Flotation

A vibrating scalping screen receives the full stream of crushed plant feed and discharges its oversize material into the primary mill feed chute. The scalping screen undersize material is fed to a flash flotation section. Flash flotation concentrate bypasses the primary milling and rougher flotation stages and is directed to the primary concentrate cleaning circuit to avoid unnecessary milling and minimize degradation of the most valuable coarse-flake material. Flash flotation tails combine with primary mill discharge to constitute the feed to the primary mill classifying screen. Classifying screen oversize is directed to primary mill feed, while the undersize feeds the rougher flotation section.

A single stage primary milling circuit is employed, incorporating a closed circuit linear classifying screen. The primary ball mill size is 4.3m diameter (inside new liners) x 4.6m (EGL) with an installed motor power of 1000 kW.

The primary milling circuit capacity is designed to accommodate a throughput of 862.5 ktpa.

17.3.3 Rougher Flotation

Rougher flotation feed, consisting of undersize from the primary mill classifying screen, gravitates to the feed box of a bank of forced-draught, trough type flotation cells configured in series. Rougher concentrate merges with flash flotation concentrate and the combined stream is pumped to the primary concentrate cleaning circuit. Rougher tailings gravitate to a combined flotation tailings sump, where they merge with the final cleaner flotation tailings. The combined final tailings stream is pumped to the de-gritting guard cyclones ahead of the tailings thickener.

The combined flash and rougher flotation circuit is designed to process the full plant throughput of 862.5 ktpa at an average plant head grade of 7.04% C(t).

17.3.4 Primary Concentrate Cleaning Circuit

The primary cleaning circuit feed, consisting of flash and rougher flotation concentrates along with primary cleaner scavenger flotation concentrate, is pumped to a dewatering screen ahead of a primary cleaner polishing mill. Dewatering screen oversize feeds the mill, while the undersize bypasses the mill, combines with the mill discharge, and is pumped to a primary cleaner column cell.

The primary cleaner column cell concentrate gravitates to a classifying screen, from where the large-flake oversize is directed to a high rate thickener located ahead of a final concentrate attritioning mill. Primary cleaner classifying screen undersize is pumped to a dewatering screen located ahead of a fine-flake polishing mill. The primary cleaner column cell tailings are pumped to a bank of forced-draught, trough type primary cleaner and primary cleaner scavenger flotation cells configured in series.

Concentrate from the primary cleaner cells is recirculated to the primary cleaner polishing mill discharge sump, while the primary cleaner scavenger concentrate is recirculated to the primary cleaning circuit dewatering screen feed as noted above. Primary cleaner tailings gravitate to feed the primary cleaner scavenger cells, while the latter's tailings gravitate to a final cleaner tailings sump and are pumped to the final tailings disposal circuit via the combined flotation tailings sump.

17.3.5 Fine-Flake Concentrate Cleaning Circuit

The fine-flake cleaning circuit feed, consisting of primary cleaner classifying screen undersize and fine-flake cleaner scavenger flotation concentrate, is pumped to the dewatering screen ahead of the fine-flake polishing mill. Dewatering screen oversize feeds the mill, while the undersize bypasses the mill, combines with the mill discharge, and is pumped to a fine-flake column flotation cell.

The fine-flake column concentrate is merged with primary cleaner circuit classifying screen oversize to constitute the total feed to the final concentrate attritioning and cleaning circuit. The fine-flake column tailings are pumped to a bank of forced-draught, trough type fine-flake cleaner and fine-flake cleaner scavenger flotation cells configured in series.

Concentrate from the fine-flake cleaner cells is recirculated to the fine-flake polishing mill discharge sump, while the fine-flake cleaner scavenger concentrate recirculates to the fine-flake cleaner circuit dewatering screen feed, as noted above. Fine-flake cleaner tailings gravitate to feed the fine-flake cleaner scavenger cells, while the latter's tailings are routed to the final tailings disposal circuit via the final cleaner tailings sump.

17.3.6 Final Concentrate Attritioning and Cleaning Circuit

A merged stream comprising primary cleaner classifying screen oversize and fine-flake column concentrate, constituting the total feed to the final concentrate

cleaning circuit, gravitates to a high rate thickener ahead of the final concentrate attritioning mill. Thickener underflow feeds the attritioning mill at a solids concentration of approximately 35% by mass, while the overflow gravitates to the plant process water circuit. Final concentrate attritioning mill discharge feeds the final cleaner circuit column flotation cell.

Final cleaner column concentrate is pumped to the final concentrate thickener as the final plant concentrate, while its tailings are pumped to a bank of forced-draught, trough type final cleaner and final cleaner scavenger flotation cells configured in series.

Concentrate from the final attrition-cleaner cells is recirculated via the final concentrate attritioning mill discharge sump, while the final cleaner scavenger concentrate recirculates to the final concentrate attritioning mill thickener feed. Final attrition-cleaner tailings gravitate to feed the final cleaner scavenger cells, while the latter's tailings gravitate to the final cleaner tailings sump, from where they are routed to the final tailings disposal circuit.

17.3.7 Concentrate Thickening

The flotation circuit design anticipates an average final graphite concentrate production rate of approximately 53 ktpa over the life-of-mine. This fluctuates between 48 ktpa and 61 ktpa, depending on the stage of mine development reached with the passage of time, and assumes a mine-forecast average plant head grade of 7.04% C(t) within an overall range of 6.5 to 7.5% C(t). The concentrate thickening section is designed for the highest expected production demand assuming a plant utilization of 92%. The total concentrate produced is transferred at up to this utilization rate to a high rate thickener ahead of the filtration, drying, sizing and bagging processes.

Thickened concentrate slurry is pumped to a filter feed stock tank. Overflow from the concentrate thickener gravitates to a plant process water tank. Concentrate slurry is pumped on a demand basis from the stock tank to a linear vacuum belt filter for further dewatering.

17.3.8 Concentrate Filtration and Drying

Final concentrate is pumped from the stock tank to the belt filter on demand. The dewatered filter cake so produced is fed into a rotary kiln drying circuit, where the remaining moisture and entrapped volatiles are driven off. Filtrate from the belt filter reports to the plant process water tank.

17.3.9 Concentrate Sizing and Despatch

A three-stage, twin stream vendor-package sifting plant screens the dry concentrate into the different size fractions required as saleable product.

An associated vendor-package bagging plant is employed to receive the different size fractions discretely. The final concentrate is weighed, sampled and bagged into 1 m^3 bulk bags (±600 kg each, depending on flake-size class) for despatch.

The bulk bags are sealed, covered with ultraviolet resistant plastic and placed on wooden pallets for storage ahead of despatch. Despatching takes place on a daily basis using standard 40' shipping containers. A pre-determined number of bags is packed into each container, the containers are loaded onto trucks, and the trucks transfer the final product to the port at Fort Dauphin ahead of sea freighting to the required destination.

17.3.10 Final Tailings Disposal

Combined rougher and cleaner flotation final tailings are pumped to a guard de-gritting cyclone installation ahead of a high rate final tailings thickener. Cyclone overflow feeds the thickener. Cyclone and thickener underflows combine and are pumped for final disposal to the TSF via a single, common, above-ground HDPE-lined mild steel pipeline using positive displacement piston diaphragm pumps (one operating, one standby). Thickener overflow is collected in the plant process water tank for re-use to minimize make-up water consumption.

17.3.11 Services

Instrument quality air is produced by a compressed air installation at the plant consisting of two automatically modulating screw type compressors with associated dryers and receivers, and distributed via a discrete, integrated header system. Blower air to service the forced-draught flotation cells is supplied using two automatically modulating Roots-type positive displacement blowers and distributed via a discrete, integrated header system.

The plant water circuit is configured to minimise raw water usage. Raw make-up water supplied from well field boreholes into the plant raw water pond is used as-is for process top-up water, and filtered for reagent make-up, potable water generation, dust suppression, pump gland service, mill bearing cooling and fire water.

Process water is supplied to the various points-of-use within the milling and flotation circuits via an integrated header system. Potable quality water for human consumption and "domestic use" within the plant area is produced by a vendor-package potable water treatment plant and distributed via a header system as required.

Pollution control dam ("PCD") water and treated effluent streams from the various sewage plants are recycled via the plant process water pond.

Plant thickener overflows, concentrate filter filtrate and decant and seepage water from the TSF return water dam are reused via the plant process water tank.

17.3.12 Reagents

Diesel fuel collector is pumped from the mine tank farm to maintain a level in a plant collector storage tank. A manifold system on the storage tank 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) liquid frother is delivered by road to a plant reagent store in 1m^3 IBCs or 210 litre steel drums. The drums are collected by forklift as required and the contents pumped into a frother storage tank. A manifold system on the storage tank supplies multiple variable speed peristaltic pumps, which discretely pump the frother at set rates to the various points-of-use within the flotation circuits.

Flocculant powder (Magnafloc 919 and Magnafloc 24 for concentrate and tailings respectively) is delivered by road to the plant reagent store in 25kg 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, one each being dedicated to supplying the concentrate and tailings thickeners due to the flocculant types required being different for each application. Variable speed peristaltic pumps discretely pump the flocculant at set rates to the thickeners' points-of-use.

Coagulant powder (Magnafloc 1707) 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.

17.4 Tailings Storage Facility (TSF)

Total plant final tailings are pumped to the TSF via a single, common, HDPE-lined mild steel overland pipeline as a slurry containing ±54% solids by mass, by way of positive displacement piston diaphragm pumps (one operating, one standby). Hydrocyclones located at the TSF are utilized on the tailings stream to produce a coarse underflow for TSF wall construction activities. Cyclone overflow slimes report open-ended to the TSF beach drainage areas.

The TSF, designed by SRK Consulting to accommodate approximately 20.5 Mt of Molo final tailings over a 26 year period at an average plant treatment rate of around 862.5 ktpa and mass pull to tailings of approximately 94%, is located ±1.5 km west of the plant.

While the TSF itself is unlined, the associated return water dam is lined to reduce water loss through seepage into the underlying aquifer.

Stage capacity assessments conducted to limit initial CAPEX commitment indicate that a 30 ha Phase 1 compartment will have the capacity to accommodate 2 years

of operation before a 60 ha Phase 2 compartment is required. The 60 ha Phase 2 compartment will be needed to provide the necessary capacity for the remaining life-of-mine. For the Phase 1 compartment a ~10m maximum height (pending varying ground slope) starter wall constructed from borrow materials is required around the perimeter of the footprint. For the following Phase 2 development, a maximum starter wall height of approximately 8m is envisaged. A borrow area located approximately 6 km west (10 km by access road) of the TSF provides material for the construction of the TSF starter walls. The final maximum wall elevation of the TSF will be approximately 39m and allows for 1m of freeboard.

TSF drainage systems, including underdrainage and a penstock decant structure, both reporting to the return water dam (RWD), are allowed for to accommodate process water and storm events. Storm water diversion drainage is provided to divert clean water around the facility. The RWD, located immediately downstream of the TSF, has an approximately 5m high wall and stores water reporting from the decant system draining water from the surface of the TSF, the perimeter surface "solution" drains, and from the underdrains at the base of the TSF. The RWD also accommodates all storm water recovered in the PCDs on site, which are maintained empty to accommodate storm water inflow. Recovered return water is pumped overland back to the plant process water system. The RWD has a storage capacity of approximately 571 000 m^3.

17.5 Energy, Water and Process Materials Consumption

17.5.1 Energy

The plant energy requirements are determined from the mechanical equipment list applying a utilization percentage to each processing unit. The plant installed power is 4.7 MW, with the primary milling plant constituting 26% of the installed power. The operational equilibrium power draw is anticipated to be ±2.7 MW. The electrical power is supplied to the plant from the main Mine diesel power generation plant (refer to Chapter 18 Project Infrastructure).

17.5.2 Water

The major raw water source was selected by conducting a historical, prevailing and forecast surface and underground geo-hydrological study. Water is recovered from sustainable well fields thus identified utilizing several strategic boreholes, and pumped overland to storage facilities at the plant.

In terms of the processing plant, the boreholes supply the make-up water directly to a dedicated, 2 500 m^3 raw water pond. Raw water is pumped from the raw water pond for utilization as follows:

- Via a self-cleaning filtration system into a clean water storage tank, from where it is distributed as required to the reagent make-up, dust suppression, centrifugal pump gland service water, mill bearing cooling water and fire water

systems. In addition to these, the filtered raw water supplies feed to a potable water treatment plant, the latter being a discrete vendor-package system which clarifies and purifies the water for human consumption purposes

- As is, for last resort make-up on demand to maintain an adequate level in the 5 000 m^3 plant process water pond, which is interconnected with the plant process water tank. Process water is used for slurry dilution in the milling and flotation operations and for general purposes

Provision is made for transferring surplus water accumulated at the mine and plant locations to the return water dam via the return water line using the plant raw water pumps, in order to facilitate flexibility in water distribution across the operation.

17.5.3 Process Consumables

Reagent regimes employed were established during the course of flotation and settling test work conducted by SGS and Metpro in consultation with relevant vendors as needed.

The nature and consumption of grinding media was determined based on a combination of comminution test work, vendor information and estimates and assumptions used in relevant milling simulations.

Table 51 below summarizes the extent of the consumption of these various commodities.

Table 51: Energy, Reagent and Water Consumption Rates

Description	Units	Consumption
Electrical Power		
Plant Consumed Power	kW	2 700
Reagents		
Collector	$g.t^{-1}$ of plant feed	80
Frother	$g.t^{-1}$ of plant feed	130
Flocculant	$g.t^{-1}$ of plant feed	130
Coagulant	$g.t^{-1}$ of plant feed	102
Water		
Total Clean Water Usage	$m^3.h^{-1}$	35
Dust suppression	$m^3.h^{-1}$	6
Gland seal water	$m^3.h^{-1}$	16
Reagent make-up	$m^3.h^{-1}$	12
Potable water make-up (plant area only)	$m^3.h^{-1}$	1
Process Water Make-up	$m^3.h^{-1}$	8
Raw Water Make-up	$m^3.h^{-1}$	46
Return Water to Plant	%	65

17.6 Equipment List

Table 52 presents the list of all major process equipment used as the basis for developing the capital cost estimate.

Table 52: List of Major Mechanical Equipment

Description	Specification
Mineralized ROM Feed Crushing	
Primary Crusher	30’ × 42’ single toggle, jaw
Secondary Crusher	Cone 38 mm CCS (extra coarse liner) – 132 kW
Tertiary Crusher	Cone 16 mm CCS (fine liner) – 132 kW
Crusher Closed-Circuit Screen	8’ × 18’ vibrating, double deck; upper and lower deck apertures 35 mm and 16 mm respectively
Milling, Classification and Flotation Dewatering	
Primary Mill	4.3m × 4.6m
Primary Polishing Mill	1.6m × 2.5m
Fine Flake Polishing Mill	1.6m × 2.5m
Final Concentrate Attritioning Mill	SMD 18.5P
Primary Mill Scalping Screen	1.8m wide × 3m long, horizontal, vibrating single-deck, 10 mesh
Primary Mill Classification Screen	25 x 30 mesh linear, drainage area 9 m^2
Primary Cleaner Dewatering Screen	Vibrating 4’ x 8’ DWS, 200 mesh
Primary Cleaner Classifying Screen	Vibrating 2 deck stacker sizer, 65 mesh
Fine-flake Cleaner Dewatering Screen	Vibrating 2 deck stacker sizer, 200 mesh
Final Concentrate Attritioning Mill Thickener	7m diameter high-rate c/w feedwell dilution system, bolted design
Flotation	
Flash Flotation Cell	2.2 m^3
Rougher Cells	7 × 8.5 m^3 trough cells, c/w froth scrapers
Primary Cleaner Column Cell	3.2m diameter x 8m height

Description	Specification
Primary Cleaner Cells	4 x 1.4 m^3 trough cells, c/w froth scrapers
Primary Cleaner Scavenger Cells	2 x 1.4 m^3 trough cells, c/w froth scrapers
Fine-flake Cleaner Column Cell	2m diameter x 8m height
Fine-flake Cleaner Cells	2 x 2.8 m^3 trough cells, c/w froth scrapers
Fine-flake Cleaner Scavenger Cell	1 x 2.8 m^3 trough cell, c/w froth scrapers
Final Attritioning Cleaner Column Cell	2m diameter × 8m height
Final Attritioning Cleaner Cells	2 × 1.4 m^3 trough cells, c/w froth scrapers
Final Attritioning Cleaner Scavenger Cell	1 × 1.4 m^3 trough cell, c/w froth scrapers
Final Concentrate Handling	
Final Concentrate Thickener	7m diameter high-rate c/w feedwell dilution system, bolted design
Final Concentrate Dewatering Filter	2.4m wide x 10m long belt filter, sized for ≥7.6 $dmt.h^{-1}$
Final Concentrate Dryer	3m ID x 15m length rotary kiln; vendor package sized for ≥7.6 $dmt.h^{-1}$ at feed moisture ≤25%
Final Concentrate Sizing Sifters	>50, >80 and >100mesh twin parallel vibrating screen sets; vendor package sized for ≥7.6 $dmt.h^{-1}$
Final Concentrate Bagging Plant	>50, >80, >100 and <100mesh bagging system; vendor package sized for 2 bulk bags per hour
Final Tailings Handling	
Final Tailings Guard (de-gritting) Cyclones	2 × 500mm diameter (operating and standby)
Final Tailings Thickener	15m diameter high-rate c/w feedwell dilution system, bolted design
Final Tailings Pumps	2 × piston diaphragm positive displacement (operating and standby)

Description	Specification
Services	
Clean Water Filter	200 µm; vendor package
Potable Water Plant	Containerized vendor package
Instrument Air Compressors	2 off modulating screw type, 7.59 $m^3.min^{-1}$ delivery at 10 bar (FAD)
Flotation Air Blowers	2 off modulating; 40 kPa
Reagents	
Collector Mixing System	2 m^3 storage tank; 11 peristaltic dosing pumps
Frother Mixing System	2 m^3 storage tank; 11 peristaltic dosing pumps
Flocculant Mixing System	Vendor package
Coagulant Mixing System	Vendor package

17.7 Chemical Treatment

No financial, or operational calculations and/or scenarios have been provided for in the Molo 2015 FS financial model with regards to downstream value-adding processing of the graphite concentrate. This includes, amongst others, hydrometallurgical purification, spheroidization coating for battery grade graphite, and thermal expansion for specialty graphite applications, such as foils.

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18 PROJECT INFRASTRUCTURE

18.1 General Infrastructure Overview

The infrastructure for the Molo Graphite Project located in southern Madagascar has been designed and specified to support mining, crushing and screening, processing plant, bagging and logistics and administration. The infrastructure was positioned on site to optimise the operation of the mining fleet, the processing plant, tailings storage facility and the construction and permanent camps.

The permanent infrastructure to be installed on site at the mine as part of the project will include the following:

- Mining
- Plant
- Shared and Services
- Tailings Storage Facility

The temporary infrastructure and facilities that are to be installed to support the construction activities will be erected in the form of a construction camp and associated facilities. The proposal for the construction camp is to accommodate all staff (management, skilled and unskilled) in a tented construction camp.

The plant and mine layout plan is included in Figure 96, below and details the infrastructure to be installed as part of the Project.

