Zenito

Document:

NI 43-101 Technical Report - Woxna Graphite

112-RPT-GEN-001 Rev 0

Project:

112 - Woxna Graphite Project

Effective date:

09 June 2021

Prepared for:

Woxna Graphite AB

WOXNA GRAPHITE

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Refer to QP Certificates appended for approval.

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STANDARD TERMS AND NOMENCLATURE

The following abbreviations initialisms are used throughout this document.

Abbreviations and initialisms

Acronym	Description
AACE	American Association of Cost Engineers
ABB	ABB Ltd – an automation company
Aminpro	Aminpro Chile S.P.A.
ANFO	Ammonium Nitrate Fuel Mix
AQS	Air Quality Standard
BCM	Bank Cubic Metre
BET	BET analysis – Surface area analysis
BGRIMM	BGRIMM Technology Group
BLS	Barren Leach Solution
BoD	Basis of Design
BWI	Bond Work Index
CAB	County Administration Board
CAPEX	Capital Expenditure
CDA	Canadian Dam Association
CEN	European Committee for Standardization
Cg	Carbon in the form of Graphite
COG	Cut Off Grade
CP	Clarification Pond
CSPG	Coated Spherical Purified Graphite
DD	Diamond Drillhole
DFS	Definitive Feasibility Study
DRI	Drilling
EDS	Energy Dispersive x-ray Spectroscopy
EIA	Environmental Impact Assessment
ELV	Emission Limit Value
EPC	Engineering Procurement Construction
EPCM	Engineering Procurement Construction Management
EQS	Environmental Quality Standards
FS	Feasibility Study
Ga	Giga Annum (billion years)
GA	General Arrangement (drawing)
GARD Guide	Global Acid Rock Drainage Guide
GISTM	Global Industry Standard on Tailings Management
Golder	Golder Associates AB (Sweden)
GruvRIDAS	Swedish mining industry guidelines for Dam Safety
HDPE	High Density Polyethylene
HFO	Heavy Fuel Oil
HPSG	High Purity Spherical Graphite
HV	High Voltage
IBC	International Bulk Container
ICP	Inductively Coupled Plasma

Acronym	Description
ILS	Intermediate Leach Solution
IRR	Internal Rate of Return
LCA	Life Cycle Assessment
LCIA	Life Cycle Impact Assessment
LCM	Loose Cubic Metre
LED	Local Environmental Department
LEM	Leading Edge Material Corp
LoI	Loss on Ignition
LoM	Life of Mine
LoP	Life of Project
LV	Low Voltage
m	Meters
M	Million (except when referring to a currency)
Ma	Mille Annum (million years)
MAC	Mining Association of Canada
mamsl	Metres above mean sea level
MCC	Motor Control Centre
MEND	Mine Environment Neutral Drainage
MIBC	Methyl isobutyl carbinol
Minviro	Minviro Ltd
MPlan	M.Plan International Limited
MV	Medium Voltage
NAG	Non-acid Generating
NGO	Non-Government Organisation
NPV	Net Present Value
NPVS	NPV Scheduler
OPEX	Operating Expenditure
OSF	Overburden Storage Facility
PAG	Potential Acid Generating
PEA	Preliminary Economic Assessment
PEM	Potentially Economic Material
PF	Price Factor
PFC	Power Factor Correction
PFS	Pre-Feasibility Study
PIXE	X-Ray Spectrographic elemental analysis
PLC	Programmable Logic Controller
PSA	Particle Size Analysis
QP	Qualified Person(s)
ReedLeyton	ReedLeyton Consulting
RF	Revenue Factor
RoM	Run of Mine
RoR	Rate of Rise
RQD	Rock Quality Designation
RSS	Rock Substance Strength
SAMWM	Swedish Agency for Marine and Water Management
SCADA	Supervisory Control and Data Acquisition

Acronym	Description
SEM	Scanning Electron Microscopy
SEPA	Swedish Environmental Protection Agency
SGS	Swedish Geological Society
SMHI	Swedish Meteorological and Hydrological Institute
SWEREF99	Swedish datum
TCS	Tailings Consultants Scandinavia AB
TMF	Tailings Management Facility
TSF	Tailings Storage Facility
TSXV	Toronto Stock Exchange Venture
USD	United States Dollar
VAP	Value-add Production
VSD	Variable Speed Drive
WLIMS	Wet Low Magnetic Separator
Woxna	Woxna Graphite AB
WSD	Waste Storage Dump
XRD	X-ray Diffraction
XRF	X-ray Fluorescence
Zenito	Zenito Limited

The following terms and definitions are used throughout this report.

Terms and definitions

Term	Description
Effective Date [1]	The date of the most recent scientific or technical information, included in the Technical Report.
Feasibility Study (FS) [2]	A Feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project, that includes appropriately detailed assessments of realistically assumed mining, processing, metallurgical, economic, marketing, legal, environmental, social and governmental considerations, together with any other relevant operational factors and detailed financial analysis, that are necessary to demonstrate at the time of reporting that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher than that of a Pre-Feasibility Study.
Inferred Mineral Resources [2]	An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.
Indicated Mineral Resources [2]	An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
Measured Mineral Resources [2]	A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.
Mineral Reserves [2]	A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.
Mineral Resources [2]	A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilised organic material including base or precious metals, coal and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

Term	Description
Preliminary Feasibility Study or Pre-Feasibility Study (PFS) [2]	A Preliminary Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on mining, processing, metallurgical, economic, marketing, legal, environmental, social, and governmental considerations and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve.
Preliminary Economic Assessment (PEA) [2]	Means a study, other than a Pre-Feasibility Study or Feasibility Study, that includes economic analysis of the potential viability of mineral resources.
Probable Mineral Reserve [2]	A Probable Mineral Reserve is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.
Proven Mineral Reserve [2]	A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.
Qualified Person [2]	A Qualified Person means an individual who is an engineer or geoscientist with at least five years of experience in mineral exploration, mine development or operation or mineral project assessment, or any combination of these; has experience relevant to the subject matter of the mineral project and the Technical Report; and is a member or licensee in good standing of an approved professional association.
Technical Report [2]	Means a report prepared and filed in accordance with National Instrument 43-101 and includes, in summary form, all material scientific and technical information in respect of the subject property as of the effective date of the technical report.

1 SUMMARY

1.1 Project Summary

The 'Woxna Graphite Project' comprises an advanced, brownfields exploration and development project consisting of:

- defined graphite deposits held under separate exploration permits and exploitation concessions;
- a partially depleted open-pit graphite mine (the Woxna Mine);
- its associated graphite processing facility (the Woxna Concentrator) currently kept on care-and-maintenance; and
- related infrastructure including a tailings storage facility (TSF) and administration buildings.

The 'Woxna Graphite Project' is located near the town of Edsbyn in the Ovanåker Municipality, Gävleborg County, in the Kingdom of Sweden and is owned by Woxna Graphite AB (Woxna Graphite) a 100% owned Swedish subsidiary of international Leading Edge Material Corp (LEM).

The 'Woxna Graphite Project' comprises four exploitation concessions (146.71 ha) over the Kringelgruvan (Kringel), Gropabo, Mattsmyra and Mansberg graphite deposits. Kringel has a total Measured and Indicated Mineral Resource (at a cut-off grade of graphitic carbon (Cg) of 4% Cg) of 2.61 million tonnes (Mt) at a grade of 9.13% Cg, and only Kringel is included in the PEA economic analysis.

The Woxna Mine, meaning the Kringel deposit and the adjacent concentrator and associated infrastructure, was in operation between 1996 and 2001 under various ownerships when eventually production was halted as a consequence of falling graphite prices. Approximately 152 kilo tonnes (kt) of run-of-mine (RoM) were produced for 16.5 kt beneficiated concentrate with an average grade of 92% carbon (Cg). The "Woxna Graphite Project" was acquired by Flinders Resources Limited (Flinders), now Leading Edge Materials Corp. (LEM), in 2011 which subsequently made capital investments into the project towards restarting production. In 2014 the project was recommissioned, however, only a limited test mining campaign at the Kringel deposit was undertaken due to low prices in the graphite market again. The project has since been focussed on developing the necessary downstream processes in order to target higher value growth markets. The purpose of this Preliminary Economic Assessment (PEA) is to investigate the economic feasibility of re-opening the Woxna Mine to feed an expanded and upgraded process plant and value-add downstream processing to produce coated spherical purified graphite (CSPG) as a new active anode material for the emerging European lithium-ion battery industry.

The PEA was designed to economically assess a new conventional open pit mining operation at the Woxna Mine, exclusively based on the Kringel deposit, planned to produce 160,000 tonnes per annum (tpa) over a 19-year project life. No mining production was to be included from the other three deposits, namely the Gropabo, Mattsmyra and Mansberg deposits. The PEA was in addition to investigate the following processing aspects:

- to upgrade the current design throughputs for the historical Woxna Concentrator;
- to develop a process flow and costing for a Value-add Production (VAP) facility to produce high value graphite;
- to develop process flow sheet, resulting in two specific graphite products, namely coated spherical purified graphite and micronized graphite; and
- assess the economic viability of the proposed changes.

In addition, the adequacy of the local infrastructure to support the proposed mine, the sufficiency of the legal permitting as well as the environmental aspects of the project were to be undertaken to highlight any deficiencies at this initial PEA level of investigation.

The results of the PEA are summarised as follows, and are reported according to Canadian National Instrument disclosure Guidelines (NI 43-101) (June 2016) and are signed-off by various independent specialist consultants:

- applying a discount rate of 8% the project pre-tax net present value (NPV) is USD 317.0 million with an internal rate of return (IRR) of 42.9% was estimated;
- a post-tax NPV of USD 248.2 million with IRR 37.4% was estimated;
- The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.
- the Project has an initial capital investment of USD 121.1 million and a life of project (LoP) capital requirement of USD 133.0 million;
- operating costs are estimated at USD 25.1 million per annum (pa) at design throughput;
- an estimate of the likely prices obtainable for the two graphite products at their specific physical and chemical characteristics was provided by Woxna Graphite based on a combination of independent market analysis and in-house historical price information. A CSPG price of 10,000 USD/t was used, and 1,200 USD/t for the micronized graphite;
- the post-tax payback period is 4.51 years from initial investment, or 2.51 years from initial production.
- the four graphite deposits held by Woxna Graphite have been examined at various levels of intensity with Kringel being the most advanced. The Kringel exploration programme was a successful and appropriate combination of geophysical exploration and detailed drilling programmes that resulted in sufficient confidence for the declaration of Measured and Indicated Mineral Resources. The understanding of the relatively simple deposit morphology of tabular, steeply dipping graphite enriched zones within metavolcanic and metasedimentary host units is sound and allows for a relatively simple mine design;
- the economic analysis in the PEA considers only one of the four graphite deposits held by Woxna Graphite, namely the Kringel deposit for which new Mineral Resource estimates have been disclosed within a constraining optimised open pit-shell at 2.61 million tonnes (Mt) Measured + Indicated Mineral Resources at a grade of 9.13% carbon in the form of graphite (Cg) and 0.39 Mt Inferred Mineral Resources at 8.72% Cg. The Mineral Resources of the remaining three deposits have been reported within constraining open pit-shells and represent upside potential for the Project;
- the term Woxna Graphite Project (the Project - without quotation marks) refers to this 2021 PEA including only the Kringel deposit.
- the legal tenure in terms of the exploitation concession for the Kringel deposit is current and, although the extraction/environmental permit currently held by Woxna Graphite is valid, it was granted under now defunct legacy legislation. In order to achieve the potential of this PEA an application has to be made for a new environmental permit under the new Environment Act that would increase the permission to extract 160 ktpa (as opposed to the previous 100 ktpa permit), that would permit expansion of the open-pit and associated TSF- The VAP at its final decided location could potentially be included within the same permit or require its own environmental permit;
- several different approaches to the mining of the deposit were interrogated and the most suitable was selected for both economic and production optimisation. A mine plan and mine production schedule were designed for a conventional open-pit mine at the new production target of 160 ktpa;
- the historical Woxna Concentrator at the Woxna Mine site will be de-bottlenecked and upgraded to process approximately 160 ktpa of run-of-mine (RoM) to produce an average of approximately 14.7 ktpa 14,730 tpa of graphite concentrate at a grade of 92.3% carbon (Cg);
- the concentrate from the Woxna Concentrator will be transported to the VAP. For the purpose of this PEA an existing brownfield industrial facility has been identified in the town of Edsbyn to demonstrate the process flow and economics. Two beneficiated graphite products are to be produced from the VAP. Firstly, approximately 6,604 tpa CSPG at a minimum grade of 99.95% Cg and an average of approximately 7,479 tpa 93% Cg micronized graphite;
- the design and costing of the TSF were undertaken on the basis that two tailings streams will be produced, namely the Non-acid Generating (NAG) and the Potential Acid Generating (PAG) stream. The NAG tailings will represent approximately 90% of the tailings production and will be deposited on the

existing and expanded tailings area, whilst the PAG tailings will represent approximately 10% of the total tailings production and will be deposited in a separate lined TSF adjacent to the existing TSF;
- the Project utilises low-cost hydroelectric power from the national grid, lowering economic costs, whilst also maintaining a comparatively low Project carbon footprint. Water is adequately available for the current Project design and for any future expansions;
- the clean energy supply, and absence of hydrometallurgy using chemical reagents by using thermal purification in the process flow are environmentally advantageous;
- no environmental liabilities are attributable to Woxna Graphite from the previous mining activities and given the date of the initial issuance of the exploitation concession, it is understood by Woxna Graphite that a royalty of 0.2% is applicable to the production; and
- the costing for the Woxna Concentrator and VAP were conducted at ±30% accuracy, the mine plan costing at 30%, whilst those for the TSF were estimated at 50% accuracy;

Future opportunities exist to extend both the project life and the total project production through the development of the other three deposits which are at a relatively advanced stage of exploration and are located within easy access of the processing facilities.

The VAP facility is responsible for substantial economic value add, and this fact may be leveraged via the sourcing of alternate suitable feed materials.

1.2 Introduction

As stated above, the 'Woxna Graphite Project' is an advanced, brownfields exploration and development project consisting of four graphite deposits in Sweden.

Graphite is classified by the European Union as a strategic mineral. Natural graphite of relatively low purity is suitable as a bulk commodity for use within the steel industry. However, in order to meet battery cell manufacturers specifications for use as the active anode material in lithium-ion batteries, the natural flake graphite must by sized, shaped and purified at which point the material is generally referred to as spherical purified graphite (SPG) which is the per-cursor for active anode materials. China dominates the production of SPG and the final coating step is generally done by various battery material producers in Asia.

LEM considers that it is favourably positioned in having access to a sustainable supply of commercial graphite from the Woxna Project at relatively low exploration/assessment costs as the Project has historically verified and current "Canadian Institute of Mining, Metallurgy, and Petroleum" (CIM) compliant Mineral Resources; historical mining production figures for use in future mine design and costing; excellent infrastructure in the form of roads, power, ports, water supply and services; and close proximity to major graphite customers.

In light of this corporate strategy, LEM, through its subsidiary Woxna Graphite, commissioned Zenito Limited (Zenito), an independent firm of engineering consultants, as Project leaders to re-scope the Project at PEA levels of accuracy. The re-scoping and publication of the results is in accordance with Canadian Securities Administrators (CSA) disclosure guidelines as there is a material change in the Project since previous estimations were reported. The PEA focus is the mine design, scheduling and costing as well as the expansion and upgrade of the current and new downstream processing facilities. The specialist consultants or Qualified Persons (QPs) engaged by LEM are summarised as follows:-

- Zenito Qualified Person is Christopher Stinton, BSc (Hons), CEng MIMMM;
- ReedLeyton Consulting Limited (ReedLeyton)- an independent firm specialising in estimation of mineral resources commissioned to review, validate, and update the Mineral Resource statements for the PEA. The ReedLeyton Qualified Person is Geoffrey Reed, B App Sc, MAusIMM (CP) MAIG.
- M.Plan International Limited (MPlan) - an independent firm of engineering consultants commissioned to review and update the PEA mining scope. The MPlan Qualified Person is Mathieu Gosselin, Eng.

- Golder Associates Corporation (Golder) - commissioned to review and update the design of the TSF to accommodate the production from the upgraded Woxna Concentrator and VAP and in addition ESG and permits. The Golder Qualified Person is Henning Holmström, M Sc, PhD, MAusIMM.
- Benchmark Mineral Intelligence marketing report for CSPG, which has been used with permission.

The Qualified Persons have:-

- inspected the Project site, though some of the visits were prior to and not directly related to the PEA, and have all carried out due diligence reviews of the information provided to them by Woxna Graphite for the preparation of this technical report;
- the Qualified Persons have undertaken high level verification where possible and have made reasonable efforts to verify the accuracy of the data relied on in this report;
- none of the consultancies listed above nor any of the specialists in their employ for the preparation of this technical report, have or have had any beneficial interest in the deposits of Woxna Graphite capable of affecting their ability to give an unbiased opinion and, have not and will not, receive any pecuniary or other benefits in connection with this assignment, other than normal consulting fees.
- the listed consultants have been paid fees and will continue to be paid fees for ongoing consultancy input in accordance with normal professional consulting practice; and.
- Woxna Graphite has reviewed draft copies of this report for factual errors, but edits made as a result of these reviews did not involve any alteration to the conclusions made by the Qualified Persons.

1.3 Reliance on other experts

The Qualified Persons have relied on expert opinions and information outside of the Qualified Persons' expertise. Such expert opinions and information was provided by LEM and pertained to environmental considerations, taxation, legal aspects including mineral tenure, surface rights and material contracts and marketing studies.

This information is believed to be essentially complete and correct to the best of the Qualified Persons' knowledge and no information has been intentionally withheld that would affect the conclusions made herein.

1.4 Property description and location

Woxna Graphite, LEM's 100% owned Swedish subsidiary, owns four exploitation concessions and eleven associated exploration permits (see Section 4.5.1) (Figure 1-1, and see Section 4.5.2). The Kringel, Gropabo, Mattsmyra and Mansberg deposits held under exploitation concessions form part of a 40 km mineralisation trend in central Sweden. The relative locations of the four exploitation concessions are illustrated in Figure 1-1. It is important to note that whilst this disclosure contains Mineral Resource estimates for all four deposits, the mining plan and economic analysis for the PEA considers only one of these deposits, namely the Kringel deposit.

The Kringel deposit is located at 61° 26´ 08.94˝N and 15° 36´ 16.31˝E, approximately 8 km west-northwest (WNW) of the town of Edsbyn in the Ovanåker Municipality, Gävleborg County, in the Kingdom of Sweden.

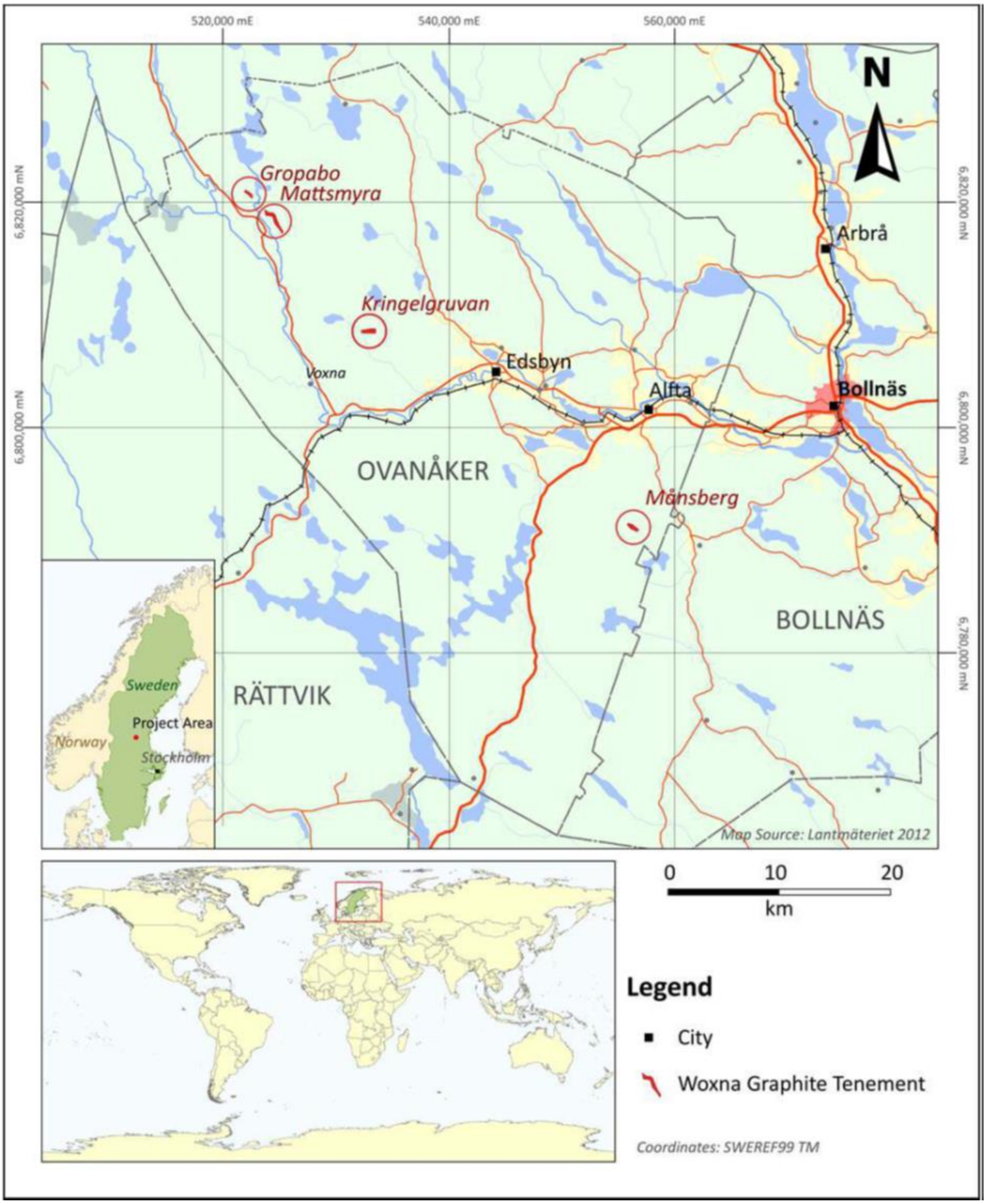


Figure 1-1: Location of the Woxna Exploitation Concessions (ReedLeyton, 2015)

1.4.1 Legal tenure and permitting

The Swedish Minerals Act relates to the exploration and exploitation of certain mineral deposits on land, regardless of the ownership of the land. The Minerals Act provisions apply to a list of mineral substances, concession minerals.

Two types of permits are provided for by the Mineral Act:-

- *Exploration permit* (undersökningstillstånd):- gives access to the land and an exclusive right to explore within the permit area. It does not entitle the holder to undertake exploration work in contravention of any environmental regulations that apply to the area. An exploration permit is initially valid for a period of three years, after which it can be extended up to a total of 15-years if special conditions are met. Compensation must be paid by the permit holder for damage or encroachment caused by exploration work;
- *Exploitation concession* (bearbetningskoncession) gives the holder the right to exploit a proven, extractable mineral deposit for a period of 25-years, which may be extended. Permits and concessions under the Minerals Act may be transferred with the permission of the Mining Inspector. An exploitation concession relates to a distinct area where a mineral deposit is discovered that is technically and economically recoverable. The application for an exploitation concession must be accompanied by an environmental impact assessment (EIA);

In addition to above, in order to commence mining operations a permit under the Environmental Code is also required:

- *Environmental permit*:- under the rules of the Swedish Environmental Code, a special EIA for the mining operation must always be submitted to the Environmental Court, which examines the impact of the operation on the environment in a broad sense. The Court also stipulates the conditions which the operation is to meet. The conditions that require fulfilment for the current Environmental Permit over the Project and VAP areas are provided in Section 20.2.3.

A summary of the Woxna Graphite legal tenure and permitting status of the four deposits is provided in Table 1-1. The valid extraction/environmental permit currently held by Woxna Graphite for the Kringel deposit was granted under now defunct legislation. Whilst the permit is theoretically valid, the environmental QP has recommended that application be made for a permit under the new Environment Act. Such a new application would include permission to extract 160 ktpa (as opposed to the previous permits 100 ktpa), would permit expansion of the open-pit and associated TSF, and potentially as well include the development of the VAP in its finally decided location.

The surface freehold owners are mostly forestry companies. The extent and location of these surface right holders in the vicinity of the Kringel exploitation concession are known to Woxna Graphite and access rights have been verified.

Table 1-1: Woxna Graphite legal tenure

Property	Type of Mineral Tenure	Reference number	Area (ha)	Valid for (years)*	Date of issue or receipt	Expiration date	Conditions/comments
Kringel	Exploration permit	Kringeltjärn Nr 8	157	3+1	29-Nov-19	29-Nov-23	Dnr BS 200-955-2019
		Kringeltjärn Nr 2001	275	3	04-Nov-20	04-Nov-23	Dnr BS 200-446-2020
		Kringeltjärn Nr 2002	413	3	04-Nov-20	04-Nov-23	Dnr BS 200-477-2020
		Kringeltjärn Nr 2003	147	3	04-Nov-20	04-Nov-23	Dnr BS 200-448-2020
	Exploitation concession	Kringelgruvan K Nr 1	30.76	25	03-Nov-16	03-Nov-41	New issue from Mining Inspectorate - Document number Dnr 22-1271-2015
	Environmental permit	Woxna Graphite AB	30.76		Sept/Oct 1992	None	100 kt per annum mineralized material extraction
Gropabo	Exploration permit	Gropabo Nr 5	58.37	3+1	20-Feb-20	20-Feb-24	
	Exploitation concession	Gropabo	18.20	25	21-Feb-00	21-Feb-25	
	Environmental permit	Gropabo 4:1 and Norra Svensbo 14:1	18.2	7	21-Feb-05	21-Mar-12	Mål nr M 3659-02. The condition of issue was that mining would begin within 7 years hence the expiration
Mattsmyra	Exploration permit	Mattsmyra Nr 7	42.18	3+1	20-Feb-20	20-Feb-24	
		Mattsmyra Nr 8	37.54	3+1	23-Oct-19	23-Oct-23	
		Mattsmyra Nr 9	44.19	3+1	23-Oct-19	23-Oct-23	
		Mattsmyra Nr 10	30.26	3+1	21-Nov-19	21-Nov-23	
		Sub-total	154.17				
	Exploitation concession	Mattsmyra	72.97	25	21-Feb-00	21-Feb-25	
	Environmental/extraction permit	Gropabo 4:1 and Norra Svensbo 14:1	72.97	7	21-Mar-05	21-Mar-12	Mål nr M 3659-02. . The condition of issue was that mining would begin within 7 years hence the expiration
Mansberg	Exploration permit	Mansberg Nr 4	65.57	3+1	21-Nov-19	21-Nov-23	
		Mansberg Nr 5	240.86	3+1	21-Nov-19	21-Nov-23	
		Sub-total	306.43				

Property	Type of Mineral Tenure	Reference number	Area (ha)	Valid for (years)*	Date of issue or receipt	Expiration date	Conditions/comments
	Exploitation concession	Mansberg	24.77	25	27-Dec-99	27-Dec-24	
	Environmental/extraction permit	No application filed					

Source: Woxna Graphite 2021

* In June 2020 legislation was introduced that extended existing exploration permits by one year due to the effects of the Covid pandemic.

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

The Project is accessible from the tarred east west Route 301, which is an all-weather road. Local access to the Kringel exploitation concession is on unsealed all-weather forestry roads.

The operating season is all year round, with possible short and minor disruptions at the height of winter with snowfall and very low temperatures.

The climate is comparatively temperate and is typical of Fennoscandia with cool summers and cold winters. At Edsbyn, some 10 km to the southeast of the project area, the monthly average minimum temperature ranges from -8°C to +11°C and the range of average maximum monthly temperatures is -1°C to +23°C. Edsbyn receives from 30 to 70 mm of precipitation per month with fall and winter typically drier and spring and summer typically wetter periods.

Local services, in terms of machine and engineering plant maintenance, are available in Edsbyn. Regional road, rail, and service infrastructure is well developed. All industrial needs, services, provisions, and supplies are readily and commercially available. They are of high standard, typical of the modern industrial economy that is Sweden.

The national power grid extends throughout the region. Connected grid power is available at the Kringel processing plant and open pit.

Telecommunications including internet and mobile telephone services are widely available at the processing plant.

Water resources are adequate. The site water is supplied from a local source.

Sweden has a long history of mining and local and specialised labour is widely available. Skilled local labour to resource the Woxna operation is expected to be available, some of whom may even have worked at the project previously.

The Project elevation ranges from approximately 220 to 280 metres in relief and comprises NW-SE orientated low hills with trellised local stream drainage and numerous freshwater lakes, of which the Råttjärnasjön and Loftssjön are the largest. These, in the main, ultimately flow to the Woxnan River which is an incised meandering river to the south of the Kringel mineral claim. The Woxnan River is the source of hydropower in the district.

1.6 History

The initial discovery of graphite in the region was made in 1983 by a prospector engaged by the Swedish Geological Survey (SGU) as part of a regional mapping programme directed at uranium detection using airborne radiometric surveys. Following discovery of the deposits in the early 1980s, exploration proceeded under the direction of the precursors to the SGU, namely Sveriges Geologiska AB (SGAB) and Nämnden för statens gruvegendom (NSG). The historical 'Woxna Graphite Project' was explored and developed by various owners as summarised in Table 1-2:-

Table 1-2: Historical ownership and exploration of the Woxna Graphite deposits

Deposit	Date	Investigating Company	Exploration technique	Ownership
Kringel	1986	Sveriges Geologiska AB (SGAB) Nämnden för statens gruvegendom (NGS)	Slingram geophysical ground survey	Precursors to the Swedish Geological Survey (Sveriges Geologiska Undersökning - SGU)
	1987		Trenching	
	1988		Drilling	
	1989		Second drilling campaign	
	1989		Additional ground geophysics to extend the target area	

Deposit	Date	Investigating Company	Exploration technique	Ownership
	1992			Disposal by SGU to Mineral Resources AB (MIRAB)
	N/A			Disposal by MIRAB to Tricorona AB
	1996		Brought into production by Tricorona AB	Tricorona AB
	2001		Production ceased	
				Acquisition by Flinders Resources Limited
	2011	Coffey Mining Limited		NI43-101 Mineral Resource disclosure by Flinders for the Kringel deposit
Mattsymyra	1983	SGAB and NGS	Airborne EM	
			First drilling campaign	
	1989		Second phase drilling	
	1992			Disposal by SGU to Mineral Resources AB (MIRAB)
	Unknown			Disposal by MIRAB to Tricorona AB
Gropabo	1983	SGAB and NGS	Discovered by airborne EM geophysical survey	
	1991		First drilling campaign	
	1992		Second phase drilling campaign	
	1992			Disposal by SGU to Mineral Resources AB (MIRAB)
	Unknown			Disposal by MIRAB to Tricorona AB
Mansberg	1983	SGAB and NGS	Discovered by airborne EM geophysical survey	
	1991		First drilling campaign	

1.7 Geological setting and mineralisation

The geological history of Sweden is highly complex and includes at least four periods of cratonic stability with sedimentary basin development separated by at least six orogenic, mountain building events and associated magmatic activity. Broadly, the geology of Sweden consists of the following main components (see Figure 7-1):

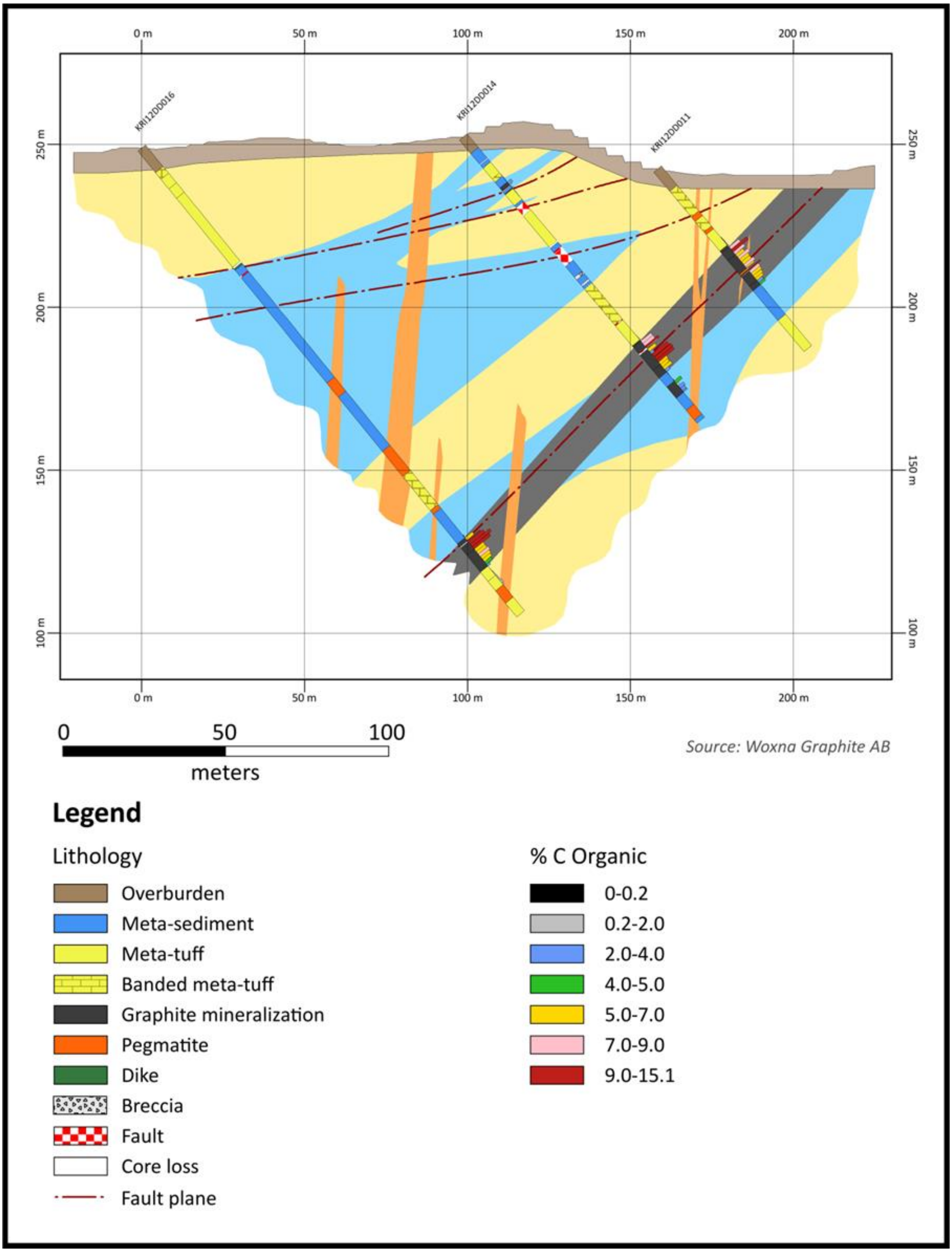
- *Precambrian basement* of gneisses and greenstone belt which forms the western part of the Fennoscandanavian stable craton (>2.5 giga annum (Ga)) in the extreme north of the country affected by the oldest identified orogenies namely the Archaean Lopian (2.8 Ga to 2.6 Ga) and Sveco-Karelian (2.0 Ga to 1.8 Ga) mountain building events;
- *younger metasedimentary and metavolcanic sequences* which are the metamorphosed remnants younger sedimentary and magmatic (2.0 Ga and 1.65 Ga) sequences deposited on top and within this ancient crust. The sedimentary units are grouped as the Svecofennian Province which occur in central Sweden. Numerous magmatic complexes also intruded into the stable craton, such as the Transcandanavian Igneous Belt (TIB) located in the west of the country. These sequences were affected by Paleo-Mesoproterozic mountain building process known as the Gothian (1.7 Ga to 1.5 Ga); Hallandian (1.5 Ga to 1.4 Ga) and Sveconorwegian (1.1 Ga to 0.9 Ga) orogenies;