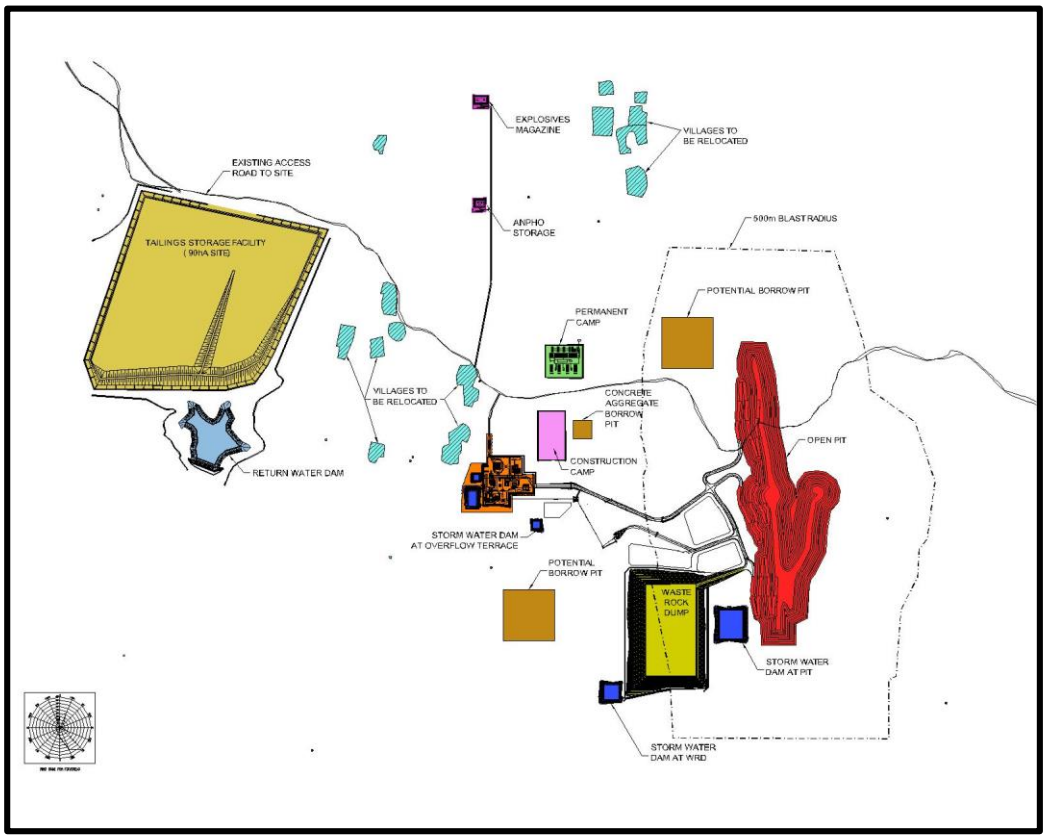

Figure 96: Plant and Mine Overall Layout

The location and layout of the plant within the mining tenement was determined utilising the environmental sensitivity plan produced by GCS Water and Environmental. As part of the environmental investigations and assessments on site, the local fauna and flora was mapped together with existing water extraction points, farmland and water courses. The plant and associated infrastructure was located on site with the aim to minimise the effect on the existing environment. The environmental sensitivity plan including the plant and associated infrastructure layout is shown below in Figure 97.

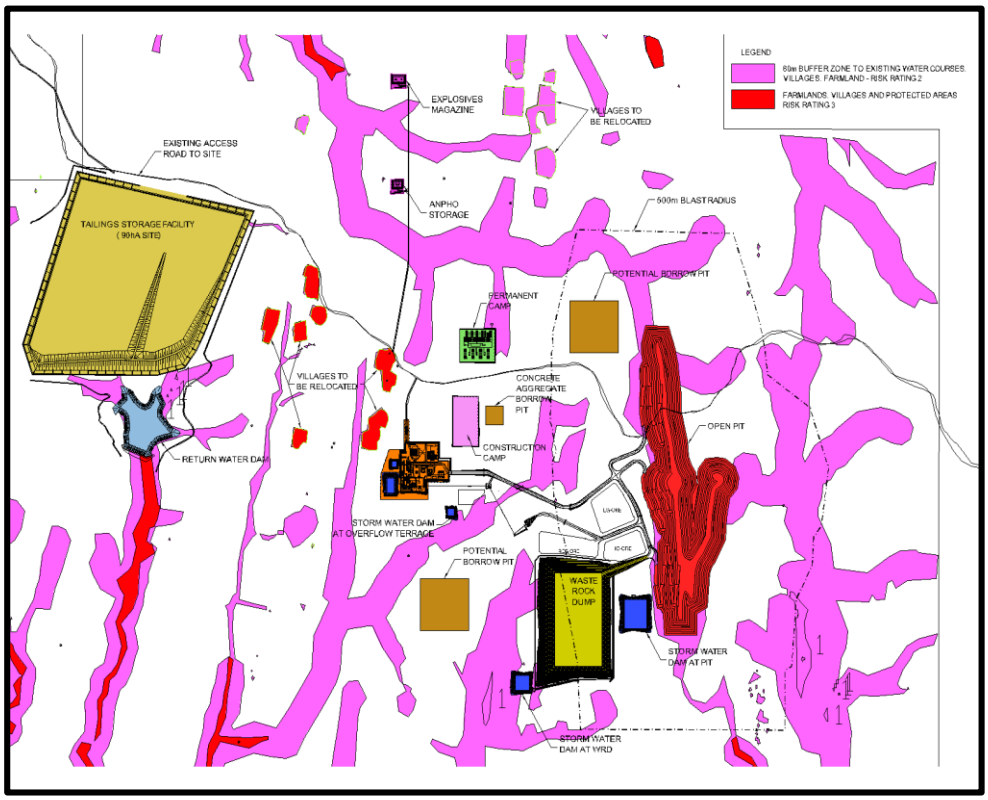

Figure 97: Sensitivity Plan

The total staff compliment for the operation phase of the mine, together with the proposed maintenance strategies for the mine were utilised to determine and size the permanent camp, plant and mine offices, training facilities, workshops, warehousing, storage, potable water and waste water requirements for the project. Temporary facilities to accommodate the construction works were also included and where equipment could be utilised during the construction phase of the works and later relocated to the permanent works, this philosophy was adopted in an effort to reduce the capital costs of the project.

18.2 Mining Infrastructure

The infrastructure to support the mining operation was developed by DRA and positioned to optimise the operation of the mining fleet.

The mining infrastructure includes:

- Mining Office
- Earth Moving Vehicle (EMV) Workshop
- EMV Wash bay

- ANFO Storage Facility
- Explosives Magazine
- Explosives Accessories Magazine
- Access road to ANFO Storage and Explosives Magazine
- Main Haul Roads to accommodate Articulated Dump Trucks (ADT)
- ROM Tip Ramp

18.2.1 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.

18.2.2 Earth Moving Vehicle (EMV) Workshop

The EMV Workshop is the main workshop which will serve the mining fleet (mainly the ADT's) 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 98 below is an illustrative layout view of the proposed EMV workshop to be installed as part of the permanent works on site.

Figure 98: 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.

18.2.3 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.

18.2.4 ANFO Storage

The ammonium nitrate / fuel oil (“ANFO”) storage building is located to the north of the mine and plant site, a minimum of 500m 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.

18.2.5 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.

18.2.6 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.

18.2.7 Access Road to Explosives Magazine and ANPHO 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.

18.2.8 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.

18.2.9 Run of Mine (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 3.5m.

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)

18.3 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

18.3.1 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.)

18.4 General Engineering Services (GES) Workshop

The General Engineering Workshop (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) 6m container offices and eight (8) 12m containers which will be used for storage.

18.5 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).

18.6 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. Demands for raw water make-up are on average 561 cubic metres per day (m^3/day) over the life of mine for the processing plant. 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.

The area covered by the 5 km radius well field is shown Figure 99 indicating existing production boreholes and target areas for future expansion. The existing test production boreholes drilled in the area identified the marble layers to the west of the site as the most productive aquifer. Five existing production boreholes have a combined safe yield of 1,322 m^3/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 m^3/day. The well field abstraction will, however, be capped at a conservative abstraction rate of about 1,430 m^3/day, taking aspects such as 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.

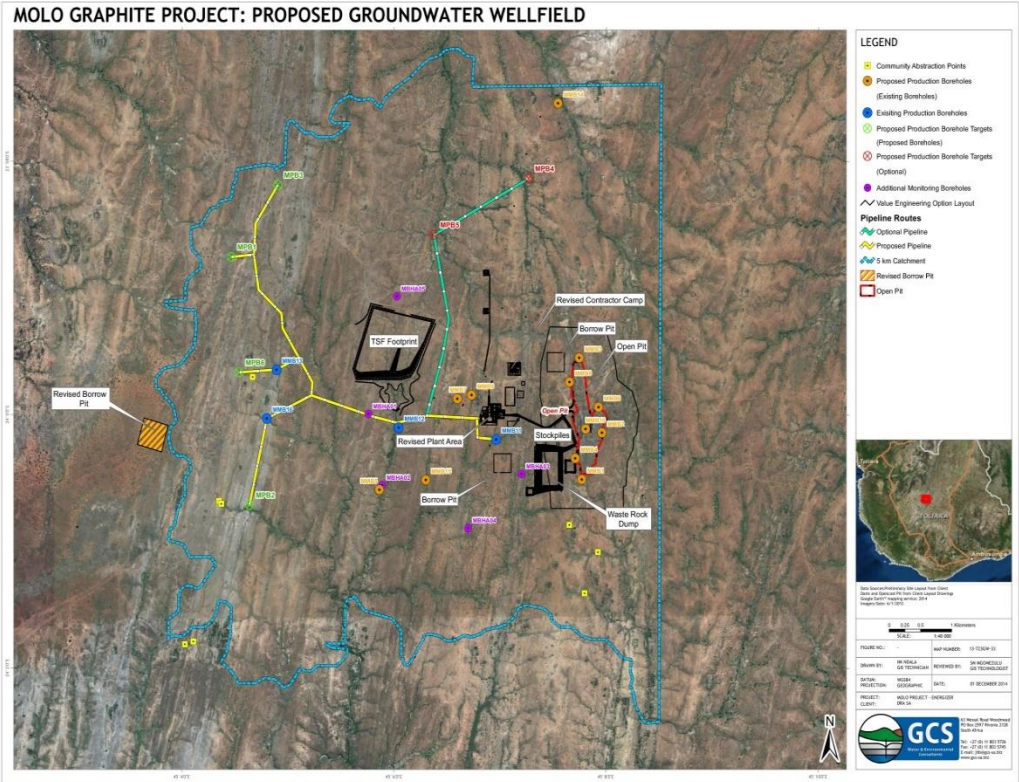

Figure 99: 5 km Radius Well Field

The raw water dam will receive water from the well field system or the return water dam, whichever is required, dependent on supply availability from each water source. If required, the raw water dam will overflow to the process water dam during storm events. Excess water at the process water dam will be pumped to the return water dam.

18.7 Process Plant, Storm Water and Raw Water Supply Dams

The process and storm water dam at the plant have been combined to reduce the footprint size of the dam. The dam is located to the west of the plant terrace, adjacent to the plant in order to minimise the piping and pumping requirements. The process and storm water dam has been sized to accommodate 12,700 m³ of water which caters for 10,000 m³ of process water and 2,700 m³ of storm water storage capacity. The volume of process water will accommodate 20 hours of plant operation in the event of failure of the water supply system from the well fields, or the return water dam.

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 optimised.

The proposed SWMP measures include the implementation of four Pollution Control Dams, 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. All Pollution Control Dams volumes were dimensioned on 24-hour storm event with a return period of 50 years. The TSF Return Water Dam was additionally sized to accommodate a portion of the overflow from the Pollution Control Dams as an additional precaution.

Potential discharge will be centralised through the Pollution Control Dams during wet periods and water quality will comply with IFC and local Madagascan standards. Process water will however be operated in a closed system which will not overflow.

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.

18.7.1 Ablution Facilities

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

18.7.2 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.

18.8 Shared Infrastructure and Services

The shared infrastructure and services to support the processing plant operation and the mining operation was developed by DRA 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 and Transformer Bays
- 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

18.8.1 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 6m wide with 0.5m wide shoulders (7m 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.

18.8.2 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 are detailed in Table 53 below:

Table 53: 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	

18.8.3 Gate House and Turnstile Access Control

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

18.8.4 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.

18.8.5 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.

18.8.6 Waste Water Treatment Plants

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 Affair (“DWA”) General Limits, South Africa.

Figure 100: Typical Configuration of Waste Water Treatment Plant

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.

18.8.7 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 VSD's 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.

18.8.8 Storm Water Drainage

Storm water run-off will be diverted around the plant and mine terrace, the construction camp terrace, the permanent camp terrace, the open cast mine pit, stockpile areas, waste rock dump and the tailings storage 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 cast mine pit, stockpile areas, waste rock dump and the tailings storage facility is considered dirty water and as such, is collected in the storm water dams located on site and discharge to the environment is limited. All the storm water dams on site have been sized to accommodate the 24 hour, 1:50 year return period storm event.

The storm water falling on the plant and mine complex terrace is collected in concrete lined drainage channels and discharged to the combined process and storm water dam located adjacent to the plant and mine complex terrace via a silt trap. The concrete lined drainage channels have been designed to match the fall on the terrace and to maintain a self-cleaning velocity to reduce settlement of sedimentation.

Storm water falling on the construction camp and the permanent camp terraces is considered clean water and as such it will be discharged direct to the environment.

18.8.9 Storm Water Dams

The storm water dam located adjacent to the surge bin overflow stockpile, storm water dam “C”, has been sized to accommodate all the storm water run-off from the surge bin stockpile area. The dam has been sized to accommodate the 24 hour, 1:50 year return period storm event and will accommodate 2,300 m^3 of storm water.

The storm water dam located adjacent to the waste rock dump area, storm water dam “D”, has been sized to accommodate all the storm water run-off from the waste rock dump. The dam has been sized to accommodate the 24 hour, 1:50 year return period storm event and will accommodate 23,000 m^3 of storm water.

The storm water dam located adjacent to the pit, storm water dam “E”, has been sized to accommodate all the storm water run-off from the pit dewatering operation. The dam has been sized to accommodate the 24 hour, 1:50 year return period storm event and will accommodate 63,240 m^3 of storm water.

18.8.10 Consumer Sub-station

The consumer sub-station will be a containerized and installed on concrete bases and plinths with steel access stairways and landings. Access will be via the ends of the container with one side having double door openings and pedestrian access on the other side. All guard railing to the landing on the double door access side

will be removable to allow for equipment to be installed and removed as required throughout the duration of the mines life.

18.8.11 MCC's and Transformer Bays

The Motor Control Centres (MCC)'s on the project will be a containerized and installed on concrete bases and plinths with steel access stairways and landings. Access will be via the ends of the container with one side having double door openings and pedestrian access on the other side. All guard railing to the landing on the double door access side will be removable to allow for equipment to be installed and removed as required throughout the duration of the mines life. Transformer bays will be brick built structures on concrete foundations and will house the transformer units.

18.8.12 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 fence at the tailings storage facility will be 2m high diamond mesh security fence with a coil of razor flat wrap at the top of the fence. The fence will include for a 4m wide vehicle access gate and a pedestrian access gate.

The security at the plant and mine complex, the permanent and construction camp areas will be 2.4m high diamond mesh security fencing with 3 strands of galvanised barbed wire to the top of the fence. 6m 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.

18.8.13 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.

18.8.14 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 client with the assistance of GCS Water Consultants.

18.8.15 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 m^3 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 m^3.

18.8.16 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.

the process plant in addition to the storage requirements to accommodate the storm water run-off off the tailings storage facility. The total storage volume of the return water dam is 571,000 m^3.

18.8.17 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) 12m 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 cranage will be supplied by the mobile plant and no separate provision has been included for cranage at the welding workshop.

18.8.18 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) 12m shipping containers which will be refurbished and used as offices and storage.

18.8.19 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.

18.8.20 General Storage Yard

An open area has been identified on the plant terrace for a general storage yard which will be utilised to store oversize equipment and spares for the plant and mine operations that are not stored at the strategic spares storage area. Provision is included for security fencing at this location. An area of 350 m² has been allocated for the general storage area.

18.8.21 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 101: Typical Layout of Containerized Fuel Storage Tank

18.9 Power Generation Facility

The site for the proposed Molo Graphite Project mine is located in Southern Madagascar where no utility power is available. In order for the mine to operate, power must be generated on site, the cost of which makes up a significant portion of the overall production cost of the final processed product.

18.9.1 Predicted Electrical Consumption and Notified Maximum Demand

Power plant capacity requirements were derived from both Process Engineering load estimates and from power consumption estimates of site infrastructure. A bottom up estimating methodology was used to arrive at a predicted Notified Maximum Demand ("NMD") and electrical consumption for the proposed installation.

The complete list of connected loads, as per the MEL, was summed to arrive at a total connected load. In addition to MEL loads, estimated values of assumed auxiliary loads were also added to the total connected load, as well as assumed loads for plant infrastructure. The connected loads were reduced to connected running loads by excluding standby circuits and further reduced to absorbed power running loads by assuming a utilisation factor of 70 - 80% for most equipment. Typical motor efficiency and power factor values were also taken into account in calculating the running load.

Infrastructure loads, include mining and process plant workshops and offices as well as the on-site accommodation camp were also taken into account.

For the complete installation, power demand can be summarised as follows:

Table 54: Power Summary

Category	Value
Installed Power (MW)	6.65
Total running maximum demand (MW)	3.5
Process plant maximum running demand (MW)	2.7
Infrastructure maximum running demand (MW)	0.8

The maximum electrical step load is 1 000 kW (LRS start), when starting the primary mill.

18.9.2 Proposed Power Supply Solution

The proposed power supply solution consists of an owner owned and operated package power plant, made up of multiple generator sets, supplying power at 11 kV to meet the site load requirements, as well as providing a stable supply under maximum step-load conditions.

18.9.3 Fuel

Diesel is considered the fuel most suitable for power generation, given its availability in Madagascar. Bulk fuel storage for two weeks would be kept on site, including that consumed by the mining fleet.

18.9.4 Power Plant

Various power plant vendors were approached to submit proposals for turnkey power plant solutions that would meet the site power requirements. Power plants also needed to include redundant generating units to allow units to be taken out of service for maintenance without affecting power supply.

Vendor solutions were assessed on the basis of capital cost, but primarily on operating cost over the 26 year life of mine by taking into account fuel consumption rates across fluctuations in plant power demand, as well as maintenance costs.

The power plant solution deemed most suitable can be summarised as follows:

Table 55: Power Plant Capex

Category	Value
Power Plant Arrangement	4 x 2 MW Diesel Generator Sets
Capital Cost (USD)	10,600,000
Operating Cost (USD/kWh)	0.20

Under normal operation there will only be two units running, with a third allowed to assist with mill starting, and the fourth unit as a spare for maintenance.

18.10 Construction Camp Infrastructure

The construction camp has been designed to accommodate 800 people during the peak of construction. Provision has been made for 200 skilled personnel and 600 unskilled personnel. The preferred accommodation solution is a tented solution and includes for the following:

- 200 skilled personnel in 100 tented units 2 per unit with a shared bathroom)
- 600 unskilled personnel in 76 tented units (8 per unit with shared ablution facilities)
- Ablution facilities (20)
- Kitchen Facility
- Prep Area
- Scullery
- Diner - Skilled Personnel
- Diner - Unskilled Personnel
- Recreation - Skilled Personnel
- Recreation - Unskilled Personnel
- Laundry and Ironing
- Camp Office
- Electrical System
- Water Provision System
- Sewer System

It is proposed that the construction camp be decommissioned after the construction phase of the works with the 340 permanent personnel being housed in the permanent construction camp.

Figure 102: Typical Layout of Tented Camp Solution

18.11 Permanent Camp Infrastructure

The permanent accommodation camp facility has been designed to accommodate 340 personnel. This includes for 20 managers, 120 skilled personnel and 200 unskilled personnel.

The permanent camp 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. This form of construction has been shown to be the most economical as the shipping costs are reduced, the erection and commissioning time is reduced and the installation of all services to the building is completed at the fabricators workshops which also reduced installation time.