- *remnants of Scandinavian Caledonides*, a sequence of metasediments and metavolcanics deposited in the Lapetus Ocean (the predecessor of the present-day Atlantic Ocean) dated. 700 Ma to 400 Ma which occur as nappes thrust onto the Fennoscandanavian craton;
- Phanerozoic sedimentary sequences of shales, limestone and sandstone occur in southernmost Sweden and are remnants of Cambro-Silurian cover aged between 540 Ma and 420 Ma.
- *Tertiary age sediments* formed between 250 Ma and 55 Ma occur in the most southerly and southwestern parts of Sweden (Skåne).

1.8 Local geology

The local geology for all four of the Woxna Graphite deposits has been reported in this technical document in support of the re-estimation of the Woxna Graphite Mineral Resources in totality. However, for the purposes of the PEA, the Kringel graphite deposit local geology is summarised as follows:-

- the mineralisation is hosted by a sequence of steeply dipping metasedimentary and metavolcanic units metamorphosed to sillimanite grade and intruded by igneous units ranging from alkali pegmatite to granite;
- the local geology within the exploitation area is dominated by steeply-dipping, calcareous quartz-rich meta-tuff, with interbedded metasedimentary units and cross-cutting pegmatite;
- trace to massive graphite mineralisation in two discrete tabular zones is developed in association with pegmatitic intrusions (see Figure 1-2);
- the mineralisation is tabular in shape, and occurs late in the structural history, postdating and cross-cutting any remnant tectonised and metamorphosed lithologies;
- the graphite is considered hydrothermal/metasomatic in nature; and
- the Kringel deposit area has variable cover of 2 m to 15 m of Quaternary age glacial moraine.



Source: Woxna Graphite AB 2013

Figure 1-2: Cross section of the Kringel deposit

1.9 Deposit types and mineralisation

Graphite is a naturally occurring allotrope of carbon produced by the metamorphism of organic material originally deposited as sediment or mixed with sediment. As organic material is metamorphosed, hydrogen and oxygen are driven off as water, leaving the carbon behind to form graphite. On an atomic level, carbon atoms are arranged in sheets and held together by strong, covalent chemical bonds. These carbon sheets are only weakly bonded together with those above and below by surface attractions.

The following types of graphite are found in nature:-

- *Flake graphite*:- crystalline small flakes of graphite which occurs as isolated, flat, plate-like particles;
- *amorphous graphite*:- very fine flake graphite is sometimes called amorphous;
- *lump graphite (or vein graphite)*:- occurs in fissure veins or fractures and appears as massive platy intergrowths of fibrous or acicular crystalline aggregates, and is probably hydrothermal in origin;

Furthermore, *highly ordered pyrolytic graphite* refers to graphite with an angular spread between the graphite sheets of less than 1°.

The mineralisation occurring within the Woxna Graphite exploration permits and exploitation concessions comprises metasomatically/hydrothermally formed graphite in association with prominent pegmatitic intrusions into steeply-dipping, calcareous quartz-rich meta-tuff, with interbedded metasedimentary material. Essentially the morphology of the mineralisation conforms to that of the host units resulting in tabular orebodies in the vicinity of cross-cutting pegmatites (see Figure 1-2).

The nature of the graphite mineralisation at Kringel is summarised below:

- the Kringel mineralisation was intersected on all the drilling sections suggestive of continuous mineralisation over the concession area;
- the mineralisation is tabular and conformable with steeply dipping host metasediments and metavolcanics;
- the mineralisation is known to extend to at least a depth of 150 m below the surface;
- the mineralisation strikes eastwest (E-W), and dip varies between 60 ^{0}and 80^0 to the south;
- grade distribution varies both laterally and vertically;
- based on grade distribution six main higher grade Type A zones have been identified with a Type A cut-off grades of +7% Cg;
- outer, lower grade Type B (<7% Cg) domains were identified within which 11 small, mineralised envelopes/bodies exist;
- the mineralised envelopes/bodies vary in width 5 m to 15 m (averaging 10 m);
- coarse, medium, and fine-grained graphite is developed as blebs in monomineralic zones (Figure 8-1, Figure 8-2).

It is the opinion of the Reedleyton that the nature and genesis of the mineralisation at all 4 deposits is adequately understood and that the exploration programmes conducted were appropriate to the style of the deposit (with the addition of the geophysical data) and the genetic geological model. The exploration programme has proved successful and can be adapted to other graphite deposits in the region.

The overburden is composed of Quaternary aged glacial moraine ranging in thickness from 0.5 m to 20 m over Kringel area with an average thickness of 3.5 m.

1.10 Exploration

The historical exploration of the Kringel deposit has been summarised in Table 1-2. Systematic exploration took place from 1985 onwards using geophysics, in the form of electromagnetic (EM) and other techniques, as the primary anomaly definer. Very Low Frequency (VLF) Slingram methods at 3.6 kHz and 60 m coil spacing proved to be the optimal setting and the deposit was covered at a 100 × 80 m to 200 × 80 m profile spacing.

Follow-up diamond drillhole (DD) drilling took place from 1988–1989 comprising 51 DDs on 6 cross-sections in local grid coordinates extending over an approximate strike length of 600 m with drillhole spacing at a nominal 20 × 50 m for a total of 2,909 m. All core from this campaign was sampled on a continuous basis and submitted for analysis to the Sveriges Geologiska AB in Luleå using the Leco thermal IR (infrared) methodology for carbon and sulfur content.

This historical drilling and sampling campaign was investigated by the Reedleyton (Flinders 2013, 2015) and found to be appropriate to the style of mineralisation and reliable for Mineral Resource estimation. No information is available for the core recovery of the historical programme, but visual inspection of remnant core indicates that sample recoveries were <95%.

During 2012 Flinders completed a DD campaign of 41 DDs for a total 3,673 m over the Kringel exploitation concession at a drillhole spacing of 50 m and a drillhole core size of 42 mm. Drilling was undertaken by a contractor independent of Flinders and sampling was supervised by suitably qualified personnel.

The drillhole core was sampled at an average of 1 m intervals and sampled continuously based on the logged geology. Core recovery from Flinders drilling campaign was generally >95%.

Combined with the historical data, the total number of drillholes is 92 for a total length of 6,581 m. Data and plans relating to the collar locations, drillhole collar orientations and drillhole surveys for both the historical SGU data and the 2012 Flinders campaign were examined by the Reedleyton and most drillhole collars were located in the field.

1.11 Drilling

The historical and current 2012 Flinders drilling campaigns are briefly described in Section 1.10. All remnant drill core, after sampling, is stored in wooden core boxes at the Woxna Mine site which is a key access only facility, and there is no evidence that samples have been disturbed in any way since cutting.

1.12 Sample preparation, analyses and security

The core samples taken at site were transported by truck to ALS Chemex preparation laboratory in Piteå for core cutting, where core was split by diamond saw under supervision by Flinders staff. The cutting of core at ALS Chemex laboratory in Sweden is in keeping with industry practice, and security of the delivery chain is more than adequate (Flinders 2015). One half was retained as a check sample and the remainder bagged for analysis. The core was cut in consideration of the main foliation/banding of the rock.

The retained core was examined by the Qualified Person in 2013 and was considered fresh with little or no secondary mineralisation.

The laboratory that completed analysis of the samples from the historical drilling programme was the Government owned SGAB ANALYS. Both the laboratories that carried out the sample preparation and analysis were independent of Woxna Graphite. No details of certification by any standards associations and the particulars of any certification are known, however the laboratories were well regarded and applied best practice of the day.

The historical drilling programme resulted in the following analytical dataset:-

- 374 graphite analyses by Leco analyser;

- 52 sulfur assays by Leco analyser;
- a selection of samples analysed by whole rock Inductively Coupled Plasma (ICP) for major and minor elements and loss on ignition (LoI); and
- 25 samples density measurements using the Archimedes (water immersion) method by the petrophysical laboratory of Sveriges Geologiska AB.

The Flinders drilling programme resulted in the following dataset:-

- 1,345 graphite analyses by Leco analyser;
- between 33% and 50% of the samples were analysed for sulfur;
- 8% of the samples were assayed by ICP-MS for major and minor elements;
- blank samples were inserted at a rate of 1 in 15;
- density measurements were conducted by Flinders staff using the Archimedes method amounting to 1,424 measurements covering each assay interval as well as the lithologies in the foot and hanging walls.

1.13 Data verification

The assay data in original laboratory sheets from the 1988 and 1989 drilling programme has not been examined by ReedLeyton. The paper records for collar, assay, survey, and geology data were digitised for the 2002 Claesson et al historical Mineral Resource estimate for Kringel and this dataset was compared to the digital data available to Woxna Graphite. Claesson (2002) concluded that the historical data is of sufficient quality and traceable provenance that it is useable as exploration data. The then supervising geologist, who is a QP under current NI 43-101 protocol, also verified the provenance of the data supplied. The historical data for Kringel, Mattsmyra and Gropabo was independently reviewed by Coffey Mining Limited for a Mineral Resource estimate published in 2011 and was considered appropriate for use in a resource estimate. ReedLeyton concurs with this conclusion based on its own verification of the historical data for Kringel, Mattsmyra and Gropabo.

Following the acquisition of the historical database by Flinders, the capture of 2012 drilling campaign digital data was completed by Flinders staff. Hard copy data has been verified and all data is stored in a database and managed by Woxna Graphite. The digital data has been both randomly and systematically checked by ReedLeyton and shown to be correct using a number of checks.

A sampling verification exercise was conducted by ReedLeyton on the 2012 drilling samples which included 59 samples from 12 drillholes retained under ReedLeyton's supervision, and personally delivered to the ALS Chemex laboratory manager for further processing and transport.

The results of the verification exercise showed extremely good agreement between the individual samples.

All QA/QC data for this Project has been deemed acceptable for the purposes of the Mineral Resource estimation (ReedLeyton 2021).

1.14 Mineral processing and metallurgical testing

Several series' of testwork have been carried out on samples of mineralised material and flotation concentrates before and since 2012 by various laboratories and specialist consultants. These testwork programmes included the following:

- graphite flotation including locked cycle flotation tests;
- dewatering;
- upgrading the flotation concentrates by spheronizing;
- hydrometallurgy; and
- pyrometallurgy.

Table 1-3 summarises the testwork since 2012.

Table 1-3: Post 2012 Testwork Programmes

Date	Tests
27/12/12	a) Dewatering concentrate
Aminpro 13/07/13	• Metallurgical Testwork • Front End Engineering
BGRIMM 01/06/16	a) Flotation b) Hydo-purification on concentrate
2016	a) Grinding b) spheronization
2016	a) Grinding b) spheronization
15/11/16	a) Flake Analysis
02/03/17	a) Standard Sample Analysis b) Spheronization
08/05/17	a) Standard Sample Analysis b) Spheronization c) Purification d) Cell Assembly and Testing
05/01/17	a) Comminution & shaping
09/02/18	a) High gradient mag sep to remove iron containing minerals
10/10/18	a) Micronizing b) spheronization
30/01/19	a) 17 heat treatments
30/01/19	a) Purification
	a) Modal composition b) Elemental comp c) Elemental distribution d) Mineral association e) Mineral locking f) Particle size distribution g) Mineral grain distribution
29/05/20	a) Waste Characterization b) Waste Classification
02/06/20	a) Produce d_{50} of 4µm

Refer to Section 13 for further detail.

1.15 Mineral resource estimates

Historical Mineral Resource estimates (*L-A Claesson, 2002)* have been reported by Tasex 2011 and Flinders 2013 and 2015 for the four Woxna graphite deposits. The historical Mineral Resource estimates have been provided in Section 6.1.1.4.

The Historical Mineral Resource estimates (*L-A Claesson, 2002* and Previous Mineral Resource estimates (Flinders 2015) for the four mineral deposits were developed without the constraint of an applied mine plan and open-pit shell. In the light of more rigorous compliance requirements, the Mineral Resources were reported by ReedLeyton within the constraints of the 2021 PEA mine plan as a means of demonstrating "reasonable prospects for economic extraction" as required by numerous international reporting codes. No new exploration data was included in the reporting process.

All of the estimates were prepared in accordance with the NI43-101 Standards of disclosure and the classification of levels of confidence are considered appropriate on the basis of drillhole spacing, sample interval, geological interpretation and all currently available assay data.

The Kringel Mineral Resourcane was based on:-

- 92 DDs totalling 6,581-metres. Approximately 45% of the DDs are considered historical with no information regarding the drilling protocols applied at the time. Nonetheless, the Qualified Person reviewed the drillhole collar positions in the field, interrogated the historical database, examined the drillhole data in the light of historical production information and undertook a sampling verification exercise and is satisfied that this geological information can be relied upon in a mineral resource estimate;
- the integrity and accuracy of the database has been both randomly and systematically checked by ReedLeyton and shown to be correct;
- the Kringel drilling campaign was conducted on a 50 × 50 m grid and drillholes drilled mostly at two orientations 12^0-degrees and 348^0. For wire framing purposes the strike direction of the mineralisation varied between 80^0 to 100^0 and a 90^0 strike was considered the optimal orientation.
- a total of 374 samples from the 51 historical drillholes and a total of 1,433 samples from the 41 2021 drillholes were analysed in total. The sample preparation and analysis were undertaken by independent laboratories of good reputation. No quality control/quality assurance data is available for the historical data however the verification sampling campaign provided assurance as to the accuracy, reproducibility, and reliability of the historical data;
- geological wireframes were generated from the logging information;
- composites of the drillhole samples were generated at 1 m intervals to honour the geological wireframes and were separately verified;
- ReedLeyton applied the density values provided by the analytical campaigns and the density determinations are considered suitable for the deposit type and of sufficient quantity to be representative;
- the mineralisation is generally tabular in nature conforming with the host sequences in the region. A strike bearing for the mineralisation was 90^0;
- a single block model was constructed with a parent block size of 5 × 25 × 5 m with sub blocks at 1.25 × 5 × 1.25 m. An offset of 1500 × 600 × 400 m was applied;
- an estimation area of an area approximately 1,200 m by 100 m to 200 m;
- intersection of mineralisation in all drillholes suggestive of continuous mineralisation throughout the area;
- depth determination of 150 m below surface;
- mineralisation dip of 60^0 to 90^0 S;
- mineralisation thickness varying between 5 m and 15 m averaging 15 m;
- the mineralisation grade distribution is not uniform and higher grade mineralised envelopes or bodies were identified according to cut-off grades;
- six main higher grade Type A zones were identified with a cut-off grade >7% Cg;
- outer, lower grade Type B domains <7% Cg were identified within which 11 small, mineralised envelopes/bodies exist;
- faulting is present although no geological loss was determined;
- the mineralisation remains open laterally and at depth;
- grade interpolation was undertaken using inverse distance defined by the domain wireframes. The allocations of composites were calculated using a hard boundary at the domain wireframes;
- the search parameters applied in the estimation have been provided;

- a cut-off grade of 7% Cg was applied to the Mineral Resource estimation modelling to define Type A and Type B Graphite zones. A grade cut off of 4% Cg has been applied to the Mineral Resource estimation for reporting purposes of Type A and Type B Graphite;

MPlan 2021 prepared constraining pit shells for ReedLeyton to be used for Mineral Resource estimation reporting of the Kringel deposit using optimised pit shells generated using Datamine™ NPVS software. The key assumptions used in the generation of the resource constraining pit shells for the Kringel deposit were:

- overall slope angle for resource pit shell: 55 degrees;
- mill cut-off grade = 4.00%;
- break even cut-off grade = 4.21%;
- process cost: USD 84.18/t mill feed
- dilution 2.5%
- mining recovery 97.5%
- process recovery 93.7%

The cut-off grades assumed:

- average value-added graphite price of: USD 2,320;
- recovered value of USD 2,103/t after applying costs, taxes, mining, and process recovery factors; and
- mining cost: USD 4.51/t rock mined

Note that the Kringel mine depth is limited to 70-metres (applied due permitting), and by the mining concession limits at its geographic boundary. The proposed mine plan fully exploits the available resource within these limits and is therefore relatively insensitive to other factors that might otherwise expand the pit, such as an increase in product pricing, meaning the pit shells generated using lower product pricing remain applicable. The Kringel deposit Mineral Resource estimate is provided in Table 1-4 and Table 1-5 and the plan view extent of the resource shown in Figure 1-3

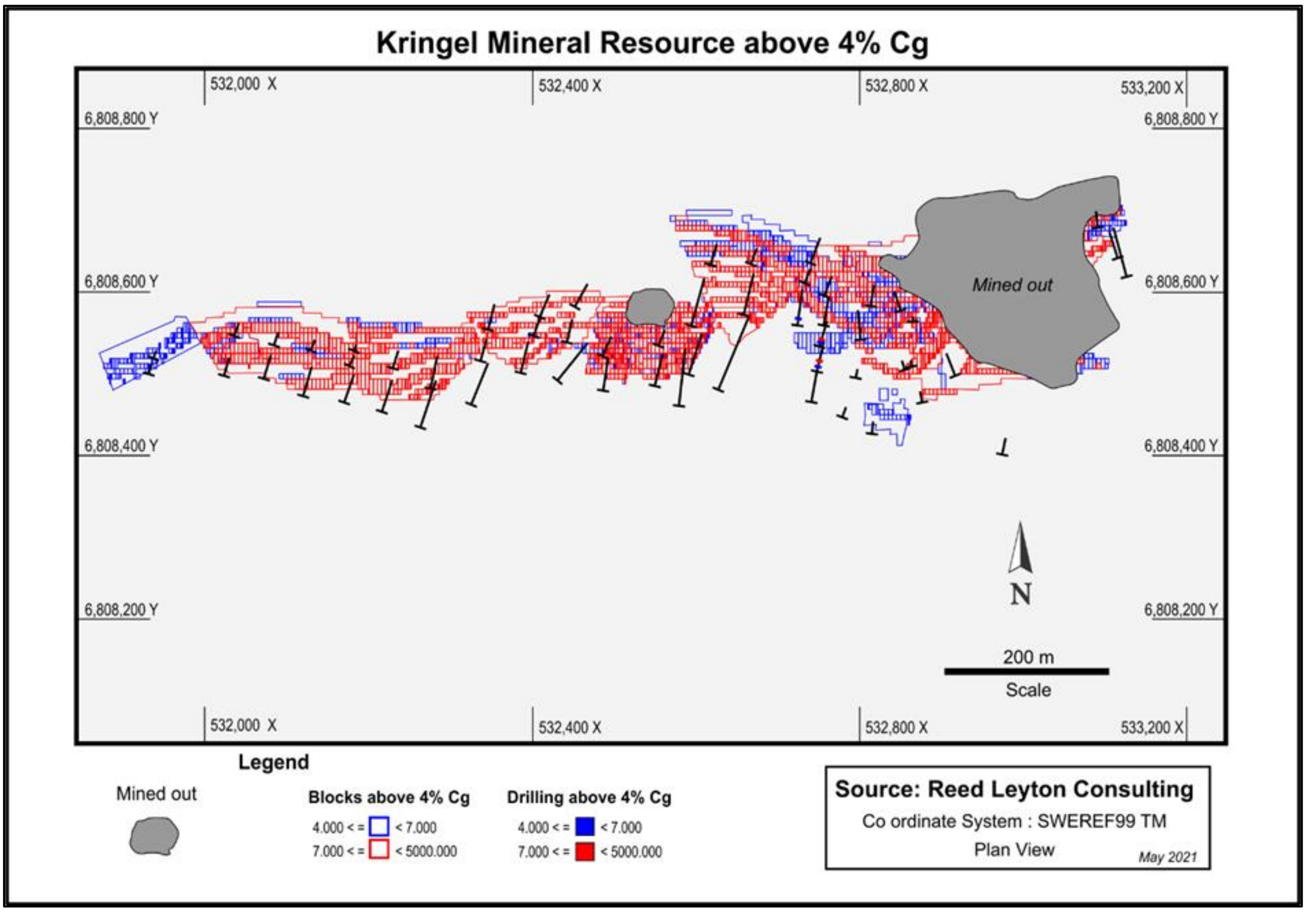


Figure 1-3: Mineral Resource – Kringel

Table 1-4: Kringel Measured and Indicated Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Measured	0.96	9.21
Indicated	1.65	9.09
Total	2.61	9.13

Source: ReedLeyton 2021

- Reported according to CIM Definition Standards 2014
- No geological losses applied
- Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.7 t/m^3 applied to Graphite Domains.
- 4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
- The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

Table 1-5: Kringel Inferred Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	0.39	8.72
Total	0.39	8.72

Source: ReedLeyton 2021

- Reported according to CIM Definition Standards 2014
- Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.7 t/m^3 applied to Graphite Domains.

- 4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
- The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

The previous Kringel Mineral Resource estimate (Flinders 2015) only reported Type A graphite above a 7% Grade cut off and the Kringel Mineral Resource estimate was also modelled using a 7% Grade cut off without any applied mining parameters. The estimation resulted in Measured and Indicated Mineral Resources being reported.

The current Kringel Mineral Resource estimate now reports Type A and Type B graphite to a 4% cut-off off. The Kringel Mineral Resource estimate is now constrained by the Mplan 2021 pitshell for reporting purposes. The estimation resulted in Measured, Indicated and Inferred Resources being reported.

The estimation procedures and verification for the Gropabo, Mattsmyra and Mansberg deposits are provided in detail in Section 14 and results are summarised below. The reporting of the Mineral Resource at a lower cut-off grade than previously applied in 2015 has resulted in expected increases in tonnage and lower grades.

Table 1-6: Mattsmyra Indicated Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Indicated	5.83	7.14
Total	5.83	7.14

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.82 t/m^3 applied to Type A Graphite and Density of 2.86 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

Table 1-7: Mattsmyra Inferred Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	1.51	8.06
Total	1.51	8.06

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014

Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.82 t/m^3 applied to Type A Graphite and Density of 2.86 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

Table 1-8: Gropabo Indicated Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Indicated	2.33	7.72
Total	2.33	7.72

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.81 t/m^3 applied to Type A Graphite and Density of 2.83 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

Table 1-9: Gropabo Inferred Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	0.61	8.07
Total	0.61	8.07

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014

Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.81 t/m^3 applied to Type A Graphite and Density of 2.83 t/m^3 applied to Type B Graphite

4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell

The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

It is the opinion of ReedLeyton that these Mineral Resource estimates for Kringel, Mattsmyra and Gropabo satisfy the definitions Measured, Indicated and Inferred Mineral Resources as per the CIM Definition Standards 2014. The estimate for the Mansberg deposit remains historical in nature and will be superseded when additional exploration and verification has been undertaken.

1.16 Mineral reserve estimates

Mineral Reserve has not been estimated for the Project given the PEA level of project definition.

1.17 Mining Methods

The mine design, scheduling and costing for the Woxna Mine was undertaken by independent mining specialist MPlan. The selection of the most appropriate and cost effective method of mining was based on both historical information, the characteristics of the mineralisation and testwork results. A Basis of Design (BoD) was prepared which provides an effective tool to clearly present the 2021 decisions, assumptions and specifications that were used to develop the mining scenario for Kringel deposit at the Woxna Mine.

The methodology used for selection of the most appropriate mining method for the Kringel deposit comprised the following three interconnected stages:

- Stage I: interrogated many site-specific parameters such as rock properties of waste and mineralised material;
- Stage II: selection of the top-ranked mining methods;
- Stage III included a more detailed examination of the top-ranked mining methods in terms of mining selectivity, planned production rate and economic considerations.

The Kringel deposit geometry combined with a relatively flat topography favours surface mining methods which was the historical method of mineralisation extraction. Different mining methods were considered, and two approaches were used to rank the mining methods and to evaluate their technical feasibility and relevance, namely the University of British Columbia (UBC) method and the Nicholas method. The top-ranking method proved to be surface mining at the planned mining production rate objective of 160 kt RoM per annum delivered to the primary crusher or RoM pad stockpiles. The reference point at which RoM is defined is at the point where the RoM is delivered to the Woxna Concentrator, i.e. primary crusher or RoM pad stockpiles.

Kringel LoM production plan was scheduled from the most economical part of the deposit to the lesser areas and the following steps were undertaken:

- NPV Scheduler pit optimisation of the geological 3D block model;
- mining plan starting point and mine sequence was matched to the NPV Scheduler preliminary results using different Revenue Factors (RF); and
- LoM plan scheduled in Datamine™.

The life of mine (LoM) is 15-years, and the life of project (LoP) is 19-years with the Woxna Concentrator feed from stockpiles in the last few years. A RoM graphite grade of 10.2% C on average over the first 15-years of production is planned. During that time, Type B mineralised rock will be stockpiled in order to be processed during the final years of the LoP. In total the LoP production plan will generate approximately 258,152 MtC in concentrate.

The stripping ratios vary throughout the scheduled LoM and range from 7.84 in Year 1 reducing gradually to 1.52 in Year 15 (see Table 1-11).

The overall mine design parameters are summarised in Table 1-10. A slope design safety analysis was undertaken to determine an optimal slope that will achieve the best safety factor as well as ensuring the best mineralisation mining recovery, and financial return in the context of maximising graphite production. The pit slopes will incorporate single benches with face height to a maximum 10 m in the waste for a safe mining operation. The graphite mineralisation benches height will depend on the morphology of the mineralisation domains and will have a maximum height of 5 m, in accordance with the size of the excavator selected for mining. Overall slope angles achieved for the Woxna Mine open-pits will be flatter than the maximum inter-ramp angle listed, due to the inclusion of access ramps and safety berms. The resultant open-pit design is illustrated in Figure 1-5 and Figure 1-6

Table 1-10: Overall open-pit design parameters

Parameter	Unit Symbol	Value
Maximum bench height in overburden	m	10
Maximum bench height in graphite	m	5
Face angle	°	80
Berm width	m	5
Ramp width	m	15
Ramp gradient	%	10
Final slope angle	°	62
Minimum mining width	m	40

The mine will have in average 350 work-days annually. The mining workforce will work on 8-hour shifts, with two shifts per day, seven days per week and three rotating crews working for continuous coverage.

The RoM will be extracted from the open-pit within the Kringel exploitation concession as shown in Figure 1-4. All beneficiation and administration infrastructure are located northwest of the concession area for which surface rights permission has been secured. The mine maintenance area is located near the concentrator offices on the access road to the mine site. The explosive storage is located north of this access road within prescribed safety distances.