The permanent accommodation camp will include for the following:

- Manager Accommodation – 20 personnel (20 units)
- Skilled Accommodation – 120 personnel (60 units)
- Unskilled Accommodation – 200 personnel (20 units)
- Laundry
- Kitchen / Diner / Mess
- Ice Rooms
- Camp Office
- Ablution Blocks (20 - included in unskilled accommodation units)

The permanent camp will be located on a separate terrace and the buildings will be situated on concrete slabs which will be installed by the civils and earthworks contractor as part of the overall civil and earthworks package. All buildings will be provided with a roof structure and steel stair access and walkways.

In addition to the provision of accommodation for the plant and mine personnel operating the plant and mine, provision has been included for in the accommodation for the camp management personnel, site security, haul truck personnel, visitors and Client representatives.

There is no provision for dedicated storm water reticulation on the permanent camp terrace and storm water will run off the terrace and into the environment.

Figure 103: Typical Permanent Camp Modular Units

18.12 Electrical, Control and Instrumentation (EC&I) Infrastructure

Due to the remote location of the project site, no utility power is available, thus creating the need for on-site power generation. Electrical power is generated by a 8MW diesel driven power plant and distributed at 11 kV to the main Process Plant area and accommodation camp. There are seven remotely located boreholes which will each be equipped with a local 17 KVA, 550V generator.

11 kV is distributed from the power plant via dual feeders to the 11 kV consumer sub-station located adjacent to the Mill building. 11 kV is distributed, either by cable, or overhead line, from the consumer sub-station to various load centres, where the voltage is stepped down to low voltage levels for local distribution in process and infrastructure installations.

Table 56: Power Distribution Summary

	Voltage Levels
Medium Voltage Distribution	11 kV
Mill Motor (1000 kW)	11 kV
Process Plant Equipment	525V
Small Power and Lighting and General Infrastructure	220/380V

Plant voltage levels can be summarised as follows:

The distribution design uses 11 kV/550V transformers to step down the voltage for larger process plant installations located in the main plant area, and 11 kV/550V miniature sub-stations for smaller, more remote process plant areas. Power flow to individual process plant equipment is controlled by local Motor Control Centres (MCC) that are fed by 11 kV/550V transformers, or miniature sub-stations.

Small power and lighting and general infrastructure loads, including the permanent accommodation camp, are fed by either 11 kV/400V miniature sub-stations, or appropriately rated transformers with 400V secondary's. Distribution to individual loads is via 380V distribution boards and 220V sub-distribution boards.

In general, appropriately rated medium and low cables were used in the power distribution design, but 11 kV overhead lines were used to feed load centres situated more than 500m from the nearest 11 kV distribution point.

All medium and low voltage sub-stations will be made up of modified shipping containers.

Apart from containing Medium Voltage Switchgear and Low Voltage Motor Control Centres, sub-stations shall also house the following equipment:

- 525V variable speed drives
- Local sub-station small power and lighting distribution boards
- Lighting and small power transformers to feed plant located lighting and small power distribution boards (located in outdoor transformer bays where applicable)
- Control System Network and Remote I/O Panels
- Uninterruptible power supplies (UPS) to power Network and Remote I/O Panels and field instruments (Dual redundant configuration)
- Fire protection system
- Air conditioning and ventilation system

Remote MCC's with a small number of motor loads will be housed in weatherproof outdoor panels.

18.13 Site Access and Logistics

18.13.1 Access Road to Site

DRA commissioned Bolloré Africa Logistics Madagascar to consider the following potential routes to the Molo site and to recommend the best route and alternatives for the Project:

- Tulear to site (western side of Madagascar)
- Soalara to site (western side of Madagascar)
- Fort Dauphin to site (eastern side of Madagascar)

Detailed route surveys were undertaken for each of the above mentioned routes. These surveys included a detailed photographic report showing distances, way points, estimated travel times, and all hazards and restrictions. In particular for each route the Report summarized the following:

- Maximum useful mass that can transported (excluding vehicle mass)
- Maximum cargo length
- Maximum cargo height
- Maximum cargo width
- Any community constraints in terms of dust, noise and limitations on travel times
- Any weather and seasonal constraints – especially where roads may be temporarily impassable during the rainy season

18.13.2 Road Restrictions

There are existing restrictions on the transport of plant, equipment and containers on the roads in Madagascar, these include:

- The decree n° 2008-1030 issued on 23rd March 2008 which defines the maximum trucks' technical specifications
- The '61 A' booklet which sets the assumptions to design bridges in Madagascar

The Ministry of Public Works and Meteorology (MTPM) may be approached and are able to authorize exemptions.

The height limit according to the 2000-187 decree is 4 meters. The physical limits are either the height of 'closed' steel bridges, or in the cities, the numerous TELMA phone wires which should be over 5m from the ground. The JIRAMA (the Malagasy Water and Power Company) electrical wires should be over 6m from the ground.

However, the dirt road conditions have to be taken into account as a trailer may overturn if loaded with a high centre of gravity load when it falls into a rut. The load height limit depends on the kind of trailer: the bed of a trailer varies from 0.7m for some low bed trailers, to 1.3m for usual trailers.

We have assumed, according to the experience of some construction companies, that 1.0 m would be the right height.

The weight limit according to the 2008-1080 decree is 38T for a tractor, (or truck) and trailer with 4 axles and 44T for a 5 axle one if the bridges have a no weight limit. Over 44T, the Ministry of Public Works and Meteorology (MTPM) should give an authorization. A tractor and trailer weight is about 14T. Hence the weight load limit without authorization is 30T.

18.13.3 The Road from the Molo Site to Fort Dauphin

The distance from the Molo Site to Fort Dauphin is 339 km according to the road map, 348 km according to the odometer to take 18 hours 22 minutes in total (20.9 km/h speed). The most recent route survey was done by Colas in March 2015. See Figure 104 below:

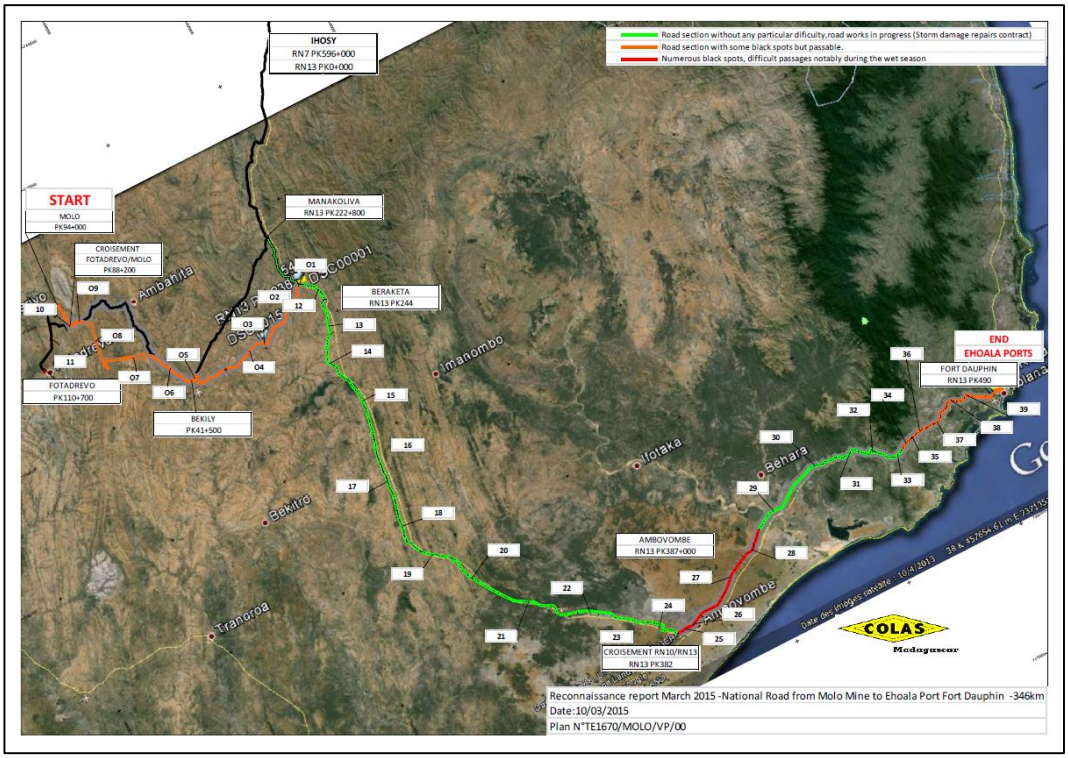

Figure 104: Molo to Fort Dauphin

18.13.4 The Road from Tulear to the Molo Site

The distance from Tulear to the Molo Site is 337 km according to the road map, 331 km according to the odometer to take 9 hours 31 minutes in total (32.3 km/h speed).

Figure 105: Molo to Tulear

The green sections indicate the sections of the Tulear Molo road in good condition, the orange the acceptable sections and the red indicate sections in very poor condition. Figure 106 below shows the Tongobory bride on this route which is restricted to 16 tons gross.

Figure 106: Tongobory Bridge

18.13.5 Inbound and Outbound Logistics

DRA commissioned Bolloré to study the In Bound logistics and a costing based on a project BOQ of:

- Containers
- Break bulk
- Abnormal loads

Tractors and trailers will have to be used to transport containers, reinforced steel, steel beams etc. The trucking companies use as much as possible trucks and trailers as they are more efficient during the rainy season.

Figure 107: Typical Tractor and Trailer Used for Container Haulage

Furthermore the trucks and trailers are raised to have a minimum clearance of 1.20 m in order to be able to navigate the ruts filled with water during the rainy season.

It is recommend having a fleet with both tractors and trailers and trucks and trailers, and the clearance of the trucks is at least 1.20 m high

Dimensions and weight restrictions

According to the 2008-1030 decree, a load is classed as abnormal if:

- Over 2.5 m wide, or
- Over 3.0 m high, or
- Over 12 m long, or
- Over 30 T

If a load is classed as abnormal, an authorization should be obtained from the Ministry of Public Works and Meteorology. A global authorization may be negotiated.

A license must be granted by the Ministry of Defense and the Ministry of Mines to import and sell explosives.

Only three companies have the required license: Madecasse, Crack and SRCD. They may import, customs clear and sell explosives. The major construction companies and gravel pit buy explosives to these 3 companies.

In order to import explosives through these 3 companies, Energizer will have to obtain a prior authorization. The transportation must be organized with a military escort. Explosives must be stored in a powder magazine inside a “Gendarmerie” camp. “Gendarmes” are regular soldiers acting as policemen outside cities. They are similar to the Royal Canadian Mounted Police.

In the case of the Molo site, the explosives will be transported to the Bekily Gendarmerie camp. Explosives for the mining operation will be collected from Bekily as and when they are required. The existing magazine store will need to be extended to accommodate the additional supplies required for the Project. The existing stire has been sized to accommodate the supplies for the existing Red Granite operation only.

The costs associated with the transport of explosives to Bekily is approximately US$4285.00/ton.

There are only four (4) companies who are registered to and may transport fuel. They are:

- Total Madagascar, a subsidiary to the French Total group
- Vivo Energy, a RSA Shell licensee
- Jovenna (Madagascar Company)
- Galana (Madagascar Company)

Against a long term contract (5 years minimum), these companies may invest in the tanks required by Energizer on the Molo site.

Cost estimates for the shipping of containers to Ehoala Port at Fort Dauphin were obtained from CMA CGM, MSC and UAFL. They are the only shipping lines which might call at Ehoala Port.

Table 57: Shipping Freight Costs (US$)

Cost in US $	40 ft	20 ft
New York	8 290	4 545
Shanghai	3 820	1 880
Durban	3 230	1 690
Johannesburg	4 230	2 690
Rotterdam	4074	2042
Container handling at Ehoala Port (empty containers return included)	563.8	281.9
Trucking from Fort Dauphin to Molo	4 224	4 224

Table 58: Terminal Handling Charges at Port of Origin

Cost in US $	40 ft	20 ft
New York	390	390
Shanghai	190	130
Durban	185	125
Johannesburg	185	125
Rotterdam	276.25	276.25

The total cost for the freight and handling of containers from the port of origin to Ehoala Port at Fort Dauphin are detailed below in Table 59. These costs include for the Terminal Handling Charges at the Port of Origin, but does not include the Malagasy VAT, Customs duties and a 0.55% fee on the CIF value of the imported goods. The 20 ft containers have been split into 2 parts, namely over 12 T weight which will be transported one by trailer and less than 12 T weight which will transported two by trailer.

Table 59: Total Sea Freight Costs for Containers offloaded at Ehoala Port

Cost In US $	40 ft	Over 12 T 20 ft	Less than 12 T 20 ft
New York	13 468	9 441	7 329
Shanghai	8 798	6 516	4 404
Durban	8 203	6 321	4 209
Johannesburg	9 203	7 321	5 209
Rotterdam	9 139	6 825	4 713

Cost estimates for the transport of heavy plant and machinery, were obtained for the Molo 2015 FS. This plant and equipment included:

- 1 mining excavator KOMATSU PC 1250 split into 2 pieces (37 T under carriage and 31 T upper structure) ex North America
- Caterpillar 740 Articulated Dump Truck ex Johannesburg (33 T)
- Caterpillar 980 c Wheel Loader ex Johannesburg (28 T)
- Caterpillar 974 D L Hydraulic Excavator ex Johannesburg (71.2 T)

18.13.6 Ehoala Port at Fort Dauphin

The port of Ehoala located at Fort Dauphin was constructed in 2009 and is dedicated to the Rio Tinto ilmenite transport operation. It is managed by the Port of Ehoala Company, which is a subsidiary of the Rio Tinto Group.

The port has a 610m long break water and accommodates 3 berths. The port has the capability to accommodate most commodities. The length of berth 1, 2 and 3 are 275m, 150 and 175m respectively. Berth 3 is dedicated to the port services tug boats.

Figure 108: Ehoala Port, Fort Dauphin

The port site is subject to high winds and container handling operations can be detrimentally affected by the wind in the region. Container handling operations for stuffed containers are stopped if the wind speed increases in excess of 25 knots (13 m/s) and they are stopped for empty containers if the wind speed increases in excess of 30 knots.

The draft at the port was measured at 15.75m in 2009 and it has been assumed that this has not changed significantly in the last 6 years. It has been assumed for the purposes of this study that the draft at Ehoala Port is at least 15.5m.

All vessels calling at Ehoala Port must be geared as there is no cranage at the port to assist with loading and off-loading of the vessels. All handling operations are undertaken by SEPL.

The port has storage facilities for 400 TEU's with an overflow facility for empty containers on the yard close to the port entrance. There are two 1000 m² warehouses at the port with one being dedicated to the sisal exportation market.

18.14 Tailings Disposal and Storage Facility

A TSF is required for storage of the mineral residue, (or tailings) remaining following processing of the ore and extraction of the product at the mine plant. The TSF for the Project is required to accommodate 20.9 million tonnes of tailings over a 26 year period, with an average deposition rate of around 804 000 tonnes / annum. The tailings will be pumped to the TSF as a slurry, with 55 % solids by mass, using pressure displacement pumps.

During the initial stages of the Molo 2015 FS a site selection study was carried out. Six sites were considered and each were scored and ranked according to twelve main assessment categories; covering environmental, social, legal, technical and economic issues. Based on this assessment, a site labelled as TSF 3 was identified as the preferred site and was taken forward for further study to feasibility level. The most important reasons for choosing this site in preference to the other potential sites were:

Close proximity to the proposed plant reduces the capital costs associated with pipelines.

The site is located at the top of a catchment which minimises the amount of storm water flow that needs to be controlled.

No community homesteads are present within the footprint of the preferred site.

Operating costs over the Life of Mine would be relatively low when compared to the other sites.

No issues which would constitute a “fatal flaw” were identified for the preferred site.

The preferred site is located approximately 1 km west of the proposed mine plant. The site is on moderately sloping terrain toward the upper reaches of a minor stream catchment. Development in the area is minor, with scattered subsistence farming and no residential or other structures.

A geotechnical study of the site indicated shallow silty and clayey sand soils overlying weathered metamorphic bedrock at an average depth of 0.3 m. Due to the thin soils, there will be limited construction material for the TSF and consequently a borrow area was identified, approximately 6 km west of the site, containing thick clay, silt and sand soils. Groundwater in the area has generally been found to be shallow, at depths of between 4.5 m and 9.5 m. This may vary seasonally and may be shallower in the vicinity of drainage lines. The seismic hazard in the area is deemed to be low.

A stage capacity assessment model was developed to determine the required final dimensions and optimum development approach for the TSF over time. The model indicated that a 30 Ha Phase 1 site can contain 2.0 years of tailings at the commencement of the project and thereafter a 60 Ha Phase 2 site will need to be commissioned to provide the necessary combined capacity for the remaining Life of Mine.

For the 30 Ha Phase 1 site a starter (containment) wall, with a maximum height of 11 m (variable as on sloping ground) will initially be constructed from borrow materials around the perimeter of the TSF site footprint and from the identified borrow area 4 km west of the site. For the 60 Ha Phase 2 site a maximum starter wall height of 8 m will be required.

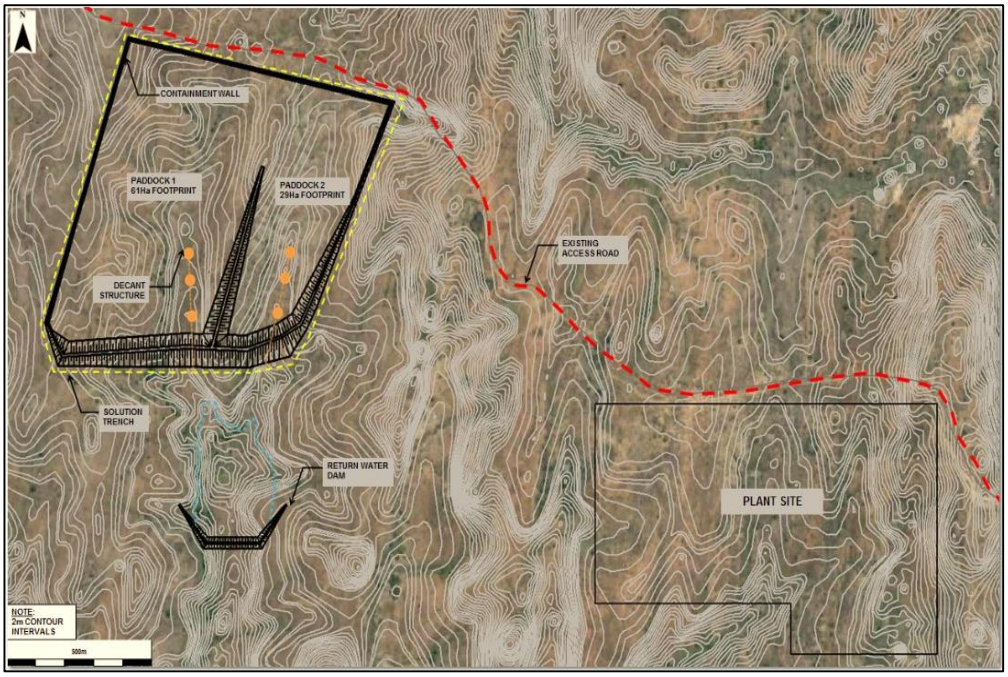

Figure 109: Tailings Dam Layout, Phase I and Phase II

Tailings will be deposited using hydrocyclones connected to the tailings slurry delivery pipelines from the plant. Initially tailings will be deposited behind the starter walls, whilst wall building above the starter walls will be carried out using coarse underflow hydrocyclone tailings, together with wall profiling by earth moving plant. The final maximum wall elevation of the dam will be approximately 39 m.

At all times the perimeter wall around the TSF will be at least 1 m higher than the tailings deposited within the body of the dam. This difference in height (the freeboard) is necessary to ensure that any water collecting in a pool on the surface of the facility, due to tailings slurry (process) water and also a maximum 1 in 100 year 24 hr storm event, can be stored on top of the dam for a sufficient period before the TSF drainage arrangements are able to drain the pool.

The drainage systems at the TSF includes underdrainage to drain water percolating to the base of the dam through the tailings.

Two drainage (penstock) towers, one on the 30 Ha site and one on the 60 Ha site, which will drain water by gravity from the surface of the dam into sub horizontal drainage (decant) pipelines at the base of the facility. The drainage system has been designed to drain a 1 in 50 year, 24 hour duration storm event within 10 days.

Storm water diversion drainage has been allowed for to divert clean water around the facility

Perimeter (solution) drains will collect water reporting from the underdrains and also run-off from the outer walls of the facility.

Geochemical characterisation of the tailings was carried out which indicates that the tailings have the potential to pollute in the long term. Contaminant plume simulation modelling indicates that the contamination plume can be effectively controlled by installing boreholes around the perimeter of the TSF to intercept the plume.

A RWD with a 5 m high downstream wall and an excavated basin up to a maximum depth of 9 m has been allowed for immediately downstream of the TSF, with a storage capacity of around 571 000 m^3. The basin would be lined with a HPDE liner to prevent unnecessary water loss through seepage and potential contamination of the groundwater. The RWD would store water reporting from the decant system draining water from the surface of the TSF, the perimeter (solution) drains, water from pollution control dams elsewhere on the mine and water reporting from the contaminant interceptor boreholes. Stored water will be pumped back to the mine plant.

Stability assessments carried out for the TSF and RWD walls indicate acceptable Factors of Safety above 1.5 under normal operating conditions. Various stability scenarios for abnormal conditions such as seismic loading and ineffective drains gave lower, but acceptable Factors of Safety.

The TSF hazard rating in accordance with South African Standards is medium. In addition, a risk assessment carried out using a Fault Event Tree approach indicated that the probability of injury, loss of life or property damage occurring is within acceptable published norms.

Closure and rehabilitation measures have been proposed to ensure that after closure the facility is both chemically and physically stable, thereby preventing air, ground and water pollution. Measures should include; decommissioning of the decant structures and pipelines, measures to minimise erosion, measures to minimise dust generation and measures to minimise infiltration of rainfall. Additional subsurface and surface drainage will be required to contain the potential contaminant plume and the RWD will need to remain.

Initial capital costs to develop the 30 Ha Phase 1 site (including the RWD) would be approximately US $15 033 700 with a further US $3 895 000 required to develop the 60 Ha Phase 2 site, totalling an estimated US $18 928 700 capital costs for the full 90 Ha facility and RWD complex. Preliminary and general costs have been omitted, and it is estimated that these will be around 65 % of the measured work. The operational costs are estimated to be approximately US $422 840 per annum based on a production rate of 804 000 tonnes per annum.

18.15 Geotechnical Report Assessment and Founding Strategy

SRK Consulting (South Africa) (Pty) Ltd (SRK) was appointed by DRA on behalf of Energizer to conduct a geotechnical investigation for the revised plant site and the camp site for the Molo Graphite Project. 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 60: 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 client 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

Energizer engaged Roskill to compile a report on markets for natural graphite up to 2020 which report was completed in January 2015. This market summary is to a large extent based on the Roskill report.

It is confirmed that the Roskill report aligns with general industry sentiment and serves as a sound forecasting basis. Market conditions improved in 2014 and the trend is predicted to continue albeit in tandem with Chinese influence related to a major industry re-organisation.

19.1 Market Demand

Graphite consumption comprises three different product lines, namely synthetic graphite, natural amorphous graphite and natural flake graphite.

Price is often the major deciding factor in choosing between natural flake and synthetic, although each also has specific characteristics which need to be considered for a particular application. For example in the production of lithium-ion anode materials, natural flake graphite may be chosen due to price although synthetic graphite may be more suitable in some formulations.

In those applications where they compete, synthetic graphite prices are higher than natural graphite prices due essentially to higher production costs. This is somewhat offset by the purification cost to raise most natural graphites to sufficient purity. It is estimated that in 2013, the difference between comparable synthetic and natural grades was US$1,000 per tonne.

Amorphous graphite is used in such applications as the refractories industry, as recarburisers, in brake linings, gaskets and clutch materials and in foundries in mould wash.

The 2013 estimated world market for natural and synthetic graphite is depicted in

Table 61 below.

Table 61: World Estimated Consumption of Natural and Synthetic Graphite 2013 (1 000 tonne)

Category	Natural Graphite	(of which Natural Flake Graphite)	Synthetic Graphite	Total
Electrodes	-	-	860	860
Refractories	511	335	-	511
Lubricants	50	12	100	150
Foundries	133	80	-	133
Graphite shapes[1]	12	1-	105	117
Batteries	74	74	27	101
Friction Products	53	22	-	53
Others	135[2]	53	520[3]	655
Total	**968**	**586**	**1,612**	**2,580**
Source: Roskill estimates				

Notes:

1 Including carbon brushes.

2 Including 35,000t of amorphous graphite in decarburising.

3 Mainly consumption in re-carburisers, but also in foundries, friction materials and refractories.

Natural flake graphite is consumed in refractories, foundry applications, batteries, as battery additives, in fuel cells, friction products, lubricants, shapes and expandable graphite.

19.1.1 Refractories

The largest consumer of natural flake graphite is the refractory industry which produces products consumed in steelmaking, non-ferrous metals, and a wide range of other high-temperature industrial processes such as cement and glass. The estimated world production of refractories by country for the period 2002 to 2013 is depicted in Figure 110.

Refractory production is to a large extent dependant on crude steel production, taking into account improved quality and the trend to continuous casting.

Consumption of refractories has failed to keep pace with rising steel, glass and cement production. The reason for this is that advances in technology and greater

use of higher grade materials have reduced the specific consumption of refractories in all industries. It is estimated that for the period from 1980 to 2014 the global specific consumption in refractories use in the steel, cement and glass industries, for example, fell from 30 to 15 kg/tonne, from 1.2 to 0.6 kg/tonne and from 12 to 5 kg/tonne respectively. Clearly this reduction has been offset by the increase in volumes of the products over the time period.

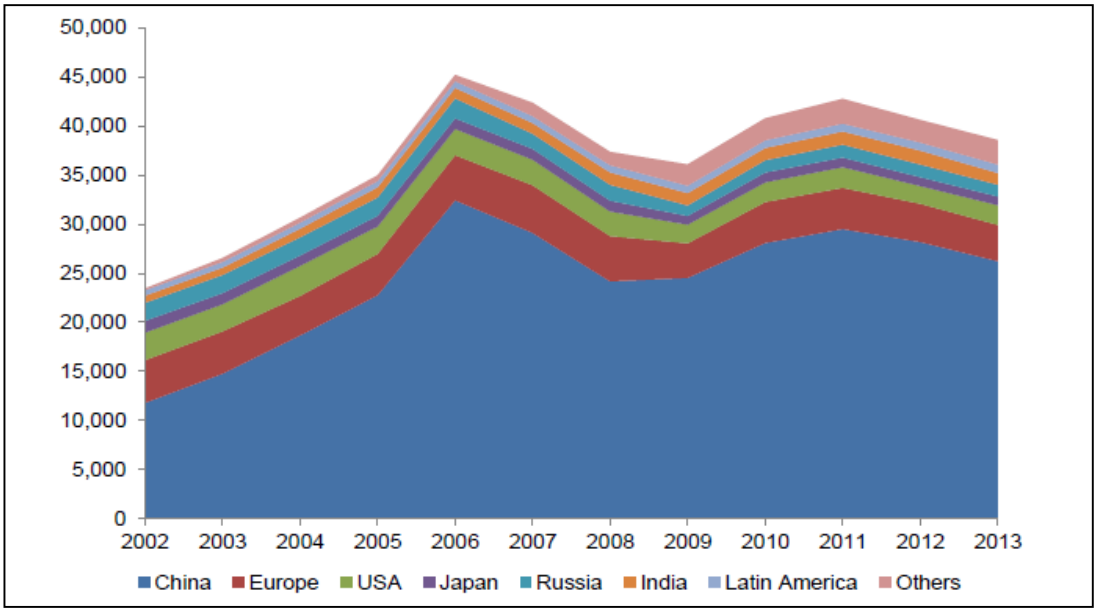

Figure 110: Estimated World Production of Refractories by Country, 2002 to 2013 (1 000 tonnes)

Source: Roskill.

Figure 111 depicts the world production of crude steel for the period 2004 to 2013.

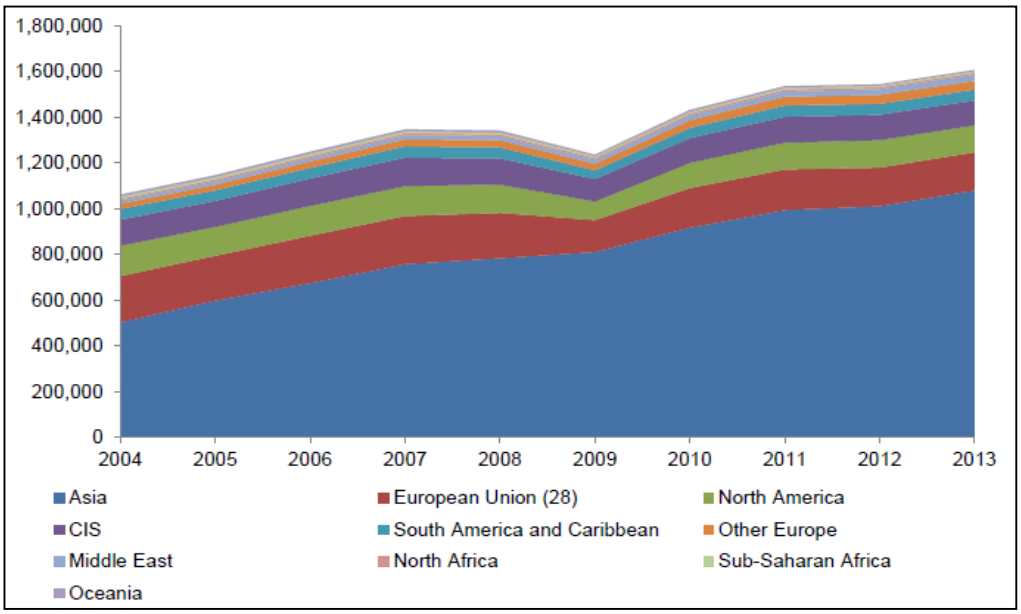

Figure 111: World Production of Crude Steel by Region 2004 to 2013 (1 000 tonne)

Source: World Steel Association.

It is estimated that of the 511 000 tonnes of natural graphite consumed in refractories in 2013, 335 000 tonnes comprised natural flake graphite (some 57% of total flake demand) and 176 000 tonnes amorphous graphite. It is also estimated that the Chinese refractory industry consumed 200 000 tonnes of natural graphite in the same year.

Table 62: World: Forecast Steel Production by Region 2015 to 2020 (1 000 tonnes)

	2015	2016	2017	2018	2019	2020	CAGR
Asia	1,171,445	1,218,040	1,261,210	1,299,781	1,333,130	1,425,817	3.7%
EU-28	173,091	174,812	176,739	178,896	181,327	165,084	-0.1%
N. America	124,200	127,096	130,059	133,090	136,191	144,686	3.0%
CIS	115,114	117,823	120,598	123,441	126,355	125,915	2.1%
S. America	49,446	49,701	49,994	50,264	50,536	66,240	4.9%
Other Europe	41,598	43,022	44,317	45,462	46,502	43,740	1.8%
Middle East	29,326	30,987	32,779	34,656	36,613	36,953	5.0%
North Africa	9,023	9,152	9,282	9,429	9,586	11,255	3.9%
Other Africa	7,465	7,500	7,539	7,583	7,633	8,839	2.7%
Oceania	5,696	5,709	5,738	5,782	5,840	6,186	1.6%
Total	1,726,403	1,783,841	1,838,255	1,888,384	1,933,712	1,972,444	3.0%

Source: Roskill.

The refractories sector is consistently the largest volume market sector for large/jumbo flake graphite. While crude steel production is expected to grow by 3% per annum to 2020 (see

Table 62) flake graphite consumption is expected to grow 3.9% per annum over the same period as a result of the trend towards higher quality refractory production such as in magnesia-carbon and alumina-magnesia-carbon refractories.

19.1.2 Foundries

Traditionally, low quality amorphous natural graphite grading 40 to 70% carbon has been used in this application in pure form or mixed. Fine flake graphite of 50 to 75 µm size can also be used. The 80,000 tonne of natural flake graphite used in this application in 2013 was all fine flake material.

The market for natural flake graphite in this market sector is expected to grow by 1.2% per annum to 2020. Secondary synthetic graphite is expected to continue to replace more expensive natural flake graphite in this application.

Most of the current demand for natural flake graphite in foundries is in China where flake graphite is widely available. Growth for natural flake graphite will mostly be in China, and to a lesser degree Brazil.

19.1.3 Batteries

The use of natural flake graphite in batteries is an important market in which flake graphite competes directly with synthetic graphite, the decision being based largely on price.

The batteries market includes the manufacture of anodes for lithium-ion batteries and the use of graphite as an additive in other battery types. The market for rechargeable lithium batteries includes:

- Computing, communication and consumer (3C) products – used in small devices that are portable or used in the home such as in mobile phones, laptops, MP3 players, etc.
- Power devices and small motive power tools and electric bicycles
- Heavy duty – on and off grid energy storage; and
- Transportation – electric vehicles (EVs) and hybrid electric vehicles (HEVs), buses and trains

An estimated 60,000 tonnes per annum of natural flake graphite is used in anodes for lithium-ion batteries (the vast majority in the production of approximately 25,000 tonnes per annum of spherical graphite) compared with 20,000t of synthetic graphite. Specifications for the grades of graphite used in batteries are tight and once a supplier has been selected, after rigorous tests have been conducted, that supplier is required to adhere strictly to the specifications. China is the largest manufacturer of anode material for lithium-ion batteries.

Over the next five years the main driver for graphite demand in batteries will be growth in production of anodes for lithium-ion batteries for use in 3C, heavy duty,

power and motive and transport applications. In terms of cells, growth is expected to be 9 to 10% per annum but, in terms of capacity, 21 to 22% per annum. Overall graphite consumption in batteries is forecast to reach 220,000 tonnes by 2020, of which 148,000 tonnes is forecast to be natural flake graphite. Growth for flake graphite in this application will thus average 10.4% per annum. The expected growth in production of EVs and HEVs significantly benefits the consumption of natural flake graphite to 2020.

About 20,000 tonnes per annum of flake graphite is used as a minor additive in other battery types, mainly lead acid and alkaline batteries.

19.1.4 Other Uses

While some 83 to 84% of natural flake graphite is consumed in refractories, foundries and batteries, the balance of 16 to 17% is consumed in fuel cells, friction products, lubricants, shapes, expandable graphite and other applications. Consumption to 2020 is expected to remain at this proportion of the market.

19.1.5 Summary: Market Outlook for Natural Flake Graphite

Table 63 is an estimate of the world natural flake graphite market in 2013 while Table 64 summarises the forecast for the natural flake graphite market worldwide to 2020, by region. Growth for natural flake graphite will average 4.7% per annum to 2020 with consumption expected to reach 808 000 tonnes. Asian markets will continue to dominate consumption and by 2020 are forecast to represent over 70% of natural flake graphite consumption.

Table 63: World: Estimated Consumption of Natural Flake Graphite 2013 (1 000 tonne)

	Asia	Europe	N. America	S. America	Others	Total
Refractories	250	40	20	17	8	335
Foundries	28	25	20	2	5	80
Batteries	62	3	3	4	2	74
Friction Products	11	6	3	1	1	22
Lubricants	7	2	1	1	1	12
Graphite Shapes	5	2	3	-	-	10
Others	28	7	6	12	-	53
Total	391	85	56	37	17	586

Source: Roskill estimates

Table 64: World: Estimated Consumption of Natural Flake Graphite 2020 (1 000 tonne)

	Asia	Europe	N. America	S. America	Others	Total
Refractories	330	45	23	27	12	437
Batteries	136	3	3	4	2	148
Foundries	34	26	20	2	5	87
Friction Products	17	8	3	1	1	30
Graphite Shapes	10	3	4	1	1	19
Lubricants	7	2	1	1	1	12
Others	38	10	11	14	2	75
Total	572	97	65	50	24	808

Source: Roskill estimates

19.2 Supply

In 2013, production of natural flake graphite totalled 427,300 tonnes. Output of flake graphite reached a peak of some 500 000 tonnes in 2012, of which 60% originated in China. Other significant producers of flake graphite are Brazil, India, Canada and Norway. Figure 112 illustrates production of flake graphite worldwide and the dominance of Asian, more specifically Chinese, production.

Chinese flake graphite production is currently fragmented and includes a significant number of small operations with 10 000 tonne per annum capacity or less. A process of consolidation is underway, which began in Inner Mongolia during 2010 and started in Shandong and Heilongjiang during 2014. This will create new industry giants in the country situated in these three centres.

Several foreign companies have invested in China, in order to secure supplies; these include Imerys of France and AMG of the Netherlands. Many existing Chinese mines are coming to the end of their working lives and a number outside China have become exhausted in recent years.

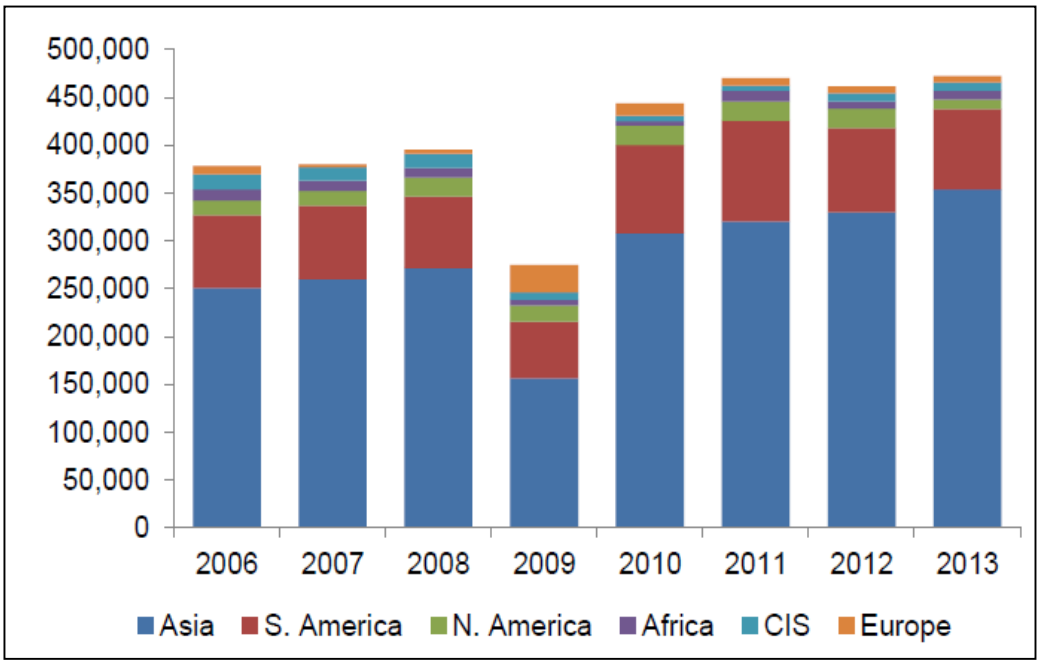

Figure 112: World Natural Flake Graphite Production 2006-2013 (tonne)

Sources: USGS, BGS, DNPM, InfoMine Research Group and Roskill

A number of new flake graphite projects that are under development will increase capacity outside China in the coming years. In a recent development a Canadian graphite project announced a significant off take agreement with a Chinese industrial conglomerate of 40,000 tonne, suggesting that security of supply is becoming increasingly important to manufacturers in China, as well as in the rest of the world.