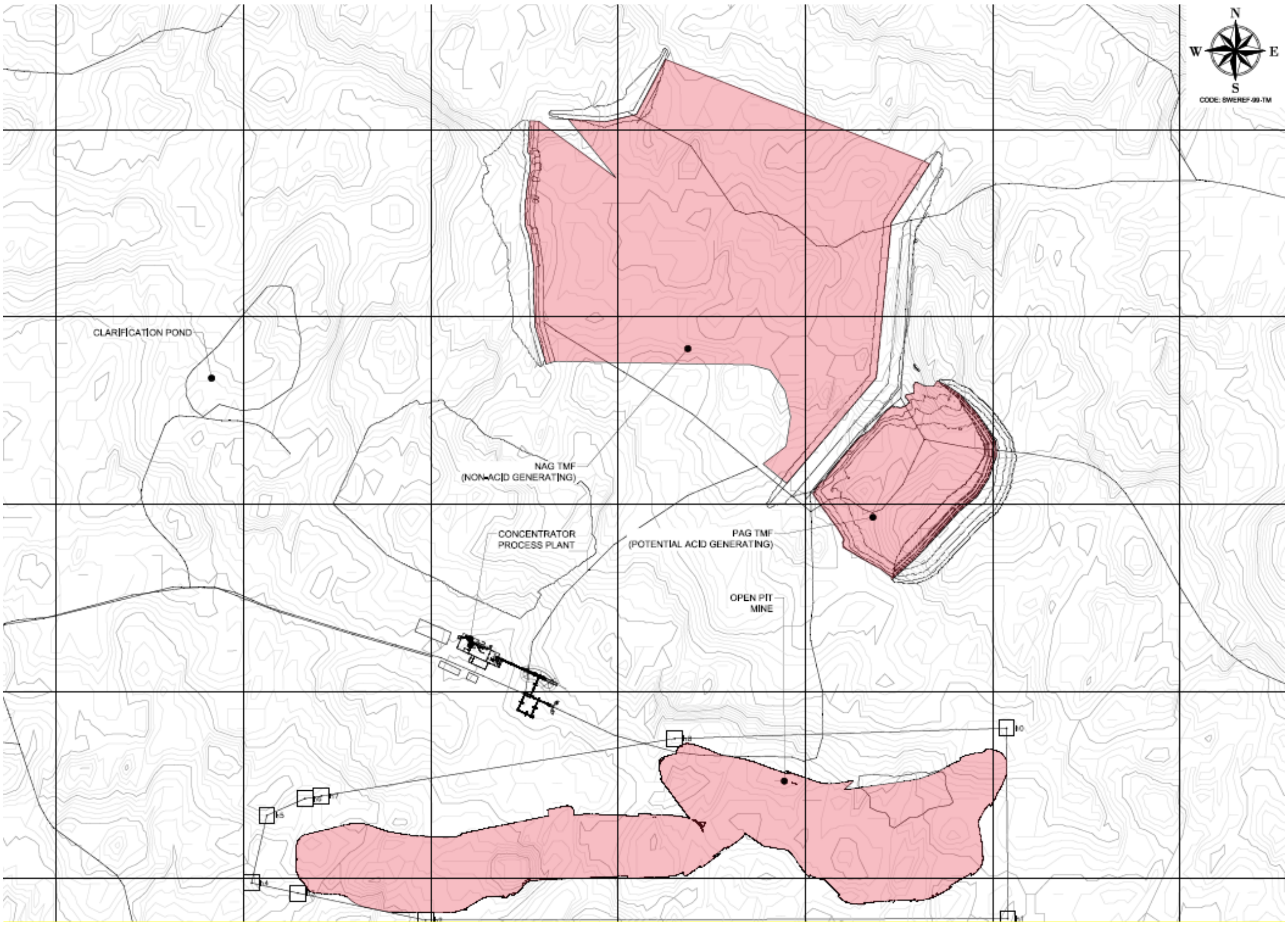


Figure 1-4: Site plan for the Kringel Mining and Beneficiation facilities

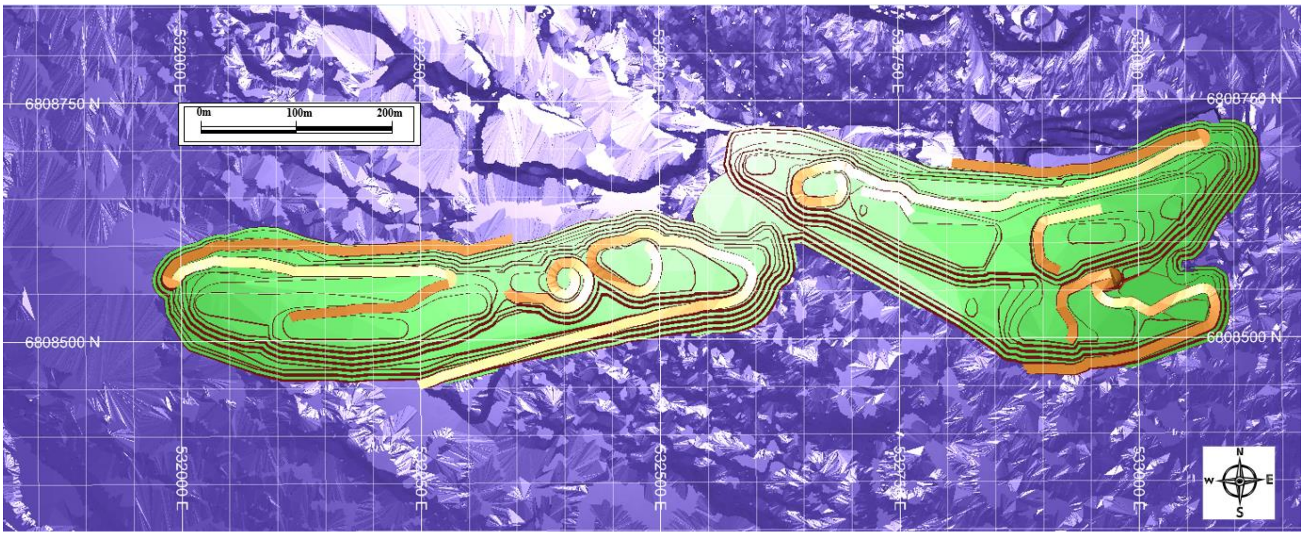


Figure 1-5: Pit Designs used as Basis for Mining Schedule

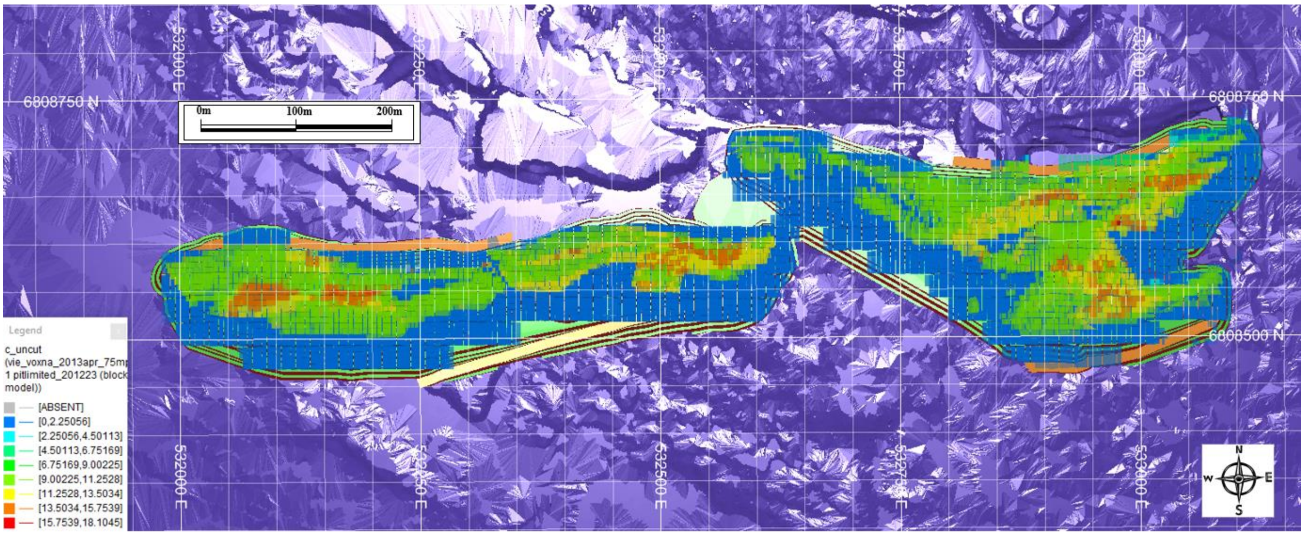


Figure 1-6: Pit design limited Block Model showing %Carbon Grades

The Woxna Mine mining schedule, incorporating the planned stockpiling strategy for Type B material is shown in Table 1-11 below. The mine plan begins with the mining of the east pit at a relatively high stripping ratio of 7.84 waste tonnes for each tonne of mill feed to give access to 160 kt of Type A graphite in year one. The mine plan is designed to provide 160 kt of high-grade mill feed to the mill every year until it runs out in the 16th year after which the lower grade Type B graphite that is stockpiled over the first 15-years is then fed into the mill. The production plan for the process plant continues until year 19 when the low-grade mill feed from the stockpile is exhausted.

Table 1-11: Kringel Planned Mining Production Schedule before Stockpiling & Process Schedule

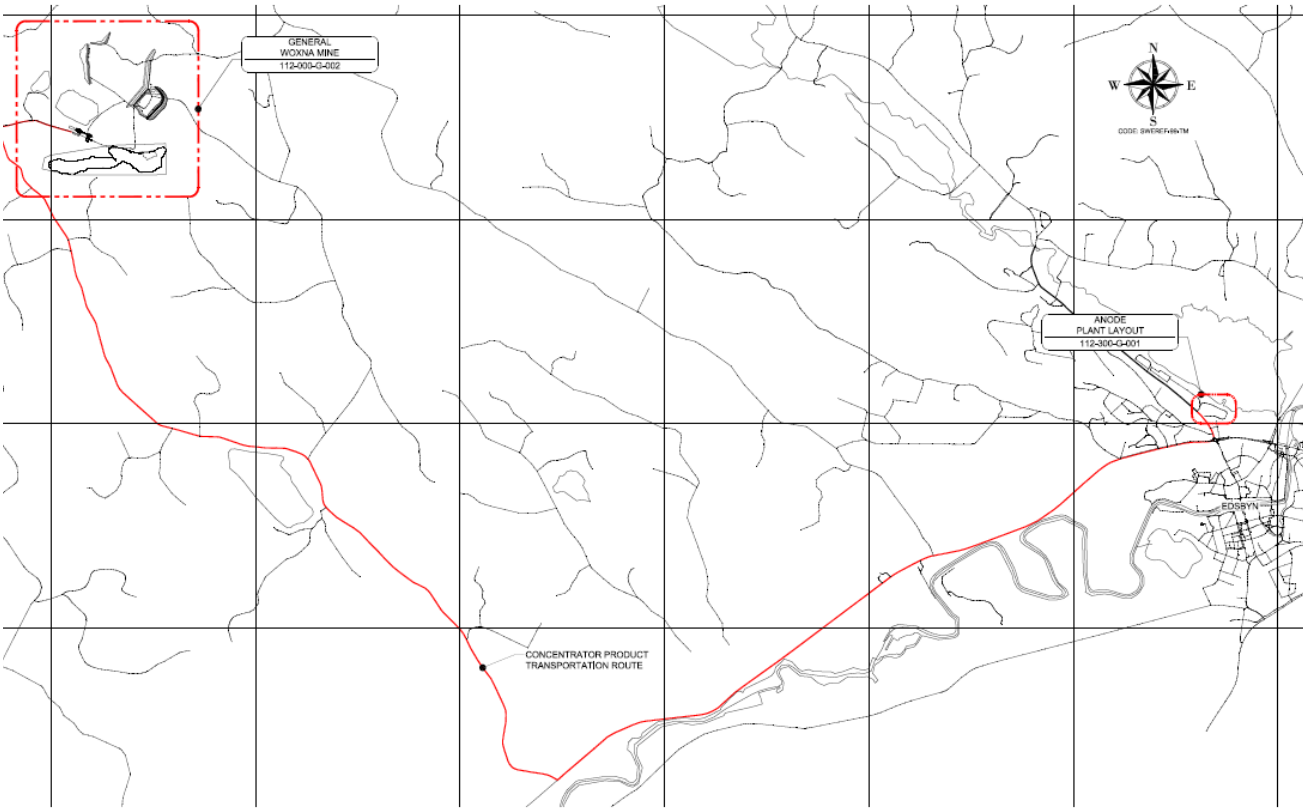
Woxna Mine Production	Units	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Total
Total Rock Mined	Mt	0.45	1.39	1.068	0.862	1.004	1.017	1.005	1.014	1.010	0.984	1.056	0.943	0.874	0.658	0.513	0.430	0.071	0.000	0.000	0.000	14.449
Run of Mine -HG Mined	Mt	n/a	0.161	0.160	0.159	0.161	0.160	0.159	0.160	0.160	0.160	0.160	0.160	0.160	0.160	0.161	0.160	0.043	0.000	0.000	0.000	2.444
Run of Mine -HG Mined	grade	n/a	10.2%	10.2%	9.9%	9.7%	9.6%	9.6%	9.9%	10.4%	10.6%	9.2%	10.1%	11.0%	10.7%	11.1%	10.8%	9.9%	0.0%	0.0%	0.0%	10.2%
Run of Mine - LG Mined	Mt	n/a	0.059	0.031	0.036	0.079	0.073	0.047	0.028	0.037	0.027	0.031	0.036	0.015	0.003	0.012	0.011	0.001	0.000	0.000	0.000	0.525
Run of Mine - LG Mined	grade	n/a	6.0%	5.3%	4.8%	4.6%	4.6%	5.0%	5.2%	5.1%	5.1%	5.1%	5.2%	4.9%	4.2%	4.6%	4.8%	4.9%	0.0%	0.0%	0.0%	5.0%
Overall Mined C_pct	%	n/a	9.1%	9.4%	9.0%	8.0%	8.1%	8.5%	9.2%	9.4%	9.8%	8.6%	9.2%	10.5%	10.6%	10.7%	10.4%	9.8%	0.0%	0.0%	0.0%	9.3%
Total Waste	Mt	0.45	1.720	0.876	0.668	0.764	0.784	0.799	0.825	0.813	0.797	0.866	0.747	0.699	0.495	0.341	0.259	0.027	0.000	0.000	0.000	11.480
Strip Ratio	t:t	n/a	5.4	4.6	3.4	3.2	3.4	3.9	4.4	4.1	4.3	4.5	3.8	4.0	3.0	2.0	1.5	0.6	0.0	0.0	0.0	3.7
Stockpiles	**Units**		**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**	**Year 16**	**Year 17**	**Year 18**	**Year 19**	**Total**
Balance in Stockpile	Mt	n/a	0.059	0.090	0.125	0.204	0.277	0.324	0.352	0.389	0.416	0.447	0.483	0.498	0.501	0.513	0.524	0.525	0.408	0.248	0.088	n/a
Graphite Content	Mt	n/a	0.004	0.005	0.007	0.011	0.014	0.017	0.018	0.020	0.021	0.023	0.025	0.026	0.026	0.026	0.027	0.027	0.021	0.013	0.005	n/a
Stockpile grade at end of year	%	n/a	6.0%	5.8%	5.5%	5.2%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.1%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	n/a
tonnes out during year	Mt	n/a	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.117	0.160	0.160	0.088	0.525
grade of tonnes out	%	n/a	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	5.15%	5.15%	5.15%	5.15%	5.15%
Processed Material	**Units**		**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**	**Year 16**	**Year 17**	**Year 18**	**Year 19**	**Total**
RoM Processed Direct from Mine	Mt	n/a	0.161	0.160	0.159	0.161	0.160	0.159	0.160	0.160	0.160	0.160	0.160	0.160	0.160	0.161	0.160	0.043	0.000	0.000	0.000	2.444
RoM Processed Grade	%	n/a	10.2%	10.2%	9.9%	9.7%	9.6%	9.6%	9.9%	10.4%	10.6%	9.2%	10.1%	11.0%	10.7%	11.1%	10.8%	9.9%	0.0%	0.0%	0.0%	10.2%
From Stockpile	Mt	n/a	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.117	0.160	0.160	0.088	0.525
From Stockpile Grade	%	n/a	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	5.0%	5.0%	5.0%	5.0%	5.0%
Processing Total from Mine & Stockpile	Mt	n/a	0.161	0.160	0.159	0.161	0.160	0.159	0.160	0.160	0.160	0.160	0.160	0.160	0.160	0.161	0.160	0.160	0.160	0.160	0.088	2.969
Processing Total from Mine & Stockpile Grade	%	n/a	10.2%	10.2%	9.9%	9.7%	9.6%	9.6%	9.9%	10.4%	10.6%	9.2%	10.1%	11.0%	10.7%	11.1%	10.8%	6.4%	5.0%	5.0%	5.0%	9.3%
Recovered Carbon	t	n/a	15,387	15,252	14,791	14,699	14,431	14,251	14,866	15,533	15,776	13,851	15,135	16,528	16,074	16,725	16,140	9,511	7,533	7,533	4,136	258,152

1.18 Recovery Methods

The basis for the design of the Woxna Concentrator and associated VAP is the beneficiation of:

- 160,000 tpa of RoM with a grade of approximately 9.18% Cg by flotation to produce-
- an average of approximately 14,730 tpa of graphite concentrate, grading at 92.3% Cg at the Woxna Concentrator on the mine site, which is then-,
- transported to the VAP which will produce an average of approximately 6,604 tpa of CSPG at minimum 99.95% Cg and an average of approximately 7,479 tpa micronized graphite at a grade of 93% Cg.

The VAP facility that has been conceptually chosen for the purpose of this PEA is located off the mine site in the nearby town of Edsbyn as shown in Figure 1-7 along the transport route from the mine site. The decision to locate the VAP plant here was based on better and more cost-effective infrastructure availability in Edsbyn.



Source: Zenito 2021

Figure 1-7: Relative location of the Woxna Concentrator and VAP

The Woxna Concentrator product will undergo several beneficiation stages through the VAP to result in the final CSPG. Sequentially these stages are micronizing for sizing, spheronization to produce 'curled' flakes of graphite; followed by thermal purification and finally coating of the purified, spheronized flakes with additional carbon to produce graphite suitable for use in battery anodes.

A set of preliminary plant design criteria and mass balance was produced to provide data for sizing the specific unit operations and the main equipment. The criteria were produced from historical production information, testwork and equipment suppliers' information. The main production criteria for the Woxna Concentrator and VAP are given in Table 17-1.

Table 1-12: Woxna Concentrator and VAP production criteria

Criteria	Value	Unit
Plant life	19	years
Concentrate plant availability	88	%
VAP plant availability	96	%
Woxna Concentrator flotation plant throughput	160,000	tpa
RoM grade	9.2	% Cg
Woxna Concentrator concentrate production	15,692	tpa
Woxna Concentrator recovery	93.8	%
Woxna Concentrator product grade	92.3	%
Spheronized graphite production	6,629	tpa
Spheronization yield	45	%
Spheronized graphite particle size d_{50}	15	μm
Purified spheronized graphite (Thermal production)	6,629	tpa
Thermal treatment temperature	2,600	°Cg
Purified spheronized graphite grade	Min 99.95	% Cg
Coated product - CSPG production	6,604	tpa
Micronized (jet milled) fines production	7,479	tpa
Micronized graphite particle size d_{50}	4	μm
Micronized graphite grade	92.3	% Cg

The envisaged process design includes the simplified block flow diagram shown in Figure 1-8:

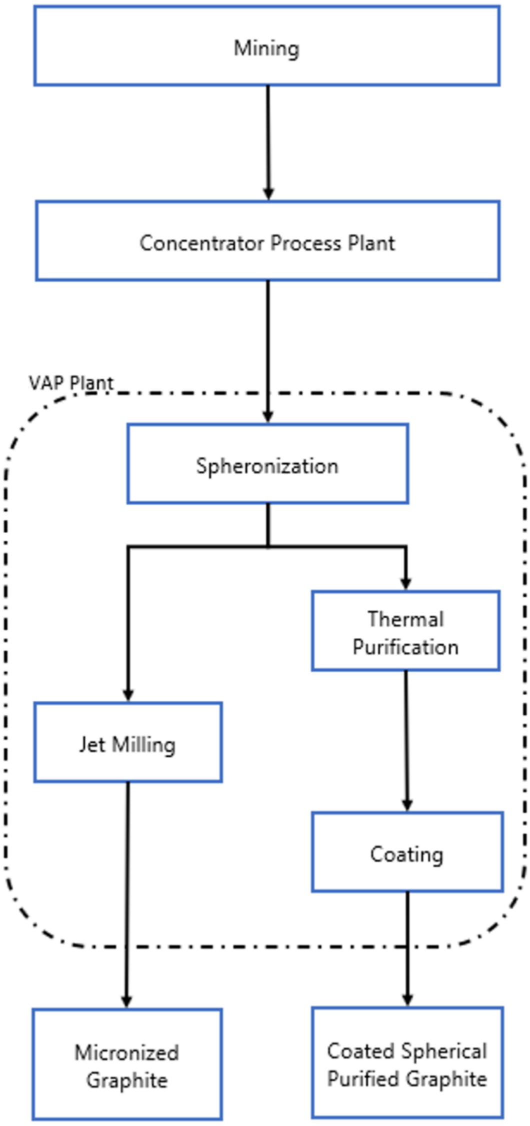


Figure 1-8: Simplified Process Block Diagram

The proposed changes to the current Woxna Concentrator to accommodate the required throughput and the additional new aspects required for the VAP are summarised as follows:-

- The basis for the comminution circuit is the existing rod mill. To maximise its throughput the comminution circuit will include a new crushing plant, crushed material stockpile and rod mill classifier.
- The flotation circuit is based on the latest flotation tests which were carried out by BGRIMM.
- The dewatering and bagging circuits remain are as before.
- Additions are dewatering cyclones in the flotation circuit to increase the slurry density and a clarifier for cleaning the cyclone overflow.
- The existing classification screens for the dry graphite are no longer required.
- The VAP is designed to include milling and spheronization of the graphite flotation concentrate followed by thermal treatment. Fines from the spheronizing are milled again to produce ultra-fine graphite. This will be all new equipment.

The basis for the VAP upgrade process is micronizing, spheronizing the graphite flotation concentrate followed by thermal treatment of the spheronized product. Fines from the spheronizing are milled again to produce fine graphite. This upgrade facility will be all new equipment.

1.19 Project Infrastructure

1.19.1 Mine and concentrator

The mine site has a partially depleted existing open pit, TSF, waste rock dump areas, mine site roads, clarification ponds and a processing facility as shown in Figure 1-4.