Apart from China, capacity is concentrated in Brazil and India but is also present in a number of other countries. Of these, the leading producer is Nacional de Grafite of Brazil, which has at least 75,000 tonnes per annum of capacity.

19.2.1 Purification of Natural Flake Graphite

In order to achieve grades of graphite which comply with a definition of high-purity (>99% C), natural flake graphite has to undergo initial beneficiation which can take the form of repeated cycles of grinding and flotation. Some high-purity applications require further thermal and/or chemical treatment to achieve the grades required for batteries, fuel cells, as well as some of the grades required for lubricants, carbon brushes and powder metallurgy.

The further purification of natural flake graphite (including flake graphite fines) results in grades with carbon contents from 99.5% to 99.99% C. Purification methods include thermal treatment and/or chemical leaching. Acid treatment produces the highest grade products for use in lithium-ion batteries whereas

alkaline leaching produces slightly lower purity grades suitable for use in alkaline batteries. The adverse environmental impact of leaching, particularly with hydrofluoric acid, has led to chemical leaching taking place predominantly in China. In other regions thermal processing is the main process. This is important because since the boom in natural flake graphite, it has proved challenging for new high-purity flake graphite producers outside China to secure off take agreements with battery manufacturers (unless they join the purification supply chain in China with their product).

Flake graphite can also be treated to produce expanded graphite, which has a high carbon content and is suitable for use in the growing markets of fuel cells, thermal management and fireproofing.

19.2.2 Synthetic Graphite

The other side to the graphite market is synthetic graphite with Asia as the main source, accounting for an estimated 45% of production. Synthetic graphite is not strictly a commodity as it is synthesised from calcined coke for particular applications. Over 54% of synthetic graphite is formed during the manufacture of electrodes; other applications include carbon shapes and batteries. Production of electrodes is growing in China, Russia and India and manufacturers in these markets are competing with more established producers in North America and Europe.

19.2.3 Outlook for Graphite Production

Natural graphite production is forecast to grow by 5.4% per annum in the years to 2020 as growing demand for flake graphite drives expansions and new product development. This estimate includes established projects realising production on their announced timeline. A total of 188,500t of new capacity could come on stream by 2020, however, more realistically this total will be in the region of 100,000t, which would provide a lower CAGR of 3% per annum. This assumes no increase in production in China, due to on-going consolidation. Production of high-purity natural graphite will continue to be concentrated in China.

There is currently significant overcapacity in the synthetic graphite market as well as increasing competition from new plants in China, India and Russia. Producers will be able to increase supply to meet likely increases in demand.

A large amount of graphite exploration has been carried out over the last five years as concerns grew over a potential future tightness/shortage of supply. Development of new capacity is focused on high grade, large flake deposits, driven by growing demand from both the traditional markets of refractories and brake linings, and from the emerging, and rapidly growing, markets of lithium-ion batteries and expandable / expanded graphite.

By 2014, many of the prospective new projects had seen little progress and only a handful have reached pre-feasibility.

19.3 Pricing

Broadly speaking graphite prices increase with flake size and carbon content. In practice other factors come into play and the price will depend on the location of the supplier and purchaser and the logistics involved, the agreed contract type and length, graphite specifications as well as underlying production and processing costs.

Published flake graphite prices for fine, medium and large flake sizes are shown in Figure 113 average of all carbon grades) illustrating the price premium by size. After the price hike in 2011, and decline in 2012 to 2013, prices have stabilised in 2014.

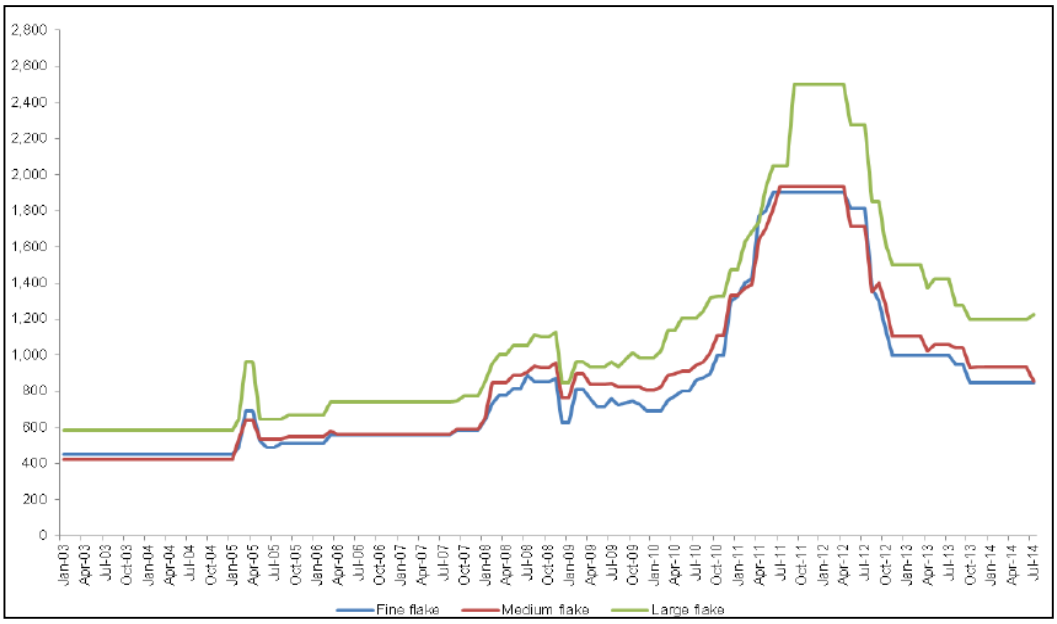

Figure 113: Average Prices of Flake Graphite by Flake Size, 2003 to 2014

Source: Industrial Minerals, Asian Metal, Industry sources.

Notes:

Average of all published carbon grades, FCL CIF main European port.

Fine flake +200 mesh, Medium flake +100 mesh, Large flake +80 mesh

Jumbo grades (regarded as plus 50 mesh or 300µm) are priced at a premium to large grades. In July 2014, jumbo grades were priced at $1,800-2,000/t FOB China for 94-97% C graphite, representing a 50-70% price premium.

Prices prevailing in Europe during the third quarter of 2014 are depicted in Table 65

Table 65: Average European Prices Q3 2014 (converted to US$)

Category	Min % C	CIF Rotterdam
Powder -200	90-94	650-750
Fine flake -100 mesh	90-94	750-850
	94/97	1,000-1,150
Medium flake +100 mesh -80	90-94	1,050-1,100
	94-97	1,100-1,300
Large flake +80 mesh	90-94	1,100-1,250
	94-97	1,350-1,450
Jumbo flake +50 mesh	90-94	1,750-1,950
	94-97	1,800-2,000
	98-99	2,200-2,600

Source: Industrial Minerals, IM Data, Roskill.

19.3.1 Short to Medium Term Price Outlook

Overall natural flake graphite prices are expected to recover in line with, and above, economic activity. The level of price recovery overall will depend on degree of consolidation in the Chinese flake graphite industry and its impact and the recovery and production levels in the steel industry.

Mining projects are commonly evaluated using two or three year trailing averages. However, the significant price spike in 2011 and 2012 distorts the picture such that historic averages are not representative. The subsequent fall of graphite prices means that the historic averages are higher than current prices. Prices have been relatively stable during 2014 and have now started to see some upward pressure. It is therefore assumed that this represents the bottom of the market.

The conclusion that this is the bottom of the market also takes into account the consolidation of the Chinese natural flake graphite sector over the next two to three years, which may constrict supply, the closure of a North American mine in 2013, and the forecast growth in demand. These combined factors should eliminate any further downside in prices from the present levels, and present opportunities for further growth. Capital and operation costs are also rising, which in the medium to longer term will eliminate more marginal producers, and keep the outlook for graphite prices healthy.

For medium and large flake sized material, new supply is not expected on stream in 2015 to step into any shortfalls from Chinese production restrictions, which could put some upward pressure on pricing levels, especially in Europe. This could cause short term fluctuations of $200-300 per tonne around the average growth rates, or even as much as $400 per tonne for jumbo flakes. In mid-2016 new scheduled production from Mozambique having a significant proportion of larger flake, could replace a portion of Chinese material. From 2017 onwards the effects of the consolidation process are expected to be largely complete and the Chinese industry reorganised into much larger enterprises.

For fine flake material of 90-96%, graphite prices are expected to rise just above economic activity, especially in refractories, foundries and crucibles markets which represent most of the volume. This material is also not expected to see quite the same upward price pressure in 2015 and 2016.

Chinese FOB flake graphite prices are expected to increase with production costs and supply restrictions within the country, especially for larger flake sizes in 2015 and potentially in 2016. This will maintain or even raise price levels during a period when there is predicted slowdown in growth in the Chinese industry and household purchasing index.

Figure 114 graphically displays the price forecasts for various flake sizes.

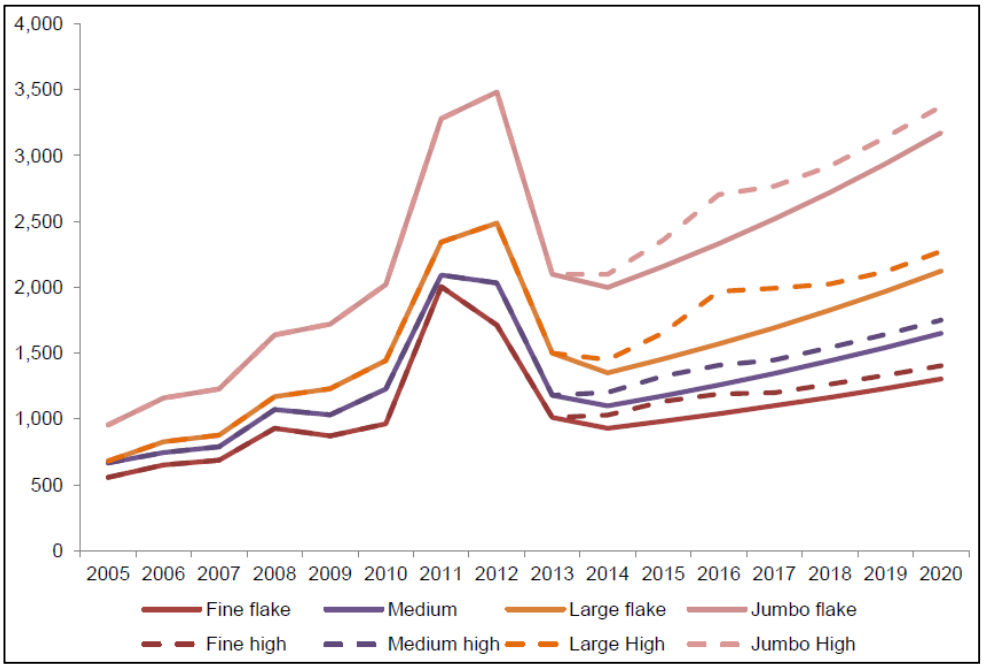

Figure 114: Price Forecast to 2020 with Base and High Case Scenarios 94-97% C, CIF Europe

Source: Roskill, Industrial Minerals, company sources

19.4 Energizer's Approach

Energizer will concentrate its marketing efforts on the back of its inherent strengths, amongst which are:

- Range of large and jumbo crystalline flake sizes
- High grade deposit with a low strip ratio resulting in cost effective extraction
- Being a non-Chinese secure supply
- Low impurities largely confined to finest material (-400mesh, 93.7% C)
- High grade concentrate produced by simple flotation (>97%)
- Large resource
- Favourable flake size distribution
- Strategic positioning between European and Asian markets, including India, Japan and South Korea; and
- Proven technology to produce high grade and high purity concentrates

Being aware of a number of challenges, including logistic hurdles and lack of track record, the approach has been that during the first year of production the company will target customers matched to the expected grade and quality of production, but will expect to sell near the bottom end of the spot market. As its reputation grows it anticipates being able to move prices to within the spot price range but conservatively forecasts that it will sell at a price pitched below the average of the forecast range. Table 66 indicates the predicted sales volumes and prices used in the Economic Analysis section of this Report.

19.5 Off-take Agreements

It is required that Energizer pursues the execution of suitable off-take agreements either with final consumers or with intermediary beneficiators. This process is yet to commence in earnest.

Table 66: Forecast Volumes and Prices by Flake Size 2017 to 2027

Flake Size	Grade		2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
Jumbo (+50 mesh)	96.9%	Volume (1 000 Tons conc.)	12.83	13.54	14.39	13.74	11.40	12.46	12.37	12.47	12.50	12.49	12.35
		Price, C&F Eur. (USD/Tonne)	2519	2801	3019	3254	3508	3782	4078	4398	4739	4834	4931
Large (+80 mesh)	97.1%	Volume (1 000 Tonnes conc.)	12.45	13.14	13.96	13.33	11.06	12.09	12.00	12.10	12.13	12.12	11.98
		Price, C&F Eur. (USD/Tonne)	1929	2026	2120	2274	2441	2821	2815	3024	3250	3315	3381
Medium (+100 mesh)	97.2%	Volume (1 000 Tonnes conc.)	3.77	3.98	4.22	4.03	3.35	3.66	3.63	3.66	3.67	3.67	3.63
		Price, C&F Eur. (USD/Tonne)	1440	1542	1643	1751	1866	1990	2122	2264	2415	2451	2489
Fine (-100 mesh)	97.6%	Volume (1 000 Tonnes conc.)	25.55	26.97	28.65	27.36	22.70	24.82	24.63	24.83	24.90	24.88	24.60
		Price, C&F Eur. (USD/Tonne)	1191	1265	1373	1404	1480	1560	1645	1734	1829	1847	1866

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY 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 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 Molo Graphite Project is substantial. The table below provides a summary of the information which supplements this Molo 2015 FS.

Table 67: 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 Energizer and the O.N.E. (regulator)	Agetipa, GCS & DRA	2014
Environmental and Social Impact Assessment and Management Plan (ESIA & ESMP) including residual baseline	GCS and Agetipa	2014
Relocation Action Plan (RAP)	Agetipa	2014
Conceptual Closure Plan	GCS	2014
Permitting and Stakeholder Register	Agetipa	2014

20.2 Applicable Laws and Standards

This study 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. There are no current environmental issues that could materially impact Energizer's ability to extract the mineral resources and mineral reserves.

Various options were considered at commencement of the Molo 2015 FS regarding the geographical placement of infrastructure, type and volume of water supply, type of electricity supply, TSF, (location and deposition strategy) and product transportation routes and shipping locations, specifically related to environmental and social benefits. Refer to Figure 115 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 study:

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. Further reference and information can be found in section 18.1.

20.3.2 Water Supply

The base case is groundwater supply via a dedicated well field, with the alternative being a water supply dam. Groundwater was eventually selected as the preferred water supply option due to the following:

- Lowest impact on existing ecological and human needs
- Most feasible economic option
- Technically proven sustainable yield

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

The base case is thickened tailings.

Originally the tailings deposition strategy considered an un-thickened design, but this design has a high make-up water requirement, which would have placed significantly larger raw water supply requirements on the project. Subsequently, after extensive groundwater yield investigations indicated that groundwater could be a viable source of water for the project, the decision was reached that a thickened tailings deposition strategy would lead to higher water recoveries within the process, (thus more re-use of water), thereby reducing the make-up water requirement and hence using less water. This proved to be equally viable from an environmental, technical and economic perspective. Detailed geochemical analysis of long term contamination risk proved that reaction rates are slow enough to allow for practical concurrent management measures which will ensure that the TSF will not contaminate any receiving water bodies or existing water resources. This effectively removed the requirement for a liner of the TSF and places a responsibility on effective concurrent rehabilitation of the TSF, as well as capping at mine closure.

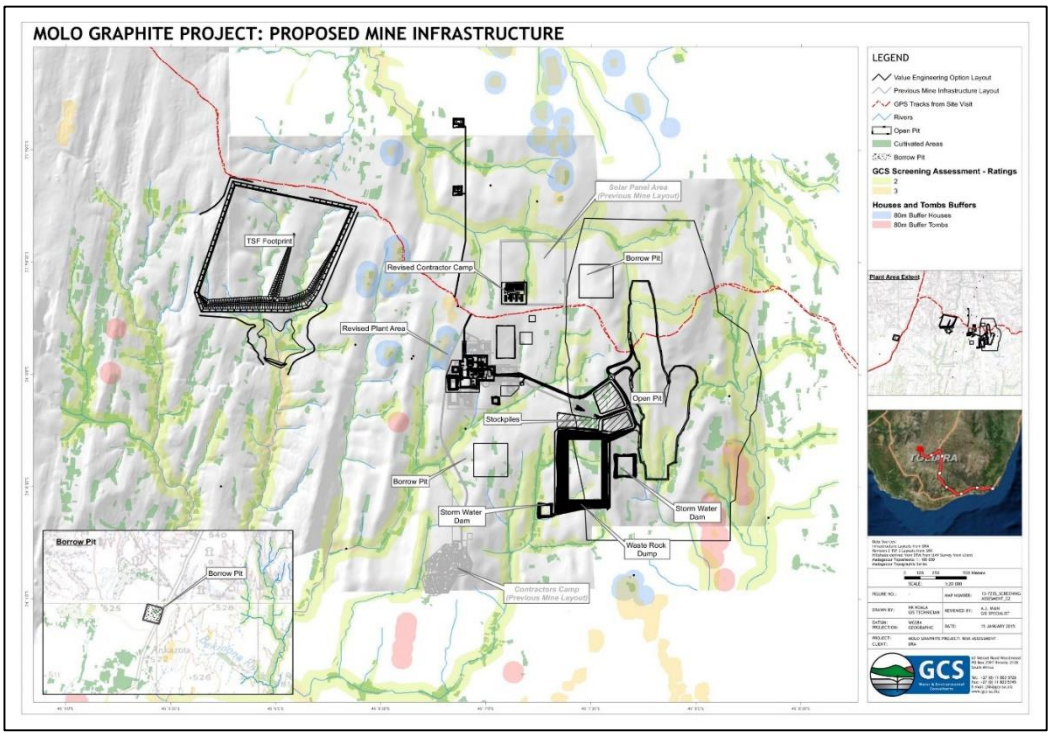

Figure 115: Infrastructure Overlaid on Environmental and Social Sensitivities

20.3.5 Transport Route and Shipping Port

The base case is Fort Dauphin via the RN10 and/or RN13 and the alternative is Tulear and Antseraky. Refer to Figure 2.

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.

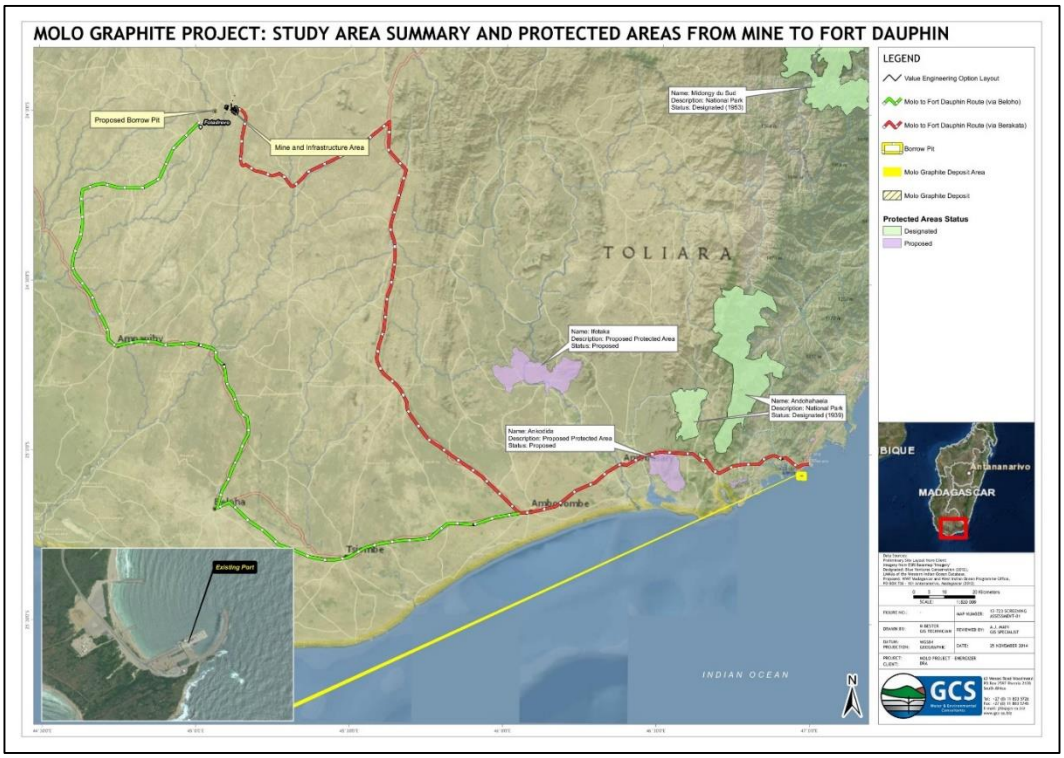

Figure 116: Transport Route Alternatives to Fort Dauphin

20.4 Environmental and Social Impact Assessment and Management Plans

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.

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 development phase. This is aimed at monitoring compliance against various specifications such as the baseline environment and predicted impact removal and reduction measures.

Post mitigation impacts are summarised as follows:

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.

20.4.2 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 TSF, which will have a visual change impact. The proposed mitigation measures are aimed at concurrent rehabilitation of the 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.

20.4.3 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 TSF, which will have a visual change impact. The proposed mitigation measures are to continue with the concurrent rehabilitation of the TSF throughout the remaining life of the facility. 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.

20.4.4 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 to not exist at present.

From a conceptual closure perspective the open pit will be left as a void upon closure. Difficulties could be experience by concurrent in-filling in those cases where the ore body is limited to a single opencast pit, as is the case with the Molo Graphite 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:

- In this case the opencast 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 measure 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 TSF, 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 TSF 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.

20.5 Waste and Water Management

20.5.1 Mine Waste Assessment

A detailed 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 methods such concurrent rehabilitation and final capping of facilities is specified to manage the risk of long-term post closure water quality emanating from residual waste product.

20.5.2 Surface Water Environment

The Molo Graphite Project site is located on the divide of three major river basins: Onilahy River Basin (32.3 km^2 catchment), Linta River Basin (6.2 km^2 catchment) and Menarandra River Basin (8.7 km^2 catchment). The calculated mean annual run-off is about 150 mm per annum, or roughly 18.8 % of the mean annual precipitation (MAP = 799 mm).

Around the Molo Graphite 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.

20.5.3 Groundwater Environment

The Molo Graphite Project site site falls within the Ampanihy shear zone with a regional north-northeast–south-south-western 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 to medium aquifer potential, with the marble layers to the west having a higher aquifer potential.

Community shallow hand-dug wells and boreholes are sparsely scattered over the area. Groundwater use is low, 15 to 20 cubic metres per day in total.

Sodium-bicarbonate and calcium (magnesium) bicarbonate dominant waters were present at the site, with subordinate sodium-sulphate (chloride) dominant waters. The total dissolved solids content (TDS) of the groundwater is typically 150 to 550 mg/l and pH levels typically range from 7 to 8.5. However, a number of groundwater points have higher TDS values, up to 2860 mg/l. The majority of constituents

analysed for were in compliance with all three standards/guidelines, with exception of the following:

- Elevated TDS levels, in terms of WHO Guidelines for Drinking Water Quality(>1,000 mg/l TDS), were observed in five (5) samples, typically associated with elevated chloride and sodium concentrations (GWH 20, GWH 21, MMB1 and MMB8). Elevated Na and Cl in water do not necessarily pose a health risk to humans, but water may have a displeasing taste
- Elevated sulphate concentrations, in terms of Decret No. 2004-635 Modification 2003-941 Surveillance de l'eau Normes eau Potable and WHO Guidelines for Drinking Water Quality (i.e. 250 mg/l SO4), were observed in three (3) samples (MMB1, MMB4 and MMB8)
- Slightly elevated iron concentrations, in terms of Decret No. 2004-635 Modification 2003-941 Surveillance de l'eau Normes eau Potable and WHO Guidelines for Drinking Water Quality, and/or manganese and/or zinc concentrations, in terms of WHO Guidelines for Drinking Water Quality, are observed in some boreholes (GWH1, GWH2, GWH7, MMB1, MMB2, MMB4, MMB8, MMB9 and MMB15). These are aesthetic parameters, however and do not pose a health risk to humans

20.5.4 Water Management

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 with different groundwater and surface water models and a significance rating was assigned.

Potential impacts relate to mine dewatering, wellfield abstraction, dirty water run-off, run-off reduction, contaminant seepage from the TSF and Waste Rock Dump ("WRD"). 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 and TSF scavenger wells.

20.6 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 inhabitants within close proximity to the project site, for which a Relocation Action Plan ("RAP") has been developed.

20.7 Mine Rehabilitation and Closure

The development of a Mine Closure and Rehabilitation Programme ("MCRP") 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 MCRP specifically for the proposed Molo Graphite 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 conceptual MCRP 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 68 below for the closure estimates, based at end of LOM at present time value of money.

Table 68: Closure and Rehabilitation Cost Estimates

Description	Base Case (Project with Thickened Tailings Disposition)	
	Sub Total	Total Life of Mine
	Owner Operator Rehabilitation	Third Party Rehabilitation
Plant Area and Associated Infrastructure	$616 010.50	$758 924.93
Accommodation Camp	$739 056.06	$910 517.07
Opencast	$558 944.50	$688 619.63
Waste Rock Dump	$639 426.39	$787 773.31
Tailings Storage Facility	$1 727 587.64	$2 128 387.98
Explosive Magazine	$35 042.21	$43 172.00
Roads and Pipeline	$80 977.29	$99 764.03

Description	Base Case (Project with Thickened Tailings Disposition)	
	Sub Total	Total Life of Mine
	Owner Operator Rehabilitation	Third Party Rehabilitation
Water Infrastructure / Water Supply	$84 907.72	$104 606.31
Sub Total	$4 481 952.32	$5 521 765.25

20.8 Permitting and Stakeholders

Energizer is in the process of renewing the exploration right over the properties in question with the eventual aim to convert these to exploitation rights through a process of feasibility studies, environmental authorisations and other permit and license approvals. As input into the Molo 2015 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.

Currently, Energizer has progressed significantly in its application for a Global Environmental Permit, the submission of is a pre-requisite to the application for an Exploitation Permit. The Global Environmental Permit application is supported by the ESIA and RAP, which will be submitted to the O.N.E. (National Environment Office) upon approval of the investment amount by the Ministry of Strategic Resources. The processing of the ESIA and RAP will inform the requirement for further sectorial EMP's (small scale reports) which will focus on certain aspects of the project's influence on environmental and social matters. These sectorial EMP's, along with other permits (listed below) will be undertaken during the early project execution phase. Additionally, a detailed process of negotiations will be undertaken to obtain final approvals for the relocation of the Soarerana Village.

The proposed Molo Graphite Project has triggered the following list of permits / licenses in addition to that which is described above, (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 Introduction

The capital and operating costs for the Project were compiled by DRA. The base date for these costs is December 2014.

21.2 Capex

21.2.1 Capex Summary

The capital costs of the Project are estimated as follows. These are the costs to construct the project. See Table 69 below.

Table 69: Capital Cost Summary

Description	US $ millions
Capital Cost	149.9
Design Development Allowance	13.8
Subtotal	163.7
Contingency	24.5
Total	188.2*
*Excludes taxes, tariffs, duties and interest	

21.2.2 Basis of Estimate

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

- Process design completed based on test work and an agreed plant capacity – in this case the ability to process 862 500 tonnes / annum of ROM ore
- The process design is then reduced to set of Process Flow Diagrams (“PFD’s”) and Mechanical Equipment List (“MEL”) where all the equipment is specified and sized.
- The MEL was then priced in the market place on competitive enquiry
- A set of drawings were developed for the process plant
- Quantities were measured off these drawings for earthworks, civils, steel plate work, and pipework

- Bills of Quantities were compiled and tendered in the market place for the following major packages:
  - Earthworks
  - Civils
  - Structural, Mechanical, Plate work
  - Piping was measured and estimated from the DRA data base
- EPCM costs were built up in detail based on the scope and time frame of the project
- Accommodation was priced competitively

The design development allowance provides a pool of money to address possible changes in quantities and rates as the project progresses into execution. This money accounts for the fact that only early engineering has been completed and as the project progresses into detailed design certain quantities and rates may change.

The contingency provides a pool of money to cover the owner against unpredictable project expenses. For example weather related delays, unforeseen ground conditions, labour disruptions, and logistics delays

The capital costs are broken down by discipline as follows. See

Table 70 below.

Table 70: Capital Cost by Discipline

Cost Centres	US $ millions
Pre-production	37.3
Tailings Storage Facility	24.3
Mechanicals	20.8
Electrical, Control & Instrumentation	20.8
External services	17.9
Earthworks	11.8
Piping	7.4
Structural	5.6
Transport	5.5
Vendor packages	3.4
Civil works	2.5
Consumables and spares	2.4
Buildings, fittings	2.1
Plate work	1.9
Total Capital Costs	163.7

As pre-production is a significant cost element a further breakdown of these costs is shown in

Table 71 below.

Table 71: Pre-production – Key Cost Elements (US $ million)

Category	Cost
Facilities management during construction	6.4
Mining fleet purchase	6.0
Owners team	3.8
Buildings	3.6
Fuel (supplied by energizer to all contractors)	2.8
Design development allowance	2.6
Vehicles	2.0
Furniture	1.4
Skilled labour – permanent accommodation	1.2
Plant commissioning assistance and ramp-up	1.1
Unskilled labour – permanent accommodation	0.8
Raw water (well field)	0.7
Tower crane	0.6
Crop and land compensation	0.5
Insurance	0.5
Manager accommodation - permanent	0.5
Medical	0.6
Security during construction	0.4
Mess haul and kitchen (complete) - permanent	0.4
Furnishing & equipment	0.4
Other costs	1
Total Pre-production	37.3

Note – Costs are rounded.

The capital costs of the Project are also broken down by area. Refer to Table 72 below.

Table 72: Capital Cost by Area

Area	US $ Millions
Mining equipment	6.0
Process plant	35.6
Infrastructure (Mine & Plant)	26.9
Electrical Infrastructure (Mine & Plant)	15.3
Accommodation (Permanent)	6.7
Accommodation & Feeding (Construction)	11.6
Project Services	21.1
Construction Services	5.6
Indirect Costs	4.8
Plant & Infrastructure – Preliminary and General	16.3
Design Development Allowance	13.8
Total capital costs	163.7

Future capital expenditures expected to be incurred have been allowed for in the financial model to cover the expansion of the TSF in Year 2, the replacement of the mine fleet, the replacement of the power plant, and for rehabilitation at the end of the project.

These expenditures are summarized in

Table 73 below.

Table 73: Future Capital Expenditure (USD Million)

Item	Y 2	Y 10	Y 15	Y 20	Y 26
Tailings Dam Expansion	7.01				
1st Generator Replacement		10.6			
Mine Fleet Replacement			6.0		
2nd Generator Replacement				10.6	
Rehabilitation on Closure					4.2
Total over LOM (Life of Mine)					38.3

21.3 OPEX

21.3.1 Overall OPEX summary

The opex costs from year 3 onwards for the Project are summarised below. In the first two years the Project has allowed for additional expatriates to assist with training. The operating expenditure costs in the first two years are discussed in more detail later in this section.

Table 74: OPEX per Tonne of Feed - Year 3 onwards

Category	Cost / tonne of Feed
Mining	US$ 3.93
Processing	US$ 11.02
General and Administrative	US$ 6.78
Total Opex per Tonne of Feed	US$ 21.73

Table 75: Opex per tonne of concentrate - Year 3 onwards

Category	Cost / tonne of Concentrate
Mining	US$ 63.79
Processing	US$ 178.92
General and Administrative	US$ 109.94
Total cost per tonne at Mine Site	US$ 352.65

All capital and operating costs expressed above are considered to be accurate to +/- 10%.

Note that these values do not consider inflation, but do apply currency depreciation where relevant, as discussed in Chapter 22 (Economic Analysis). This accounts for the difference between these numbers and those below where currency depreciation has not been taken into account.

21.3.2 Mining Summary

Table 76: Mining Operating Costs for the First Five Years – Owner Operated Model – Real Terms (no currency depreciation)

Year	Unit Costs	2017	2018	2019	2020	2021
Total Tonnes per Annum	Jul-14	1 216 846	1 108 205	1 092 126	2 195 765	2 181 619
Equipment Diesel Cost ($/liter)	$0.80	$618 098	$562 913	$554 746	$1 115 340	$1 108 154
Equipment Maint / Spares ($/t)	$0.55	$669 416	$609 650	$600 804	$1 207 942	$1 200 159
Equipment Tyres ($/t)	$0.10	$116 566	$106 159	$104 618	$210 340	$208 985
Equipment GET/Steel ($/t)	$0.05	$55 558	$50 598	$49 863	$100 253	$99 607
Drilling Diesel Cost ($/liter)	$0.80	$32 202	$29 327	$28 902	$58 108	$57 734
Drilling Maintenance ($/t)	$0.21	$253 766	$231 110	$227 757	$457 914	$454 964
Drilling GET / Steel (/t)	$0.04	$44 640	$40 655	$40 065	$80 552	$80 033
Explosives ($/t)	$0.31	$380 313	$346 359	$341 333	$686 265	$681 844
In-pit Services ($/t)	$0.14	$171 790	$156 452	$154 182	$309 990	$307 993
Labour Expat (Fixed Cost)		$521 394	$521 394	$145 787	$145 787	$145 787
Labour Local (Fixed Cost)		$559 837	$559 837	$559 837	$559 837	$559 837
Total Annual Mining Cost		$3 423 580	$3 214 453	$2 807 895	$4 932 328	$4 905 097
Annual Ave Cost/tonne		$2.81	$2.90	$2.57	$2.25	$2.25

Table 77: Mining Costs over Life of Mine - Real Terms (no currency depreciation)

Category	Value
Total Tonnes moved over LOM	40 221 483
Equipment Diesel Cost ($/litre)	$20 430 526
Equipment Maint / Spares ($/t)	$22 126 779
Equipment Tyres ($/t)	$3 852 950
Equipment GET/Steel ($/t)	$1 836 403
Drilling Diesel Cost ($/litre)	$1 064 414
Drilling Maintenance ($/t)	$8 387 965
Drilling GET / Steel (/t)	$1 475 529
Explosives ($/t)	$12 570 833
In-pit Services ($/t)	$5 678 327
Labour Expat (Fixed Cost)	$4 541 687
Labour Local (Fixed Cost)	$14 555 751
Total Annual Mining Cost	$96 521 163
Annual Ave Cost/tonne	$2.40
Total mining cost	$96 521 163
ROM tonnes	22 274 229
Operating cost $ /ROM t	4.33
Concentrate Tonnage	1 440 734
Operating cost $ /Concentrate t	$ 66.99

21.3.2.1 Mining Basis of Estimate

The Molo production schedule was put out to the market for contractor mining companies to tender. Due to the low tonnages required per annum the submissions we received were not in an acceptable price range. As a result of this the base

case for mining costing was specified to be owner operation with capitalised equipment and the alternative was owner operations with leased equipment.

The mining operating cost estimate was generated from first principles based on OEM equipment operating costs, up to date consumable quotes and up to date labour cost estimates.

Both sets of operating cost parameters where generated from first principles and bench marked against existing operations and relevant contractor rates from other projects.

The input costs for diesel is $0.801 per litre and explosives cost including accessories is $1.33 per kilogram.

The operating costs for all equipment specified were generated for OEM repairs and maintenance costs and fuel consumption estimates.

The number of expats to be used in the Molo Graphite Project has been limited to as few as possible. This is due to the mining being relatively simple, and the productivity levels being lower than normal.

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

21.3.3 *Process Summary*

Table 78: Process Operating Costs – 1st Two Years - Real Terms (no currency depreciation)

Description	Annual Cost	Unit cost	Unit cost
	$/annum	$/t Feed	$/t Product
Variable Costs	8 541 101	9.91	158.93
Power Cost	3 839 276	4.45	71.44
Liners	317 500	0.37	5.91
Grinding Media	800 439	0.93	14.89
Reagents	1 272 810	1.48	23.68
Product Drying	295 009	0.34	5.49
Product Bagging	Included in logistics	0.00	0.00
Maintenance	541 941	0.63	10.08
Laboratory	658 386	0.76	12.25
TSF Deposition Cost	325 875	0.38	6.06
Rehanding Costs	479 916	0.56	8.93
Water	9 952	0.01	0.19
Fixed Costs	2 133 851	2.48	39.71
Labour	2 133 851	2.48	39.71
Maintenance	Included above		
Power Cost	Included above		
Total Process Operating Cost	10 674 952	12.38	198.63

Table 79: Process Operating Costs - Year 3 onwards - Real Terms (no currency depreciation)

Description	Annual Cost	Unit cost	Unit cost
	$/annum	$/t Feed	$/t Product
Variable Costs	8 541 101	9.91	158.93
Power Cost	3 839 276	4.45	71.44
Liners	317 500	0.37	5.91
Grinding Media	800 439	0.93	14.89
Reagents	1 272 810	1.48	23.68
Product Drying	295 009	0.34	5.49
Product Bagging	Included in logistics	0.00	0.00
Maintenance	541 941	0.63	10.08
Laboratory	658 386	0.76	12.25
TSF Deposition Cost	325 875	0.38	6.06
Re-handling Costs	479 916	0.56	8.93
Water	9 952	0.01	0.19
Fixed Costs	1 580 914	1.83	29.42
Labour	1 580 914	1.83	29.42
Maintenance	Included above		
Power Cost	Included above		
Total Process Operating Cost	10 122 016	11.74	188.35

21.3.4 Process Basis of Estimate

21.3.4.1 Methodology

The process plant operating cost estimate is based on steady state operating conditions of 862 000tpa. The estimate has been calculated based on the process flow sheets and derived from mass balance outputs and laboratory test work results.

21.3.4.2 Foreign currency assumptions

The base currency of the Project is in United States Dollars (USD). All costs have been estimated in USD and were necessary converted to USD using the following rates: ZAR 11.31 = USD 1.00 and 0.79 EURO to USD 1.00

21.3.4.3 Labour

The process plant labour costs are based on a staffing model for similar processing plants of an equivalent size and complexity based on the input from Minopex, the DRA operational division.

Operating personnel compliments are based on rotational shift teams. Allowances have been made for leave and absenteeism within the process and engineering teams. A higher number of expats personnel is expected during the first two years of operation.

21.3.4.4 Power

The power costs used are based on the use of diesel generated power at a cost of $0.20/kWh, which was determined based on a diesel supply cost of USD0.801/L.

The total connected electrical load, inclusive of all standby units in the plant was calculated by summation of all the equipment specified on the mechanical equipment list. The estimated average running load was calculated using expected power draw from the equipment sizing calculations. These calculations showed the power cost to be USD 4.45 per tonne milled.

21.3.4.5 Water

The monthly water cost estimate is based on raw water consumption figures are from the plant mass balance and expected losses, (evaporation and interstitial losses) at the tailings storage facility.