The mine shares the access, of approximately 9.5 km, from national Route 301 to the site gate. Site roads for access and maintenance experience low levels of traffic and are of basic unsealed construction. Maintenance is performed on an as needs basis using mine and plant equipment and contributes a negligible cost to operations. Haul roads will be constructed and maintained by the mining contractor and included in their operating cost.

Current diesel storage on site is a tanked facility, positioned adjacent to the main building within a bunded area. The primary consumer of diesel, the concentrate dryer, will be converted to electric heating due to the operating and environmental cost savings provided. The mining contactor will provide and operate its own diesel storage facility.

There are currently two 1,000 kVA transformers installed in a Transformer / MV switchgear / LV switchgear building located close to the plant. A new MV 1,900 kVA substation is to be installed and supplied in a kiosk type enclosure, including MV switchgear with spare feeder to allow for future expansion. This would remove the need for the MV switchgear in the existing location thus separating the LV and MV switchgear. Power will be distributed throughout the process plant on existing and new circuits from the plant main LV switchgear assembly, which will remain in the existing substation building. Reticulation throughout the plant will be via radial circuits at 400 V. No allowance has been made for emergency power.

The mobile network coverage at the Project area is good, and an internet connection is currently established.

A warehouse/stores facility of approximately 950 m² is located adjacent to the Woxna Concentrator. Most maintenance will be performed either in situ or in a contractors' facility in the nearby town, and therefore, a large, dedicated maintenance area with cranage is not required on site. The mining contractor will provide and operate their own maintenance facilities.

There is an office that is currently in use by the mine staff of approximately 200 m². Several semi- portable office sections have been installed expanding the office space to approximately 300 m². There is a small accommodation block of approximately 120 m² adjacent to the administration building.

Potable water is currently brought in by water tanker to a small on-site storage facility. This will continue during operations due to the low cost and adequate service.

A prefabricated laboratory building has been installed adjacent to the process plant building so as to be free from contamination of dust and vibration from the plant. The laboratory is capable of processing approximately 10 mining samples, 16 plant samples, and 50 product samples per day for size analysis, carbon content, sulfur content and moisture content.

1.19.2 VAP facility

Current power supply to the VAP Facility selected for the purpose of this PEA is 2 MW and includes a transformer installation and switchgear. An additional 7.6 MW is required for the VAP facility, a total of 9.6 MW. Additional transformer rooms are available at the facility to upgrade the power supply. To meet the increased power demand, an additional 3.5 km cable will be installed from the main substation to the VAP facility. All areas in the VAP facility are equipped with lights and 220 V power supply. All major areas are equipped with 400 V 3-phase power points. No allowance has been made for emergency power.

Raw water, for the furnace cooling phase heat exchangers, comes from lake Ullungen which is next to the VAP. Potable water will be sourced from the local Edsbyn municipal water system.

The existing VAP facility building is in good shape and has a usable area of approximately 11,500 m^2. There is ample workshop space, as well as storage facilities. The building is equipped with several offices, spaced throughout the premises for operational control. Change house and ablution facilities are in place and could accommodate a total of approximately 90 employees.

1.20 Market studies and contracts

1.20.1 Demand

Flake graphite demand by end-use/market in 2020. The Li-ion battery market currently accounts for around a quarter (202,617 tonnes) of the natural flake demand.

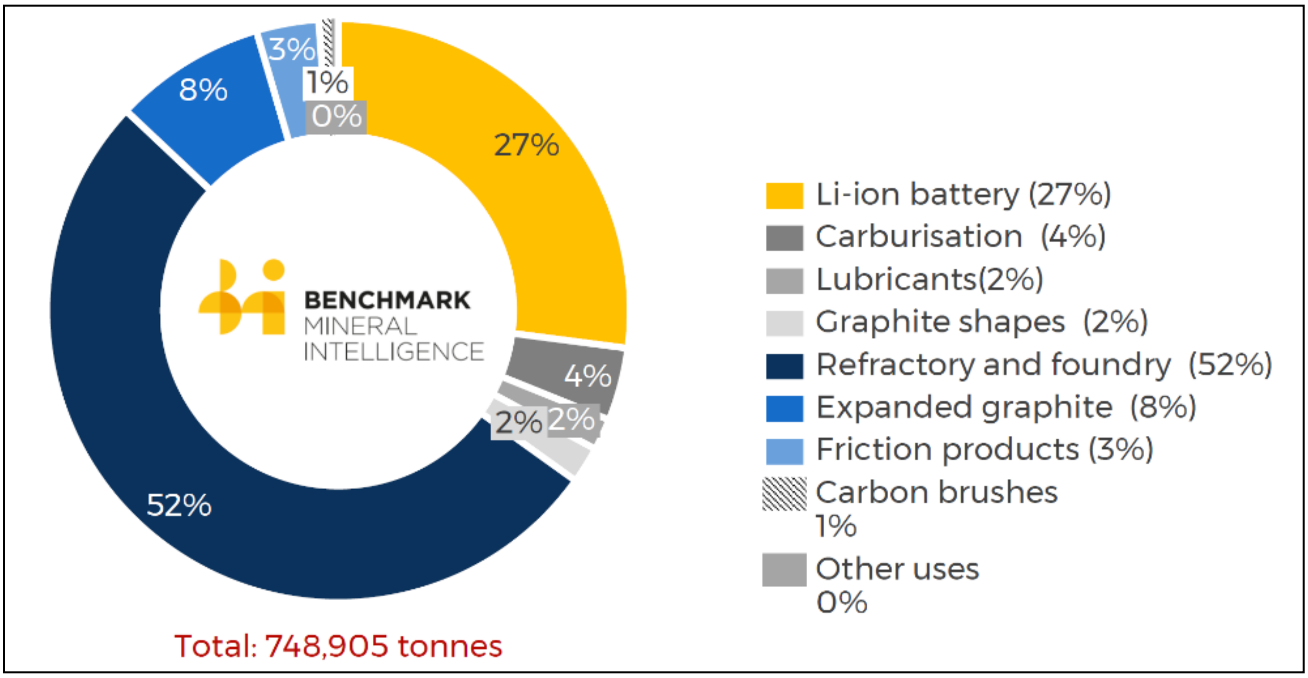


Figure 1-9: Flake graphite demand by end-use/market 2020, (Benchmark Mineral Intelligence, 2021)

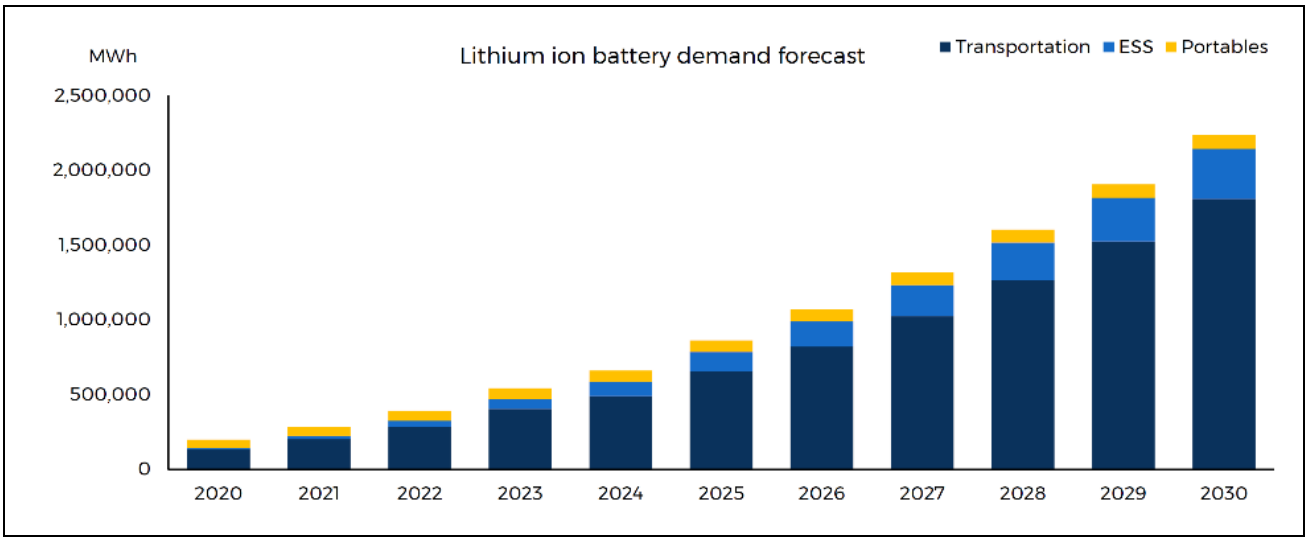


Figure 1-10: Lithium-ion battery demand forecast (Mwh), (Benchmark Mineral Intelligence, 2021)

The natural flake graphite demand for anode applications is currently around 200,000 tonnes, the Benchmark Mineral Intelligence forecast is expected to reach 1.1 million tonnes in the next five years, and 2.8 million tonnes in the next ten years.

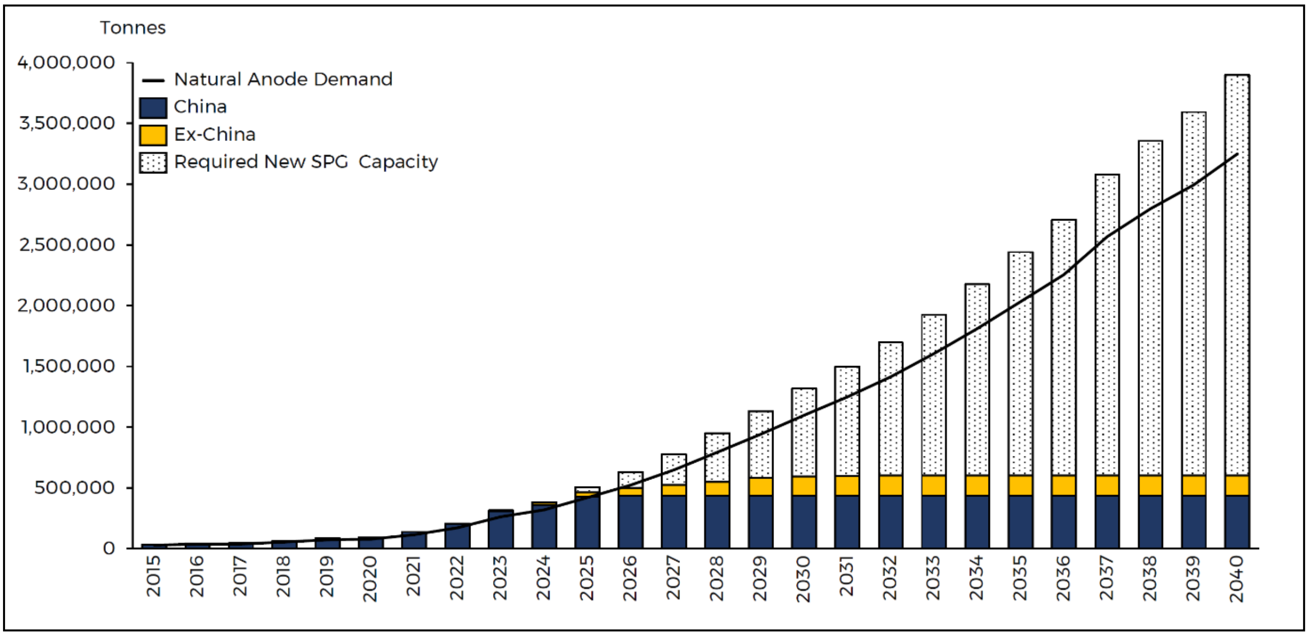


Figure 1-11: Spherical Graphite Capacity & Natural Anode Demand Forecast (tonnes) – 2015-2040, (Benchmark Mineral Intelligence, 2021)

The forecasted demand in Europe, is planned for over 700 GWh of battery manufacturing capacity, which would need approximately 700 ktpa of anode material.

1.20.2 Supply

China dominates the natural graphite raw material supply, with Spherical Purified Graphite supply currently 100% China dominated supported by an abundance of flake feedstocks for the production of anode precursors at current demand levels. Production costs for graphite concentrate are low, which sets the majority of their production towards the low end of current cost curves for flake graphite. China competes with their ready-built infrastructure of spheronizing facilities which operate in regions with subsidised energy costs, cheap use of harmful acid purification, meaning their opex for value-adding processes is very competitive.

According to Benchmark Mineral Intelligence, the capabilities for European sourced graphite products currently accounts for only 3% of extraction/mining, 0% chemical processing, 0% anode production capacity.

1.20.3 Coated Spherical Purified Graphite (CSPG) pricing

CSPG material is rarely traded in the open market. Contractual agreements are long term for materials based on highly integrated models between seller and buyer. As such there is a wide range in prices for this material, application dependent. The current and forecasted pricing for CSPG was provided in a specialist consultant report by Benchmark Mineral Intelligence, with the average EV Tier 1 pricing ranging from 9,500 to 13,750 USD/t. Refer to Figure 1-12 below. A nominated price of 10,000 USD/t has been selected for the Woxna CSPG product.

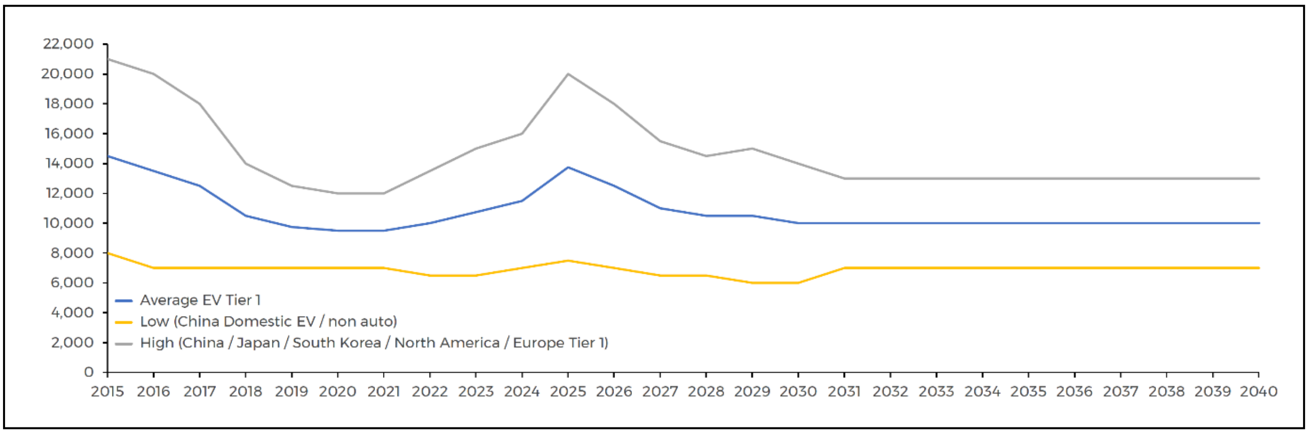


Figure 1-12: CSPG price forecast (USD), (Benchmark Mineral Intelligence, 2021)

1.20.4 Micronized graphite pricing

A European graphite specialist consultancy conducted jet mill micronizing test work with Woxna feed material in 2020 [3]. It was shown, that with both the flotation plant concentrate and spheronized fines materials, a properly jet milled micronized graphite product could be produced.

The current pricing for micronized flake graphite was provided by the consultant and is displayed in Table 19-3 below.

Table 1-13: Current pricing for micronized graphite <96% C (2021)

Particle Size		d50 ≥ 12 µm	d50 < 12 µm	Sum
Mill type		Impact mill	Jet mill	
Carbon Content		C < 96%	C < 96%	
min. price	USD/t	960	1,690	
max. price	USD/t	2,710	4,210	
average price	USD/t	1,440	2,530	
quantity	kt	9	10	**19**
total value	**M USD**	**13.0**	**25.3**	**38.3**

The total market value of micronized natural graphite with carbon content below 96% is above USD 38 million. The growth rates for these products have been strong over the past few decades.

All the Woxna micronized graphite is produced from the fines created as a by-product during spheronization and would be a 'jet milled d50 < 12 µm' product.

Testwork indicates that the carbon content of the woxna micronized graphite is 92.3% C. The annual woxna production of a 'jet milled d50 <12 µm' product is 6.6 kt. Given that 10 kt is currently consumed annually in Europe, the Woxna production may impact pricing. When considering these factors and the specialist consultant report, a price of 1,200 USD/t has been specified for the micronized graphite Woxna product. This value is considered to be conservative refering to the pricing in Table 1-13 for a 'jet milled d50 < 12 µm' product.

1.21 Environmental studies, permitting and social or community impact

1.21.1 Environmental Studies

A review of previous environmental studies, investigations and inventories has been made by an independent environmental consultancy Golder (2021). Some of the previous information needs to be updated, however the results of many of the existing studies can be used. In 2016 an environmental impact assessment (EIA) report was prepared by Poyry AB as part of the application for an extension of the exploitation concession (Kringelgruven K nr 1).

Woxna Graphite has developed an environmental monitoring programme related to the activities at the Kringel property and the latest information available is dated April 2018. Furthermore, Woxna Graphite plans to further process the graphite concentrate at a new VAP facility in the region. The VAP selected for the purpose of this PEA is considered “not likely” to present any significant environmental local impact, however an environmental permit will still be needed for its development. The permitting of the VAP can form a part of the same recommended environmental permit application for the expanded mining operation at Kringel if advantageous.

1.21.2 Environmental permitting

Woxna Graphite holds a number of environmental permits issued under previous and now superseded environmental law. Whilst these permits remain valid, Golder does not consider that these permits afford Woxna Graphite or the state the same protections as current laws/permits do. Consequently, Golder considers it unlikely that the permits can be modified and recommends that new permitting within current law be obtained and considers that such permitting would be required prior to starting up the full-scale mining activities. In any case, a new environmental permit would be required to allow the increase production rate considered in the PEA.

The permits currently held are as follows:-

- a permit according to the Environmental Protection Act granted by the Licensing Board for Environmental Protection (sv Koncessionsnämnden för miljöskydd) on 17 September 1992. Despite this permit being with an old law, it is still valid. However, it is limited to mining and processing of only 100,000 tonne per annum.
- a permit according to the Water Act that was granted by the Water Court (sv Vattendomstolen) on 15 October 1992. The Water Act is also an old law, but the permit remains valid.

Along with the granted water permit, the supervising authority (County Administration Board) has issued a number of decisions regarding operations, refer to Section 20.2.1 for further details in this respect.

1.21.3 Mining Concessions

Woxna Graphite holds a mining concession (exploitation concession) for the deposit at Kringel (K no 1) that is valid until 3 November 2041. Besides the Kringel concession, Woxna Graphite holds exploitation concessions at other places in the municipality, Månsberg, Mattsmyra and Gropabo which will expire in late 2024 to early 2025 if not extended.

1.21.4 Tailings Storage Facility

The historical Woxna Mine utilised a tailings storage facility (TSF) (alternatively tailings management facility TMF) as shown in Figure 1-4. In the PEA the planned mining and processing produce two types of tailings streams, namely the non-acid generating tailings (NAG) comprising 90% of the arisings and potential-acid generating tailings (PAG) comprising the remaining 10% of tailings volume. Two separate TSFs are considered to accommodate these proposed LoP tailings:

- the current TSF footprint will be used for the storage of the non-acid generating tailings (TMF NAG), extended to the north (see Figure 1-13) ; and
- a smaller TSF for the potentially acid generating waste (TMF PAG), built in connection to the eastern part of the TMF NAG.

The envisaged tailings production at the new Woxna Mine is summarised in Table 1-14 and the final proposed layout is shown in Figure 1-13.

Table 1-14: Estimated tailings production (LEM)

Description		Unit	Value
Life of Project (LoP)		years	19
Annual Tailings production rate		t/year	160 000
Assumed in situ density		t/m^3	1.4
Annual Tailings Volume	Non-acid generating (NAG) 90% by volume	m^3 /year	102,857
	Potential acid generating (PAG) 10% by volume	m^3 /year	11,429
	Total	m^3 /year	114,286
Total Volume over LoP	NAG (90%)	m^3	1,954,286
	PAG (10%)	m^3	217,143
	Total	m^3	2,171,429

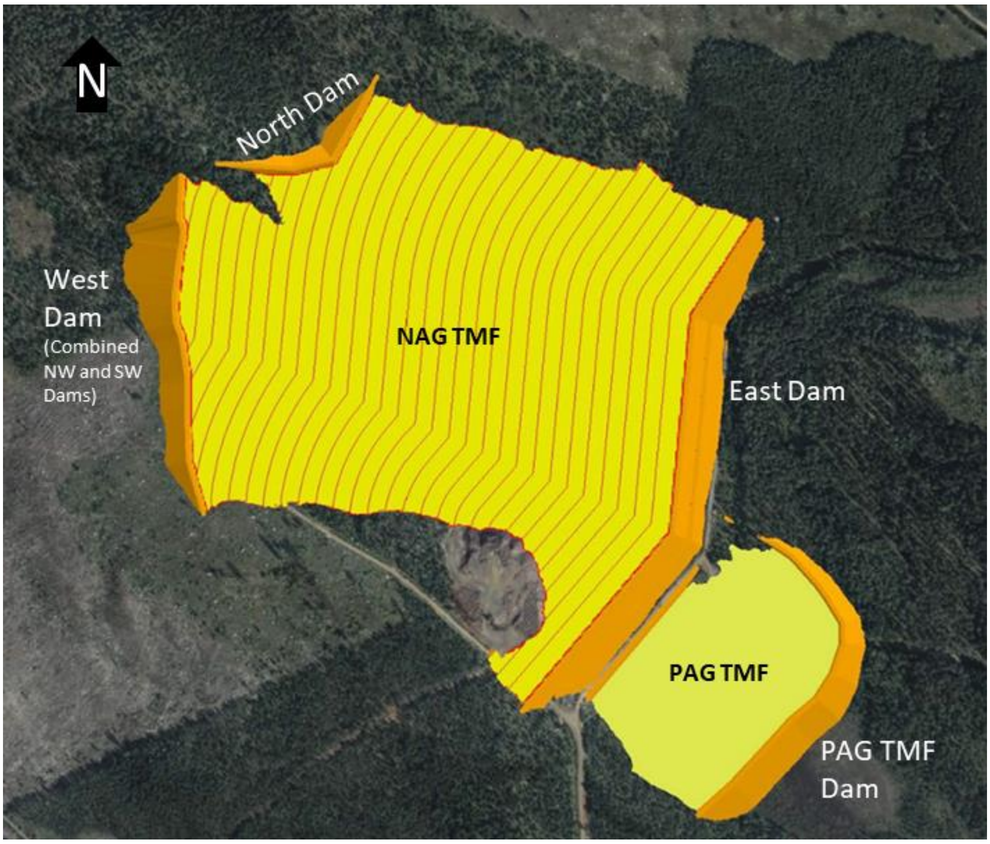


Figure 1-13: TSF Conceptual Raise Design.

The NAG TSF Raise will consist of the following over the LoP:

- West Dam – Downstream raising of the NW and SW Dam to elevation +281 m.
- East Dam – Upstream raising of the existing East dam to elevation +283 m and lengthening by 380 m to the north.
- North Dam – Construction of a new rockfill dam (with upstream moraine wedge) to an elevation of +282 m in the north valley.

The new PAG TMF will consist of the following over the LoP:

- a lined facility to prevent seepage from the PAG tailings into the environment.
- The PAG TMF will be formed by the construction of an eastern rockfill embankment dam.
- The PAG tailings will be kept 2 m under water consistently to prevent oxygen ingress and potential oxidation and acid generation from the PGAG tailings.
- The dam will be constructed in two stages based on the deposition and construction schedule.

The schedule of construction is based on the rate of filling of the facilities. It has been assumed that each raising of the embankments can be completed over a period of one year prior to that particular raise being brought into operation.

The construction programme was therefore divided into several staged raises taking place in years 0, 1, 6, 9 and 12.

1.21.5 Water Management

In the water balance of the mine site, the processing plant, the open pit, the TSF and the clarification pond were included as the active parts. The water management is presented in Figure 1-14.

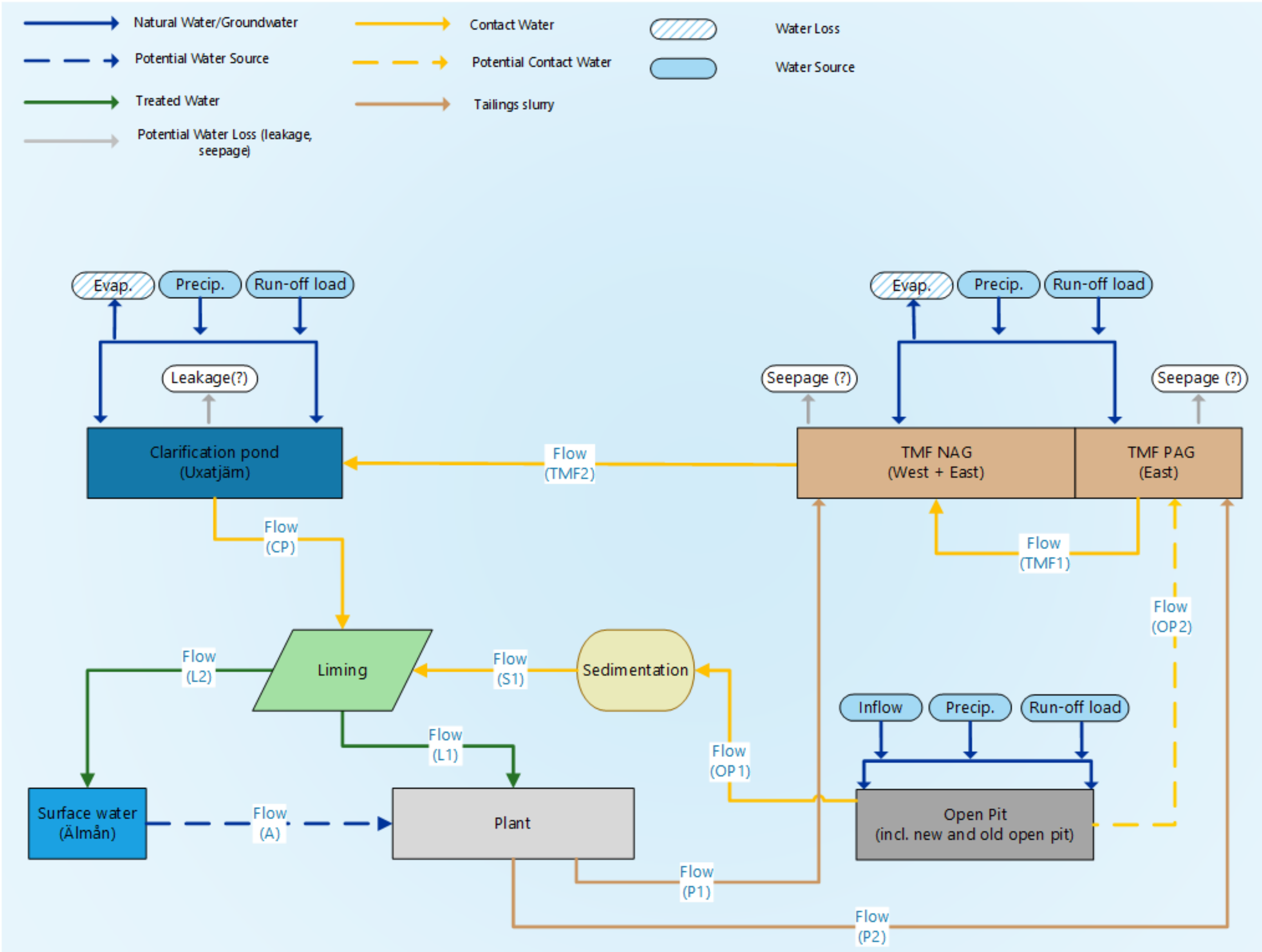


Figure 1-14: Conceptual water balance model with mining operation (Golder, 2021).

The use of the small Lake Uxatjärn, located close to the TSF, as a clarification pond for the mine will continue. There is an existing small embankment dam for the clarification pond that will be rehabilitated and raised to ensure it is brought to required technical standard as well as to provide greater operational flexibility, refer Figure 1-15.

Water from the clarification pond will be reused into the processing plant. If excess water is required to be discharged, it will undergo treatment before being released to the Älmån river.

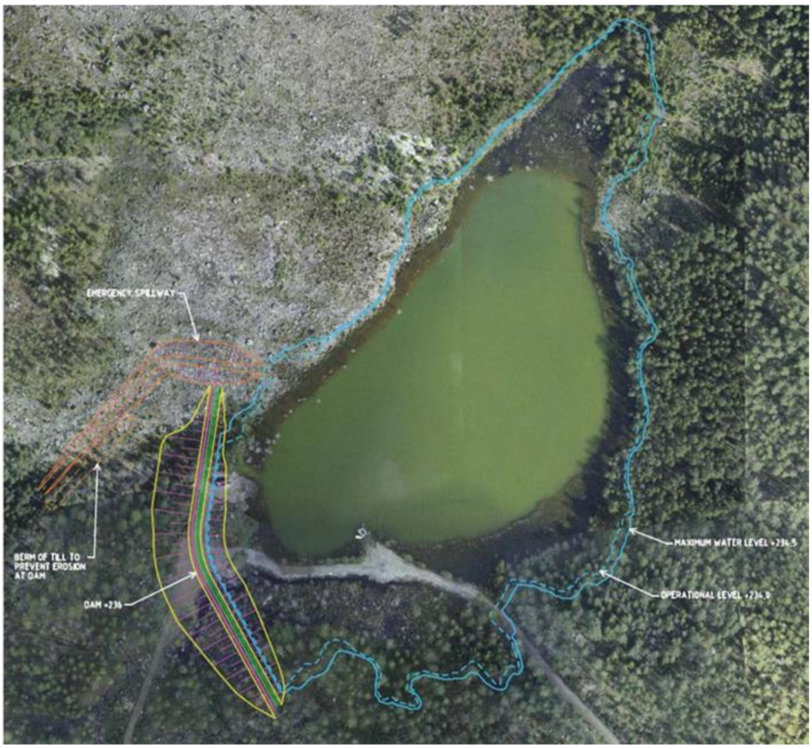


Figure 1-15: Clarification Pond Proposed Expansion.

1.21.6 Closure

Closure of operations at Kringel are based on current knowledge of the planned operation and selected from options to be the most suitable based on existing knowledge about Swedish closure strategies and practice. The overall closure objective is to provide a "walk away" solution after a monitoring period of up to 30-years according to current regulatory practise.

The closure objectives are based on the expected future land use and are as follows:

- The area around the former mine site is with time to be returned to natural/forest land.
- TMF is to, with time, to become a "natural" meadow/heath and/or forest.
- The open pit and clarification pond will with time become natural water bodies i.e. lakes.
- Tailings and waste-rock is not to produce acid mine drainage, thereby not affecting the surroundings including surface waters post closure.
- Waste rock and tailings as well as the open pit and clarification pond should pose no human or environmental risks.

Closure actions include that most surface plant that will be decommissioned and removed, capping of the PAG TMF and waste rock dump, followed by a period of monitoring. The costs of these items are estimated and included into the project.

Refer to Section 20 for further detail.

1.21.7 Life cycle assessment

A cradle-to-gate Life Cycle Assessment has been commissioned and based on the technical data acquired as part of the PEA for the production of spheronized coated graphite and micronized graphite products from natural flake graphite.

1.22 Capital and operating costs

The project cost estimates are compiled from supporting engineering documents and cost information derived from the following sources and are considered effect as of 9th June 2021:

- Quotations from equipment suppliers and local service providers.
- Historical cost information sourced from in-house and commercial databases.
- Woxna Graphite derived data from existing operations.

This Project estimate is AACE Class 5 targeting overall accuracy ±30%, with a base date of the 9th June 2021, and is reported in USD.

The initial and life of project (LoP) capital expenditure is summarised in Table 1-15.

Table 1-15: Capital Expenditure (CAPEX) Estimate – Initial and Life of Project (LoP)

Cost Centre	Initial CAPEX [USD]	LoP CAPEX [USD]
Mining	2,029,500	2,029,500
Buildings	1,689,478	1,689,478
Earthworks	2,337,400	7,007,000
Civil	-	-
Concrete	334,952	334,952
Steelwork	328,417	328,417
Mechanical	69,354,246	69,354,246
Mechanical Installation	7,880,600	7,880,600
Mobile Equipment	133,390	133,390
Electrical	3,773,693	3,773,693
Control and Instrumentation	1,022,662	1,022,662
Piping	2,110,985	2,110,985
Platework	581,168	581,168
Insurance Spares	-	-
DIRECT TOTAL (D)	**89,546,992**	**96,246,092**
Engineering, Procurement & Construction Management (EPCM)	5,000,000	5,000,000
Construction Facilities	696,188	696,188
Commissioning	882,143	882,143
Field Indirect	-	-
Owners Cost	654,890	7,832,924
Contingency	17,441,918	17,441,918
INDIRECT TOTAL (I)	**24,675,140**	**31,853,173**
FIXED CAPITAL TOTAL (F = D + I)	**114,222,131**	**128,099,265**
Cash Reserves	-	-
Inventory	4,858,455	4,457,145
WORKING CAPITAL TOTAL (W)	**4,858,455**	**4,858,455**
TOTAL CAPITAL INVESTMENT (F + W)	**121,110,086**	**132,957,720**

The following sections summarise and describe the development of the various operating cost components. The overall operating expenditure (OPEX) is provided in Table 1-16.

Table 1-16: Overall Operating Expense (OPEX) Estimate

Description	USD/a	USD/t ROM	USD/t Graphite Product
General & Administration	325,725	2.04	21.50
Mining	3,630,426	22.69	239.63
Processing	20,816,104	130.10	1,374.00
Waste	312,534	1.95	20.63
Grand Total	**25,084,788**	**156.78**	**1,655.76**

The overall OPEX figures are presented assuming a nameplate production year. This means assuming a plant feed of 160 ktpa, an average strip ratio of 3.7 for the first 15-years of mining (thereby avoiding startup and shutdown irregularities), and that annual payments through the LoP to accumulate the closure fund are incorporated.