21.3.4.6 Reagents and Product Drying

The consumption figures used in determining the operating cost estimate are based on test work data, vendor information and are supported by the plant mass balances. The reagent costs used in the operating cost estimate is based on supply rates obtained from reputable reagent suppliers.

21.3.4.7 Liners and Grinding Media

The liner life and grinding media consumption for the ball mill was estimated using abrasion index test results, mill dimensions and anticipated grinding media load and rounded up to the nearest full set figure.

Unit costs for liners were given by mill suppliers in their tenders submitted for the project. The liner life for the crushers was estimated using the abrasion index test results. Unit costs for crusher liners were given by the suppliers.

21.3.4.8 Maintenance

The costs included for stores and maintenance are based on factors applied to the relevant capital supply rates. This amounts to an estimated cost of 541 941USD/annum.

21.3.4.9 Laboratory Operating Costs

Items under this cost were obtained from a laboratory service provider. The basis of the costs includes consumables, but excludes laboratory staff. The laboratory will handle geology, mining and process plant samples.

21.3.4.10 TSF Operating Costs

SRK provided a TSF operating cost for the of USD 325 875 per annum based on a process plant throughput rate of 862 000 tpa.

21.3.4.11 Re-handling Costs

Re-handling costs were calculated from first principles assuming the operation of a front end loader at the plant run of mine feed hopper.

21.3.4.12 Process Plant Operating Cost Exclusions

The following are specific exclusions from process plant operating cost estimate:

- Concentrate off take agreement costs / penalties
- Waste rock handling (This is included in mining OPEX)
- Site closure and rehabilitation
- All VAT, import duties and / or any other statutory taxation, levies

No contingency has been allowed for in the operating cost estimate.

21.3.5 General & Admin

Table 80: General & Admin Costs – Real (no currency depreciation)

Category	Year 1&2	Year 3 onwards
1. Labour	$1 738 669	$1 101 647
2. Camp	$3 765 780	$3 586 458
3. Vehicle operation	$96 000	$96 000
4. General		
Admin		
Insurance	$425 000	$425 000
Stationery	$24 000	$24 000
Software	$24 000	$24 000
Audits	$50 000	$50 000
Tax advice	$50 000	$50 000
Communications	$120 000	$120 000
Gov Fees	$-	$-
Royalties	$-	$-
Permits	$50 000	$50 000
Customs	$45 957	$45 957
Travel	$-	$-
Business travel (marketing)	$-	$-
International Expat Flying	$158 400	$158 400
Internal Expat Flying	$132 600	$132 600
Travel to Tulear	$52 000	$52 000
Clinic	$60 000	$60 000

Category	Year 1&2	Year 3 onwards
Recruitment	$50 000	$50 000
Social Responsibility	$100 000	$100 000
Fuel depot operation	$-	$-
Tana Office	$100 000	$100 000
Total G + A (Annum)	$7 042 406	$6 226 061

Notes

1. Labour costs drop in Year 3 onwards as the expat complement is scaled down.
2. Camp costs include the cost of feeding the total staff complement
3. These costs have been complied based on information from Energizer and DRA
4. These exclude all head office overhead costs
5. The labour portion of the G&A estimate has been built up from 1st principles

21.3.6 Operating Expenditure Risks

Production rates during the ramp phase (expected to last approximately 3 to 6 months) will fluctuate and this will have an impact on the unit operating costs in this period. This has not been taken into account in the financial model.

In addition, as noted in section 13, the ore feed to the plant is highly variable, and this has the potential to result in variable product quality (affected by flake size distribution, and product carbon grade) and product quantity (affected by recovery). The basket price is a function of product quality and if the plant is unable to produce the planned quantities of graphite concentrate, at the specified grades, then operating costs will be materially affected. Laboratory scale test work has shown that an attritioning circuit will mitigate the impact of ore variability on product quality but this has not been proven at pilot or production scale.

21.4 Logistics Costs

Table 81: Product Logistics Costs

Category	Cost / FEU (USD)
Total FOB / FEU Fort Dauphin	4 712
Sea Freight to Rotterdam	4 000
Total selling cost / FEU: Rotterdam	8 712
Tonnes / container	26
Cost / tonne concentrate FOB Fort Dauphin	182
Cost / tonne concentrate shipping to Rotterdam	155
Total transport cost / tonne concentrate	337

Notes:

1. FEU = Forty Foot Equivalent Unit i.e. a Forty Foot Container
2. Information built up from a variety of independent sources including SDV, Panalpina, Unitrans, Strang.
3. Energizer elected not to take insurance on the goods in transit.
4. The costs above assume that empty containers are delivered at no charge at Fort Dauphin.
5. Some numbers rounded.

22 ECONOMIC ANALYSIS

The economic analysis is based on the results of cash flow forecast financial modelling. Inputs to the financial model were obtained from the various technical experts where possible, or from third parties considered experts in their field. In instances, where this is not possible, (such as macroeconomic forecasts), a reasonable view has been taken based on historic trends, and is disclosed in the statement of principal assumptions (below).

It is inherent in such financial modelling that all inputs are assumptions, and all cash flows and returns are forecasts.

Values shown are rounded to an appropriate level; all financial values shown in this report are based in United States Dollars (US $). All returns shown are post-tax, as pre-tax values are not considered relevant to this stand-alone project.

22.1 Cash Flow Model

22.1.1 General

The cash flow model was prepared in Microsoft Excel, modelling cash flows monthly during the construction period and quarterly during the operations phase.

The model takes account of various forecast macroeconomic effects in addition to the project-specific inputs. Where possible, costs are modelled in the currency in which they are expected to be incurred.

Returns are provided in both real and nominal terms; investors should understand the difference as follows:

- Nominal terms cash flows include all macroeconomic effects, including inflation and currency movement, and forecast the actual amount of cash movements to take place at any given date
- Real terms cash flows deflate the nominal cash flows, adjusting them to reflect the current (start of 2015) purchasing power of the monies, i.e. as if the purchasing power of the monies does not change over time. The conversion to real terms has used the US consumer price index assumptions.

22.1.2 Base Case Scenario – High Level Description

The ‘Base Case’ project models a feed to the process plant of 862 500 tonnes per annum (“tpa”), with slight annual variations, and the mine scheduled to match this requirement, over a 26 year life of mine.

The funding of the project’s capital expenditure is expected to be 50% provided by equity finance and 50% by debt finance. The debt finance facility is modelled as a ‘corporate finance’ facility, i.e. traditional balance-sheet based borrowing.

22.1.3 *Alternate Case Scenario (Equity Finance Only) – High Level Description*

The 'Alternate Case' (referred to as 'Equity Finance Only') changes the funding assumption to 100% equity finance. All other assumptions remain constant between the scenarios.

22.1.4 *Returns*

Returns are measured as at Madagascar, after all relevant in-country taxes, royalties and duties are paid. Any taxes, duties, levies, or other costs involved with the repatriation of income between Madagascar and Canada have not been considered.

22.2 Statement of Principal Assumptions

22.2.1 *Cost Estimates*

Cost estimates were sourced in the third and fourth quarters of 2014, with updates in the first quarter 2015. Costs are expected to be valid until 2nd half 2015, after which ongoing inputs are escalated. Capital expenditure has not been escalated, except for capital items incurred during the operations period, which have been escalated at the assumed US CPI rate (see below).

22.2.2 *Timing*

The financial model assumes a 19 month period from first capital expenditure to commissioning and a 26 year life of mine.

22.2.3 *Capital Expenditure*

The Capital expenditure is broken up into the following categories:

- Construction costs (including design development allowance)
- Contingency
- Funding costs, (including interest during construction and an allowance for costs of arranging funding)
- Taxation (Madagascar duties on foreign services)
- VAT (net of refunds received during the construction period)

Table 82: Capital Expenditure – Base Case versus Equity

Cost Category	Base Case	Equity Finance Only
Construction Costs	US $ 163.7m	US $ 163.7m
Contingency	US $ 24.6m	US $ 24.6m
Funding Costs	US $ 8.1m	-
Taxation	US $ 9.6m	US $ 9.6m
Net VAT	US $ 8.1m	US $ 8.1m
Total	US $ 214.1m	US $ 205.9m

Please note, totals may not add up exactly due to rounding

22.2.4 Operating Expenditure

Operating expenditure falls into the categories listed below. The Project assumes that additional expatriate staff will be on-site in the first two years of operations, therefore the 'steady state', (or 'run rate') operational expenditure is achieved from the start of the third operating year. Steady State rates are shown below; where the cost is shown twice (mining costs and process costs), this is to reflect the cost of the activity undertaken and the cost in relation to the sale price.

Table 83: OPEX Inputs

Category	Annual Cost at Steady State (US $)
Mining Costs	$ 2.1 per tonne mined
	$ 63.8 per tonne graphite concentrate
Process Costs	$ 11 per feed tonne
	$ 178.9 per tonne graphite concentrate
General, Administrative & Overhead Costs	$ 109.9 per tonne graphite concentrate
Sub total	$ 352.6 per tonne graphite concentrate

Please note, these values include the effect of currency depreciation on certain cost items which are considered in currencies other than US Dollars.

22.2.5 *Transportation*

The model assumes that the price of graphite concentrate is "C&F" (Cost and Freight) at Europe.

The cost of transportation from mine site to Europe on these terms is modelled at US $ 337 per tonne of graphite concentrate as at 2015.

22.2.6 *Ongoing Capital Expenditure*

Ongoing capital expenditure requirements are captured within the financial model, as follows:

Table 84: Stay In Business Capital

Item	Amount Before Inflation	Timing
Expansion of Tailings Storage Facility	US $ 7m	2nd Operational Year
Power Plant Replacement	US $ 10.6m	10th Operational Year
Mining Fleet Replacement	US $ 5.9m	15th Operational Year
Power Plant Replacement	US $ 10.6m	20th Operational Year
Site rehabilitation	US $ 4.2m	Post-operational period

22.2.7 *Pit Design & Production Schedule*

The production schedule assumes approximately 862,500 tonnes per annum of feed to the plant at full capacity. The actual amount of mining activity undertaken varies each year to achieve this result. The mine design reduces the feed in the final operating year, resulting in an average of 856,701 tonnes per annum over the life of mine.

Production of different flake size distribution and product grade is as follows, fixed for the LOM.

Table 85: Flake Size and Grade

Flake Size	Production	Grade
Jumbo (+50 mesh)	23.5%	96.9%
Large (+80 mesh)	22.8%	97.06%
Medium (+100 mesh)	6.9%	97.2%
Fine (-100 mesh)	46.8%	97.56%

22.2.8 Revenue and Product Pricing

Detailed product pricing forecasts were sourced from Roskill. The applied price of a tonne of average Molo graphite (i.e. the blended average) as at 2014 is US $ 1,375 / tonne C&F Europe.

Roskill have provided a forecast range of graphite pricing for each flake size and product grade per year, and the pricing applied in the model is within the lower half of the range for every year. This results in a weighted average price of US$1,689 / tonne in 2017. A full description of the pricing assumptions is available under Chapter 19 of this technical report.

22.2.9 Working Capital

The model assumes an 82 day working capital requirement, to transport graphite concentrate from the project site to Europe, including an assumption of 15 day payment terms.

Costs are assumed to be paid on 30 day terms.

22.2.10 Funding Assumptions

For both scenarios, it has been assumed at this stage that senior debt will not finance working capital, but equity will provide working capital funding requirements.

22.2.10.1 Base Case Scenario:

Funding has been assumed on these terms.

Table 86: Funding

Term	Value
Repayment period	8 years from the start of operations
Interest Rate	The all-in rate assumed is 5.75% over LIBOR, including all costs and taxation
LIBOR	Forecast to rise from 0.6 in 2015 to 3.5 in 2022 and maintained thereafter
Upfront cost of arranging	US $ 0.9m
Capital repayment commencement	First repayment 3 months after operations commence

22.2.10.2 Alternate ("Equity Finance Only") Case

All funding is expected to be sourced from equity finance.

22.2.11 Macroeconomic Assumptions

The model includes four currencies. Exchange rates are treated as follows:

Table 87: Macro Economic Assumptions

Currency	Treatment
United States Dollar (US $)	The base currency of the model, remains constant
South African Rand (ZAR)	The model assumes a 2015 exchange rate of 11.3 ZAR:US$, depreciating against US$ over time at an assumption of approximately 3.6% p.a.
Euro (EUR)	The model assumes a static exchange rate of 1.2 US$:EUR
Malagasy Ariary (MGA)	The model assumes a 2015 exchange rate of 2746 MGA:US$, depreciating against US$ over time at an assumption of approximately 5.6% p.a.

There are various inflation rates applied in the model ('CPI' refers to Consumer Price Index).

Table 88: Inflation Rates

Escalation Rate	Treatment
United States CPI	1.5% in 2015 increasing to 1.9% long term
South Africa CPI	5.0% long term
European CPI	1.1% in 2015 increasing to 1.8% long term
Madagascar CPI	7% in 2015, decreasing to 6.4% long term (following a steady reduction in historical inflation rates)

The diesel price applied in the model was based on oil pricing at US$ 75 / barrel of crude oil. The model escalates this price at the US CPI rate.

22.3 Headline Forecast Financial Statements

Refer Table 89 for the headline financial statements.

Table 89: Forecast Financials

Year	Tonnes Mined	Tonnes Graphite produced	Revenues (US$ '000s)	Construction Period net Costs (US$ '000s)	Cash Available for Equity (US$ '000s)
2015	-	-	-	(123 598)	-
2016	-	-	-	(90 488)	(0)
2017	1 216 846	54 586	92 203	-	16 443
2018	1 108 205	57 618	104 839	-	29 812
2019	1 092 126	61 216	118 149	-	45 907
2020	2 195 765	58 455	120 478	-	44 817
2021	2 181 619	48 495	106 802	-	48 740
2022	2 315 897	53 030	124 820	-	38 610
2023	2 316 905	52 626	132 431	-	53 248
2024	1 534 189	53 065	142 824	-	57 104
2025	1 739 194	53 200	153 332	-	80 300
2026	1 388 345	53 151	155 766	-	87 758
2027	1 245 742	52 557	156 337	-	87 903
2028	1 552 517	51 733	156 486	-	86 687
2029	1 254 167	51 901	159 624	-	85 639
2030	1 725 843	52 415	164 051	-	85 739
2031	1 883 589	52 769	167 944	-	90 084
2032	1 519 330	52 828	170 810	-	94 676
2033	1 755 883	53 011	174 285	-	94 927
2034	1 532 011	52 718	176 335	-	91 870
2035	1 683 636	52 340	178 335	-	89 615
2036	1 270 586	51 812	179 563	-	95 555

Year	Tonnes Mined	Tonnes Graphite produced	Revenues (US$ '000s)	Construction Period net Costs (US$ '000s)	Cash Available for Equity (US$ '000s)
2037	1 646 307	51 653	181 723	-	99 426
2038	1 783 019	52 001	186 093	-	100 660
2039	1 771 239	52 652	191 658	-	103 735
2040	864 858	52 796	195 448	-	108 304
2041	862 495	52 652	198 297	-	109 887
2042	781 170	47 150	180 646	-	140 155

22.4 IRR and NPV

The forecast IRR (Internal Rate of Return) and NPV (Net Present Value) are as follows:

22.4.1 Base Case

Table 90: Base Case Metrics

Metric	Value
Nominal IRR	31.2%
Nominal NPV at 8% Discount Rate	US$ 521.6m
Nominal NPV at 10% Discount Rate	US $ 389.8m
Nominal NPV at 12% Discount Rate	US $ 293.6m
Real IRR	28.9%
Real NPV at 10% Discount Rate	US $ 293.1m
Payback period	4.84 years

22.4.2 Alternate ("Equity Finance Only") Case

Table 91: Equity Only Metrix

Metric	Value
Nominal IRR	24.4%
Nominal NPV at 10% Discount Rate	US $ 376.7m
Real IRR	22.2%
Real NPV at 10% Discount Rate	US $ 273.6m

22.5 Taxes and Royalties

Madagascan legislation includes a law "établissant Régime Spécial pour les Grands Investissements dans le Secteur Minier Malagasy", Law 020 of 2001, updated in Act 22 of 2005, known as "LGIM", to provide for a special regime for large mining investments.

The project qualifies for LGIM, and the financial model has assumed that the preferential treatment available under LGIM will be received.

Under LGIM, the taxation and royalties considered in the financial modelling is as follows:

Table 92: LGIM

Metric	Value under LGIM
Revenue Royalties	1%
Corporate Taxation	20%
Taxation on Foreign Services	6.5%
Personal Income taxation for Employees	20%
Import Duties on equipment specified under the Investment Plan (a plan of items expected to be imported for project construction and operation)	0%
Dividend Withholding Tax (Madagascar)	0%

Please note: Energizer currently expects to invest into Madagascar through its Mauritian subsidiary.

In addition to government taxation and royalties, a Mineral Rights Royalty agreement is in place, to the extent of 1.5% of free cash flow to Malagasy.

22.6 Sensitivity Analysis

A sensitivity analysis has been conducted on the Base Case Scenario, to ascertain the effect on IRR of possible changes in certain key assumptions.

The key elements which were assessed were each assessed at a 10% adverse change, with the following effects on the Nominal Equity IRR:

Table 93: Sensitivity

Item Assessed	Effect on Nominal Equity IRR	New Nominal Equity IRR
No changes (reference point)	-	31.2%
Operating expenses up 10%	- 6.2%	25%
Selling Price down 10%	- 4.3%	26.9%
Capital Expenditure up 10%	- 2.3%	28.9%

These were then assessed in more detail. Each was assessed on a stand-alone basis (i.e. not combining the effects of two, or more, sensitivities), at changes of 10%, 20% and 30%, and the resulting Nominal Equity IRR's are shown in the following chart. See Figure 117. Steeper lines indicate a higher sensitivity to the variation applied.

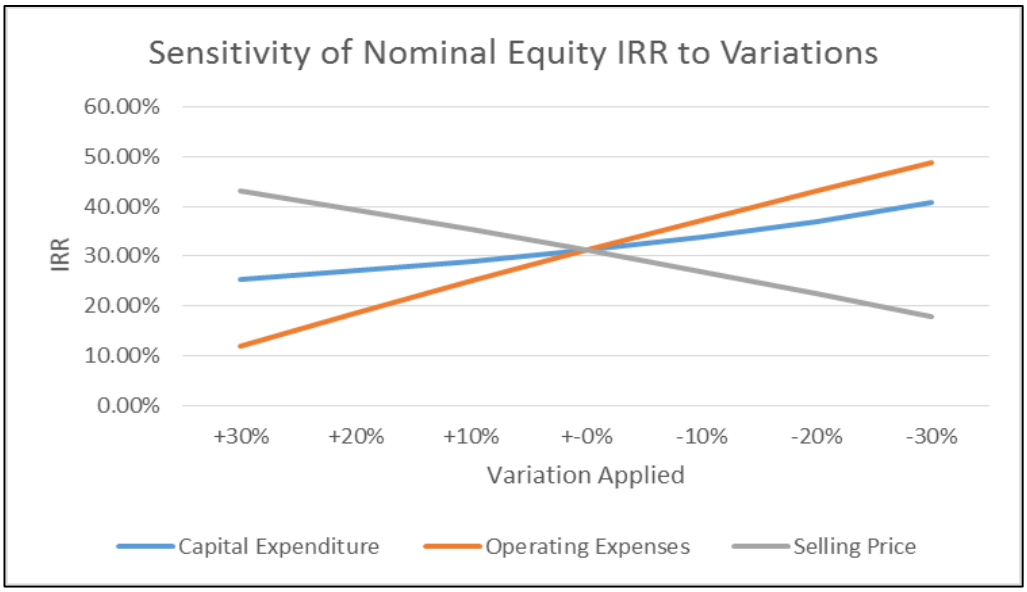

Figure 117: Sensitivity of Nominal Equity IRR to Variations

As can be seen from these results, the project is less sensitive to capital expenditure than to the selling price of graphite concentrate; however, the project is more sensitive to changes in operating expenditure than to the selling price of graphite concentrate.

22.7 Cautionary Language

The information provided in this Molo 2015 FS is entirely in the form of forward-looking forecasts. These are based on current expectations, estimates and assumptions that are subject to numerous risks (known and unknown) that have the potential to change those assumptions. Actual values and financial returns therefore may differ from the forecasts provided, in some instances materially.

No assurance can be given that any of the events anticipated by the forecasts will transpire or occur or, if any of them do so, what benefits Energizer will derive therefrom.

Readers of this report are therefore advised not to place undue reliance on the forecasts provided.

23 ADJACENT PROPERTIES

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

24 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any other information on the Project that would affect their interpretations, or conclusions regarding the Molo Graphite Project.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology

Energizer'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 updated resource statement for the Molo deposit, which is stated in accordance with the CIM Guidelines.

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 % C, have been upgraded by infill drilling from the Indicated to Measured Resource category.

25.2 Mining

Maiden mineral reserves of 22 437 000 tonnes have been declared for the Molo Graphite Project at an average grade of 7.04% and based on the information contained in the Molo 2015 FS it is possible to economically mine this deposit.

25.3 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.

25.4 Recovery methods

The laboratory, pilot and vendor test work conducted prior to and during the study defined the required process flow sheet. This was duly translated into a full-scale production plant flow sheet as described within this Report. The flow sheet unit processes were populated and individual component equipment selected according to either pilot plant precedents or, where these were not available, proven practice within the industry, in conjunction with suitably experienced vendors. All process designs and selections were based on conventional, proven mineral processing practices.

The processing selections and configurations built into the design are adequately suited to the requirements. Based on the mining and metallurgical test work

information presented elsewhere within the Molo 2015 FS, and assuming within specification ROM (Run of Mine) ore is fed to the plant, the required recovery is expected to be attainable at the throughput stated. Note that this recovery is based on lab and pilot scale test work and may reduce slightly on a full scale plant due to operational inefficiencies. This possible reduction has not been taken into account in the financial analysis.

25.5 Infrastructure

This is a green fields project in a very undeveloped area so all infrastructure for the mine and plant has to be installed as part of the Project.

25.6 Water

The detailed geo-hydrological test work has demonstrated that it is possible to supply the process and plant and supporting infrastructure from a network of bore holes.

25.7 Environmental, Social

The detailed Environmental and Social Impact Assessment has concluded that there are no fatal flaws affecting the development of the Project.

25.8 Permitting

The Projectwill have to obtain multiple permits of which the most important early permits are the environmental and mining exploration permits. Energizer does not currently have a valid exploration permit and this is viewed as a key Projectrisk.

25.9 Tailings

The site selection study evaluated six possible sites for the TSF. A suitable site has been identified and the geochemical and geohydrolical work done has shown that it is possible to construct an unlined facility, pending approval of the ESIA by the Malagasy government.

25.10 Project Summary

The key results of the Molo 2015 FS can be summarized as follows:

Table 94: Results Summary

1	Post-tax: NPV (10% Discount Cash Flow)(1)(2)	US$389,797,113
2	Post-tax: IRR(1)(2)	31.2%
3	Payback(2)	4.84 years
4	Capital cost ("CAPEX")	US$149.9 million
5	Design Development Allowance	US$13.8 million
6	Contingency	US$24.6 million
7	On-site Operating Costs ("OPEX") per tonne of concentrate, Year 3 onward)	US$353
8	Transportation per tonne of concentrate (from Mine site to Madagascar Port Year 3 onward)	US$182
9	Transportation per tonne of concentrate (from Madagascar Port to European Customer Port from Year 3 onward)	US$155
10	Average annual production of concentrate	53,017 tonnes
11	Life of Mine ("LOM")	26 years
12	Graphite concentrate sale price (US$/tonne at Start Up - 2017)	US$1,689 per tonne
13	Average Head Grade	7.04%
14	Average ore mined per annum	856,701 tonnes
15	Average stripping ratio	0.81:1
16	Average plant recovery	87.80%

Notes

(1) Assumes project is financed with 50% debt and 50% equity.

(2) Values shown based on nominal cash flows, which include the effect on inflation. Costs are increased on an annual basis by the relevant inflation

25.11 Risks and Uncertainties

A qualitative risk assessment was done as part of the Molo 2015 FS utilizing the collective experience of the study team and the results are summarized below. A total of 59 risks were identified during the review process.

Table 95: Risk Summary

Category	Risks before Controls	Risks after Controls
Extremely High	9	2
High	42	18
Moderate	6	28
Low	2	9
Very Low	0	2
Total:	59	59

The extremely high risks before controls were identified as follows:

1. The exploration permit covering the Molo pit expired in 2011 and has yet to be officially renewed (Exploration Permit #3432 is the permit in question).
2. Current delays in issuing new mining permits.
3. Requirement that all voids / excavations be backfilled without exception.
4. No power on site.
5. Relatively limited water resources on site: Moderate yielding aquifer and limited surface run-off due to locality on catchment divide.
6. Inaccurate landownership data
7. The process design may not get the balance right between:
   a. Coarse flake recovery.
   b. Product carbon grade.
   c. Overall recovery.
8. Unit costs of moving product are high
9. Theft during construction and operation (diesel, cable, etc.)

After controls the extremely high risks are as follows (reduced from 9 to 2):

1. The exploration permit covering the Molo pit expired in 2011 and has yet to be officially renewed (Exploration Permit #3432 is the permit in question).
2. Current delays in issuing new mining permits.

After controls the high risks are as follows (reduced from 39 to 18):

1. Requirement that all voids / excavations be backfilled without exception.
2. Inaccurate landownership data.
3. Unit costs of moving product are high.
4. Project NPV and IRR lower than the PEA.
5. Theft during construction & operation (diesel, cable, etc.)
6. No off take agreements signed yet or formal product specifications received.
7. The current execution strategy calls for contracts to be placed before permits are granted.
8. The project has modelled the diesel price at 0.8 USD / litre.
9. ESIA review timeframes could extend past project start date (indications are 6-9 months for ESIA approval from O.N.E.)
10. The process design may not get the balance right between:
    a. Coarse flake recovery.
    b. Product carbon grade.
    c. Overall recovery.
11. Future Land Claims (Ancestral Rights).
12. The process plant may not achieve a consistent on spec product, especially as the feed grade to the plant varies and this may make process control difficult.
13. Madagascan political situation remains potentially unstable.
14. Difficult logistics getting material on and off the island plus very bad roads.
15. Contractors P&G's high due to locality.
16. The projects returns are reliant on a real term increase in the price of graphite
17. Implementation of the preferential taxation arrangement may be difficult
18. The debt funding assumptions may not be achievable

26 RECOMMENDATIONS

26.1 Geology

No further recommendations.

26.2 Mining

The long mine life of the Molo Graphite 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 Molo Graphite 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:

- Evaluate a range of different attrition mill media 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 the mine life past the initial 5 years.

26.4 Recovery methods

Optimization and refinement opportunities exist regarding the process design which could reveal benefits over the equipment selections and unit process detail within the current design. The latter are based essentially on test work outcomes pursued and reported on thus far for study purposes.

Appropriate test work is recommended prior to the initiation or during the course of a detailed design phase preceding construction. This would include the following:

- Bulk material flow test work;
- Additional test work, in conjunction with vendors and in line with ongoing technical developments, aimed at further refinement of the polishing and attrition milling processes;

- Concentrate attritioning circuit static and dynamic thickening tests, including reagent scoping and optimization trials;
- Further investigation into potentially replacing the final tailings disposal positive displacement pumps with more common centrifugal pump trains by reducing the slurry solids concentration for overland pumping. This will include examination into whether the overall water balance and supply system can reasonably accommodate such a change.

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:

- Updating the current dynamic water balance including a dynamic TSF water balance. The current water balance only assumed average monthly inflows from the TSF into the RWD. It would be recommended to confirm the water availability on the Molo Graphite Project if drought conditions occur and the TSF model element is included in the dynamic water balance
- 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 and scavenger wells 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 Molo Graphite Project site should be done as soon as possible
- The installation and testing of the additional well field boreholes must be undertaken. The groundwater resource model must be updated to include site specific borehole data
- The environmental geochemical test work of the Molo 2015 FS should be confirmed by selective testing of samples from the latest exploration and metallurgical test programs. The geochemical model should be updated accordingly

26.7 Environmental, Social

- GCS recommends the installation of a suitable weather station at, or as near as possible to the proposed Project site, even before construction commences. 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 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

- An application for the exploration permit in Energizer's name is a critical step in the larger permitting and licensing regime and requires early attention and dedicated involvement

- Security of land tenure is a process and is estimated to take 7 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 etc should be undertaken within the ESIA review period (6 months), which is expected to be from March till August 2015
- Compilation of a comprehensive legal register
- Municipal elections are scheduled for July 2015. All above-mentioned permitting processes should commence prior to and in anticipation of these elections

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

26.9 Tailings

Additional work required during detailed design of the TSF and adjacent RWD is as follows:

- The full rheology and beaching characteristics for the tailings are not known which leaves uncertainties regarding the optimum deposition design. This will need to be investigated via large scale tests once suitably sized pilot process plant samples are available. It should be noted that such large scale tests will also provide additional more representative samples which can be used to carry out further testing of other tailings characteristics, such as consolidation, permeability and shear strength, which should be used to validate / revise the assumptions made for the stability assessments, seepage / drainage assessments and water balance
- The geotechnical investigation was carried out for the general TSF area only, and was not focused on the specific design elements as the location of these was not known at the time. Additional focused geotechnical investigations will be required to confirm the geotechnical conditions at specific locations
- The depth to groundwater is not known in the immediate vicinity of the RWD. In the event that ground water is shallow, it may not be possible to excavate the RWD basin to the required depth without employing dewatering measures, or alternatively constructing an additional RWD downstream. The depth to groundwater and any seasonal fluctuations will need to be investigated by installation of a groundwater monitoring borehole, which must be monitored during the wet season
- Water quality data is required over a period of time to provide a baseline for comparison once the TSF is commissioned. To provide the necessary baseline data, regular ground and surface water monitoring must be carried out leading up to the date when the TSF is commissioned

- The overall design will need to be developed to a level required for construction and to optimise the design with regard to technical, environmental and economic considerations, whilst taking due cognisance of additional information made available, including the additional studies detailed

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27.4 Infrastructure

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28 CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"), Part 8.1.

a) **Name, Address, Occupation:**

Doug Heher

DRA Projects SA (Pty) Ltd, DRA Minerals Park, 3 Inyanga Close, Sunninghill, 2157, South Africa

Project Manager

b) **Title and Effective Date of Technical Report:**

Molo Feasibility Study, National Instrument 43-101 Technical Report

Effective date: 6th February 2015.

c) **Qualifications:**

I graduated with a B.Sc. Engineering (Mechanical) degree from the University of KwaZulu Natal - Durban in 1992.

I am a Registered Professional Engineer with the Engineering Council of South Africa (ECSA) – Reg. No. 990333

I have completed various studies as summarized below. I have practiced my profession since 1997.

I have read the definition of "qualified person" set out by NI 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 fulfill the requirements to be a "qualified person" for the purpose of NI 43-101. My relevant experience, includes 18 years as an engineer and eight years as a project engineer and project manager.

d) **Site Inspection:**

I visited the site on 11-13 March 2015 and again on 19 August 2014

e) **Responsibilities:**

I was responsible for the following: Sections 1.1 to 1.3, 1.10,1.11,1.13 to 1.15,1.17, 1.18.4 to 1.18.10,1.19.4 to 1.19.9, Section 2, Section 3, Section 17, Section 18, Section 20, Section 21, Section 23, Section 24, Section 25.4 to 25.9, 25.11, Section 26,4 to 26.9, Section 27.4 to 27.5. Signed 23 March 2015

f) **Independence:**

I am independent of Energizer Resources Inc. in accordance with the application of Section 1.5 of NI 43-101.

g) **Prior Involvement:**

I have no prior involvement with this project.

h) **Compliance with NI 43-101:**

I have read NI 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, 23 March 2015

Doug Heher
Project Manager

Summary of recent experience

Year	Client	Commodity	Type Of Study	Project Description
2008	Red Island Minerals	Coal	PFS	PFS on Sakoa deposit in Madagascar incl. infrastructure
2008/9	Lonmin / Anglo JV	Platinum	DFS	Full process study incl. test work on the Pandora mine
2010	Confidential	Iron Ore / Coal	Concept	Bulk port on Yangtze river in China
2011	ENRC	Copper	Concept	3.6 Mtpa Copper Concentrator in the DRC
2011	ENRC	Copper	FS	3.6 Mtpa Copper Concentrator in the DRC
2012	ENRC	Copper	Execution	3.6 Mtpa Copper Concentrator in the DRC
2012	Frontier	Rare Earths	PFS	Zandkopsdrift study in Northern Cape
2012	CIC Energy	Coal	Concept	Capex estimate for complete coal complex in Botswana
2012	Keegan	Gold	PFS review	Review of 3rd party PFS for Esaase mine in Ghana
2012	Keegan	Gold	Concept	Trade Off Study for the Esaase mine in Ghana
2013	Asanko	Gold	PFS	Full PFS from mining to economics – Esaase Ghana
2014/15	Energizer	Graphite	DFS	Full DFS from geology to economic analysis

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"),, Part 8.1.

a) **Name, Address, Occupation:**

David Alan Thompson

Beacon Minerals, 133A Villiers Drive, Clarendon, Pietermaritzburg, 3201

Mining Consultant

b) **Title and Effective Date of Technical Report:**

Molo Feasibility Study, National Instrument 43-101 Technical Report

Effective date: 6th February 2015.

c) **Qualifications:**

I graduated with a B.Tech (Mining) degree from the University of Johannesburg - in 2010.

I am a Registered Certificated Engineer with the Engineering Council of South Africa (ECSA) – Reg. No. 201190010 and a Certificated Mine Manager (Coal Mines) No 5066

I have completed various studies as summarized below. I have practiced my profession since 2000.

I have read the definition of "qualified person" set out by NI 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 fulfill the requirements to be a "qualified person" for the purpose of NI 43-101. My relevant experience, includes 12 years as a mining engineer and 3 years as a mining consultant.

d) **Site Inspection:**

I visited the site on 11-13 March 2014.

e) **Responsibilities:**

I was responsible for the following sections: 1.8, 1.18.2, 1.19.2, Section 15, Section 16, Sections 25.2, 26.2, 27.2.

f) **Independence:**

I am independent of Energizer Resources Inc. in accordance with the application of Section 1.5 of NI 43-101.

g) **Prior Involvement:**

I was responsible for the review and sign off of sections 15 and 16 and section 21.2.1 and 21.3.1 of the previous NI 43-101 technical report, filed on 12th April 2013.

h) **Compliance with NI 43-101:**

I have read NI 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, 23 March 2015

David Alan Thompson
Mining Consultant

Summary of recent experience

Year	Client	Commodity	Type Of Study	Project Description
2000/08	Various	Various	Various	Various
2008	Impala Platinum	PGM	Feasibility Study	Feasibility Study on conventional vs. mechanized mining
2008/9	Kangra Coal	Coal	PFS	Western expansion pre-feasibility
2009	Teal Mining (Konkola South)	Copper	Feasibility Study	Mine Design and Costing
2010	WBJV - Massive	PGM	Feasibility Study	Mine Design and Costing
2010/11	Zululand Anthracite Colliery	Anthracite	Feasibility Study	Feasibility + Execution Ngwabe Expansion
2012/13	BECSA – BHP Biliton	Coal	PFS	Pre-Feasibility Study opencast coal mining -RSA
2013	Global Resource	Coal	Concept Study	Concept Study opencast coal mining - Mozambique
2013	Rockgate Capital	Uranium	PFS	Pre-Feasibility Study uranium mining - Mali
2014/15	ENRC -Estima	Coal	Feasibility Study	Feasibility Study opencast coal mining - Mozambique
2014/15	Elandsfontein Phosphate	Phosphate	Feasibility Study	Feasibility Study
2014/15	Maamba Colliery	Coal	BFS	SAMEC compliant resource to reserve mine planning
2014/15	Energizer Resources	Graphite	DFS	Full DFS from geology to economic analysis
2015	ICVL	Coal	Concept Study	Concept Study opencast coal mining - Mozambique

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"),, 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: 6th February 2015.

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 since 1998.

I have read the definition of "qualified person" set out by NI 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 fulfill 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 Energizer Resources Inc. in accordance with the application of Section 1.5 of NI 43-101.

g) **Prior Involvement:**

I was responsible for managing the laboratory and pilot scale programs at SGS Lakefield in the role of a consulting metallurgist.

h) **Compliance with NI 43-101:**

I have read NI 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: 23 March 2015

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	Energizer Resources	Laboratory	Process development, variability study
2014	Energizer Resources	Pilot Plant	Validation of Flowsheet, collection of engineering data

CERTIFICATE of AUTHOR

This Certificate of Author has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"), Part 8.1.

a) **Name, Address, Occupation**

John Shapton Stanbury

1st Floor, 267 West Building, 267 West Avenue, Centurion, South Africa

Director- Cresco Project Finance (Pty) Limited

b) **Title and Effective Date of Technical Report**

Molo Feasibility Study, National Instrument 43-101 Technical Report

Effective date: 6th February 2015.

c) **Qualifications**

I have gained the following academic qualifications:

B Sc (Eng) (Ind), University of Pretoria. 1967;

MBA. University of Pretoria. 1970;

B Proc. University of South Africa. 1996; and

LLB. University of South Africa. 2000.

I am a Member of the South African Institute of Industrial Engineers and am a registered Professional Engineer with the Engineering Council of South Africa. I have worked in the mining and mineral sector for more than 30 years and have been involved in project evaluation at operational, supervisory and final accountability level.

I have read the definition of "qualified person" set out by NI 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 fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.

d) **Site Inspections**

I have not visited the site

e) **Responsibility**

I am responsible for the preparation of Sections 1.4, 1.12, 1.16, Section 19, Section 22, and Section 25.10.

f) **Independence**

I am independent of Energizer Resources Inc. for the purposes of NI 43-101 applying all of the tests in section 1.5 of NI 43-101.

g) **Prior Involvement**

Prior to preparation of the report, I have not had prior involvement with the property that is the subject of the Technical Report.

h) **Compliance**

I have read and am fully conversant with NI 43-101 and Form 43-101F1, and the aspects of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

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 the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Centurion, South Africa this 23 day of March 2015.

J S Stanbury. Pr Eng

CERTIFICATE OF AUTHOR

This Certificate has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Project ("NI 43-101"), 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: 6th February 2015.

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 since 1995.

I have read the definition of "qualified person" set out by NI 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 fulfill 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 Energizer Resources Inc. in accordance with the application of Section 1.5 of NI 43-101.

g) **Prior Involvement:**

I was responsible for the sections 12 and 14 of the previous NI 43-101 technical report, filed on 12th April 2013

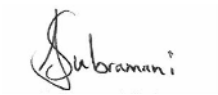
h) **Compliance with NI 43-101:**

I have read NI 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, 23 March 2015



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	Energizer Resources	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 Mineral Projects ("NI 43-101"), 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: 6th February 2015.

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 NI 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 fulfill 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 Energizer Resources Inc. in accordance with the application of Section 1.5 of NI 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 NI 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, 23 March 2015

Philip John Hancox (Pr.Sci.Nat.)

General Manager Africa and Australasia

Caracle Creek International Consulting (Pty) Limited