1.23 Economic analysis

The economic modelling input criteria applied in the financial model are summarised in. Table 1-17:-

Table 1-17: Base case financial inputs

Description	Value	Unit	Reference
Corporation tax rate	20.6	%	Client
Discount rate	8	%	Client
Finance rate	0	%	A
Royalty rate	0.2	%	Client
Base currency code	USD	-	Client
Price Coated Spherical Purified Graphite	10,000	USD/t	Benchmark Mineral Intelligence
Price Micronized Graphite	1,200	USD/t	Client

The results of economic modelling for the production schedule under the capital and operating cost, and the economic parameters as described in the relevant sections are presented in Table 1-18 below.

The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Table 1-18: Economic summary

Parameter	Value	Unit
Life of Project	19	years production
Initial Capital Investment	121,110,086	USD
Life of Mine Capital Investment	132,957,720	USD
Price Coated Spherical Purified Graphite	10,000	USD/t Product
Price Micronized Graphite	1,200	USD/t Product
Revenue	480	USD/t ROM
	5,326	USD/t Product
Cost of goods sold	165	USD/t ROM

Parameter	Value	Unit
	1,828	USD/t Product
Gross Margin	316	USD/t ROM
	3,499	USD/t Product
	65.7%	
Pre-Tax		
Pre-Tax IRR	42.9%	
NPV @8%DR	317,031,929	USD
Total Payback Period	4.24	Years (from initial investment)
Payback after first production	2.24	Years (from first production)
Post-Tax		
Post-Tax IRR	37.4%	
NPV @8%DR	248,167,633	USD
Total Payback Period	4.51	Years (from initial investment)
Payback after first production	2.51	Years (from first production)

1.24 Adjacent properties

There are no known operators of any relevant activities on directly adjacent properties or locally adjacent properties.

1.25 Other relevant data and information

MPlan 2021 prepared constraining pit shells for ReedLeyton to be used for Mineral Resource estimation of the Kringel deposit using optimised pit shells generated using Datamine™ NPVS software. The key assumptions used in the generation of the resource constraining pit shells for the Kringel deposit were:

- overall slope angle for resource pit shell: 55 degrees;
- mill cut-off grade = 4.00%;
- break even cut-off grade = 4.21%;
- process cost: USD 84.18/t mill feed
- dilution 2.5%
- mining recovery 97.5%
- process recovery 93.7%

The cut-off grades assumed:

- graphite price of: USD 2,320;
- recovered value of USD 2,103/t after applying costs, taxes, mining, and process recovery factors;

Note that the Kringel mine depth is limited to 70-metres (applied due to permitting), and by the mining concession limits at its geographic boundary. The proposed mine plan fully exploits the available resource within these limits and is therefore relatively insensitive to other factors that might otherwise expand the pit, such as an increase in product pricing, meaning the above work using lower product pricing remains applicable.

1.26 Interpretation and conclusions

The PEA investigated at 30% to 50% levels of accuracy the technical and economic aspects of the Project so as to ascertain the economic viability of re-opening the historical Mine, expanding the historical Woxna Concentrator, building a new VAP to produce CSPG and to assess the costs involved with upgrading the infrastructure to support such an operation. The specialists involved with the technical and economic assessment are all Qualified Persons according to NI 43-101 criteria, independent of Woxna Graphite and capable of providing opinions in relation to their expertise.

Historical Mineral Resource estimates have been developed by various parties and at different points in the development of the four Woxna graphite deposits. These historical and previous Flinders 2015 Mineral Resource estimates were developed without the constraint of an applied mine plan and open-pit shell. In the light of more rigorous compliance requirements, the Mineral Resources were reported by ReedLeyton within the constraints of the 2021 PEA mine plan as a means of demonstrating "reasonable prospects for economic extraction" as required by numerous international reporting codes. No new exploration data was included in the re-estimation process but the database provided by the exploration campaigns was verified and is considered reliable for the use in the Mineral Resource estimation.

All of the Mineral Resource estimates were prepared in accordance with the NI43-101 Standards of disclosure and the classification of levels of confidence are considered appropriate on the basis of drillhole spacing, sample interval, geological interpretation and all currently available assay data.

In summary, the Mineral Resource estimation for the Woxna Project based solely on the Kringel property deposit is:-

- Measured plus Indicated Resources of 2.61 Mt at an average in situ grade of 9.13% Cg which have been classified as Type A at a cut-off grade of >7% Cg and Type B at a cut-off grade of ,7% Cg; and
- Inferred Mineral Resources of 0.39 Mt at a grade of 8.7%Cg exist but are not included in the mine scheduling.

The Gropabo and Mattsmyra deposits have a combined Inferred Mineral Resource of 2.12 Mt at a grade of 8.6%Cg and although not included in the PEA economic assessment, these resources represent upside potential for the Project that could be exploited should additional resources be desired. The Mansberg deposit represents a future exploration target.

Although the mineralisation appears to be open to at least a depth of 150 m, the lateral and depth extent of the existing and future open-pit mine at Kringel is and will be, constrained by adjacent nature reserve. The existing pit will require de-watering and the methodology will be a simple, standard, low risk conventional drill and blast combined with trucks and shovel mining method.

The mine plan begins in the east pit and is designed to provide approximately 160,000 t of high-grade 'Type A' process plant feed every year until it runs out in year 16, after which the lower grade 'Type B' material that has been stockpiled until then is now fed to the plant. The production plan for the process plant continues on until year 19 when the 'Type B' stockpile is exhausted.

The mining operations will be conducted by a specialist mining contractor, to maximise the operational efficiency, provide flexibility, and lower investment risk to the Project.

The Project aims to produce high purity graphite materials suitable for the lithium-ion battery industry. To achieve this, the existing Woxna Concentrator facilities will be upgraded, and a VAP facility will be constructed. The concentrator upgrade includes a new crushing circuit, and additional flotation and milling. The VAP facility includes dry product sizing, spheronizing, high temperature thermal treatment for purification, and coating plant.

The concentrator is designed to process 160,000 tpa of mineralised material with a grade of approximately 9.2% C to produce a flotation concentrate 14,730 tpa of graphite at a grade of 93% C. The flotation concentrate is fed to the VAP which will produce:-

- an average of approximately 6,604 tpa spheronized, thermally treated and coated 99.95% C battery grade graphite; and
- an average of approximately 7,479 tpa 93% C jet milled graphite.

The infrastructure exists to support this operation with sufficient power, water and surface land use rights to be adequate for the Project current and future requirements. The Project utilises low-cost hydroelectric power from the national grid and with a clean energy supply, and no hydrometallurgy in the process, the products are responsibly produced. The existing TSF will be expanded and a new separate TSF are planned for the storage of both; the 1.954 million cubic meters (Mm^3) of non-acid generating tailings in a northerly extension; and the storage of 217 Mm^3 potentially acid generating tailings in an easterly extension. These tailings facilities holding a total of 2.171 Mm^3 of material will be developed in five stages, and ultimately will be sufficient to accommodate the planned LoP tailings.

The legal tenure regarding Woxna Graphite's right to explore and exploit the graphite properties is valid and current. Some environmental permitting will be required to adequately cover the expansion of the Woxna Mine, TSF and plant as well as the new proposed VAP in Edysberg. The environmental studies already conducted for the previous operations have contributed to the knowledge base and it may be possible to simply update some these studies. No environmental liabilities for the previous mining operation are attributable to Woxna Graphite.

The Project has an initial capital investment of USD 121.1 million and a LoP capital expenditure of USD 130.0 million. Operating costs are estimated at USD 25.1 million per annum.

With Coated Spherical Purified Graphite price of USD 10,000 per tonne, and USD 1,200 per tonne for micronized graphite, and applying discount rate of 8% the Project pre-tax NPV is USD 317.0 million with IRR 42.9%, and post-tax NPV is USD 248.2 million with IRR 37.4%. The post-tax payback period is 4.51 years from initial investment, or 2.51 years from initial production.

The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The Project economics show that the VAP facility is responsible for substantial economic value add, and this fact may be leveraged via the sourcing additional suitable feed materials. Future opportunities exist to extend the Project duration, and/or increase the project throughput via development of the other held deposits in the area.

The Qualified Persons consider that the Project at this level of study is a potentially economic proposition that is well positioned geographically to benefit from the current growth in graphite demand in the EU and elsewhere. The methodologies and processes to be applied are all proven historically to be sound and the existence of the previous mining operation bodes well for the application of the necessary environmental permitting to expand the operations. There are no risks in terms of legal tenure, infrastructure inadequacies or material risks from an environmental perspective that cannot be mitigated. In addition, there is potential to more than double the available input material from self-owned graphite deposits.

In the light of these conclusions, progression of the Project from this PEA level of accuracy to a PSF is justified.

1.27 Recommendations

Based on the favourable results of the PEA, it is recommended that a PFS be completed to provide an in depth and more accurate analysis of various options to provide an established and considered basis for the design of a DFS. It may also be appropriate to develop a study solely focussed on the VAP facility, since it this facility that requires most of the additional definition work, limiting the scope in this way would increase the efficiency of this work, and furthermore it could make a lot of sense to consider the VAP as a stand-alone project.

Further definition of the project is required to complete a PFS and therefore it is recommended that various further studies and testwork be undertaken to prepare for the PFS execution phase.

2 INTRODUCTION

2.1 General

The 'Woxna Graphite Project' comprised an advanced exploration and development project consisting of historically defined graphite deposits held under separate exploitation concessions; an historic, partially depleted open-pit graphite mine (the Woxna Mine); its associated graphite processing facility (the Woxna Concentrator) and related infrastructure, owned by Woxna Graphite AB (Woxna Graphite). The Swedish term for "limited company" or "corporation" when used in company names, is Aktiebolag, abbreviated AB and is roughly equivalent to the abbreviations limited (Ltd) and public limited company (PLC). Woxna Graphite is a 100% owned Swedish subsidiary of Leading Edge Material Corp. (LEM).

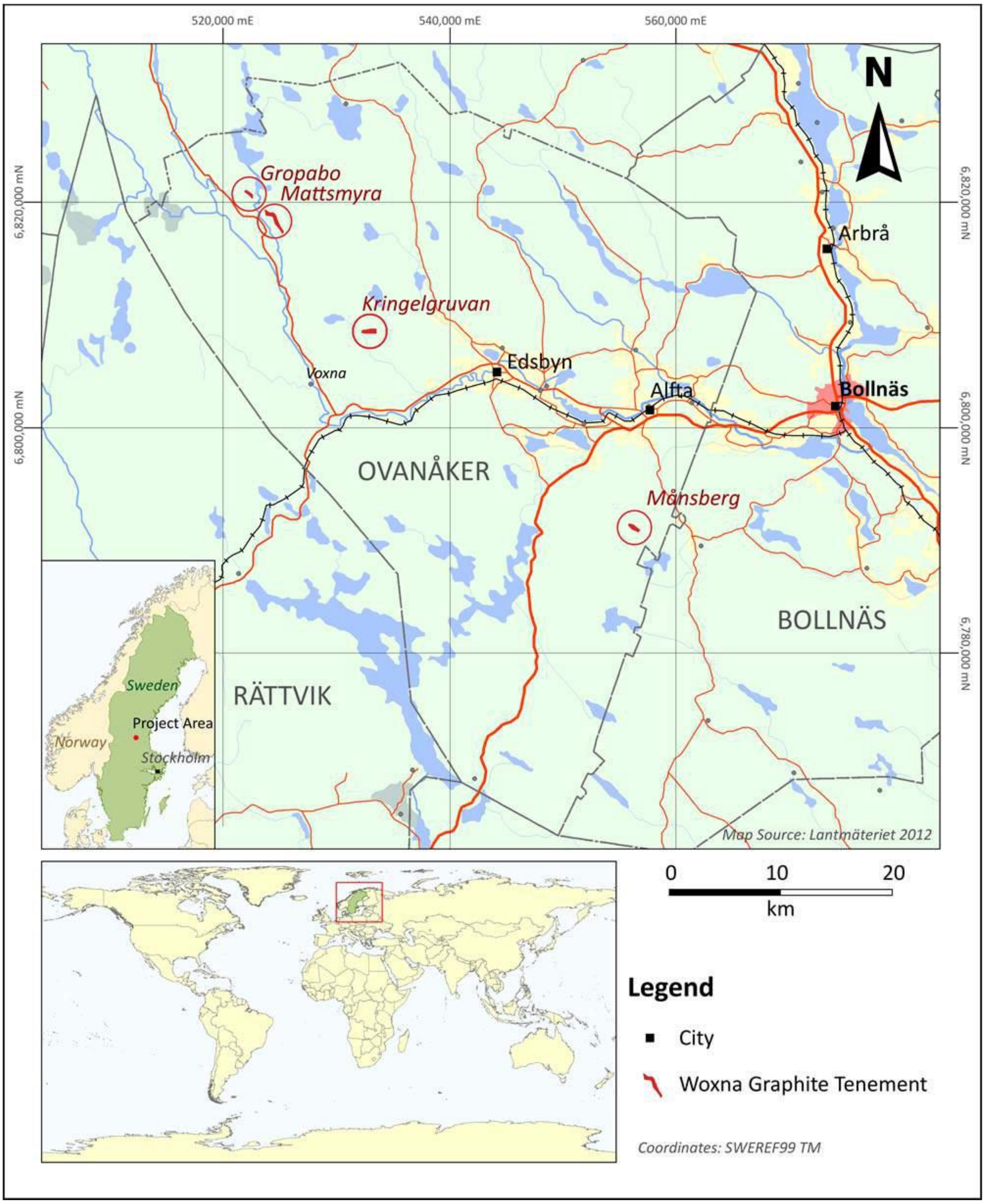
The exploitation concessions and mine/processing infrastructure are located near the town of Edsbyn in the Ovanåker Municipality, Gävleborg County, in the Kingdom of Sweden (Figure 2-1). Historically, the 'Woxna Graphite Project' comprised four exploitation concessions (146.71 ha) over the Kringelgruvan (Kringel), Gropabo, Mattsmyra and Mansberg deposits Kringel has a total Measured and Indicated Mineral Resource (at a cut-off grade of graphitic carbon (Cg) of 4% Cg) of 2.61 million tonnes (Mt) at a grade of 9.13% Cg, and only Kringel is included in the PEA economic analysis.

The geological information and Mineral Resources pertaining to the Kringel, Gropabo, Mattsmyra and Mansberg deposits have been disclosed previously in the public domain. For the purposes of this 2021 disclosure the Mineral Resources for Gropabo and Mattsmyra have been reported but are not included in the economic analysis of this 2021 PEA. No mineral resource is reported for the Mansberg deposit. For the purposes of this 2021 PEA the term Woxna Graphite Project (or the Project) includes only the Kringel deposit.

The graphite deposits are located within a 40 km metasediment and metavolcanic mineralisation trend in central Sweden which was discovered in 1983 by the Swedish Geological Survey (SGU) as part of a regional mapping programme. Exploration concessions were granted 1992 to a Swedish private company Mineral Resources AB (MIRAB) and later sold to Tricorona AB, the latter of which, brought the Woxna Mine into production in 1996.

The Woxna Mine was in operation until 2001 when production was halted as a consequence of falling graphite prices. Woxna Graphite AB was acquired by Burke Resources Limited, a Canadian private company, in 2011. Burke Resources subsequently merged with Tasex Capital Corp, a capital pool company listed on Toronto Venture Exchange, and subsequently changed its name to Flinders Resources Limited (Flinders). Flinders invested raised capital into refurbishing and upgrading the operations at Woxna and eventually recommenced graphite production in July 2014. However, falling graphite prices again led to production being suspended. A strategy was launched to develop the required downstream processes to produce higher value products, most importantly graphite for lithium-ion battery anodes. Production has not been reinstated and the Woxna Concentrator is currently maintained on a production-ready basis whilst downstream development continues.

Graphite displays unique characteristics such as resistance to oxidation, thermal shock and almost all chemical agents. Graphite combines electrical and thermal conductivity with lubricating effects due to its unique crystal structure. Thanks to its properties graphite has found its way into many industrial uses such as refractories, lubricants, electric motors, fire retardants and insulation. In addition to these existing markets, graphite is increasingly used in emerging technologies such as lithium-ion batteries, fuel cells, thermal energy storage and graphene applications. In order to be used as the anode active material in lithium-ion batteries, the natural flake graphite must by sized and shaped into small spheres, purified to at least 99.95% purity, and lastly coated, at which point the material is generally referred to as a Coated Spherical Purified Graphite (CSPG).



Source: ReedLeyton 2015

Figure 2-1: Woxna Graphite Project – location

China controls close to 100% of the production of Spherical Purified Graphite (SPG) which is then further refined into CSPG in China or other parts of Asia. With the expected dramatic growth in lithium-ion battery factories being planned in Europe LEM is looking to develop Woxna into a new supplier to the European market for CSPG.

LEM considers that it is favourably positioned in having access to a sustainable supply of commercial graphite from the Woxna Project at relatively low exploration/assessment costs as the Project has:-

- historically verified and current "Canadian Institute of Mining, Metallurgy, and Petroleum" (CIM) compliant Mineral Resources;;
- historical mining production figures for use in future mine design and costing;;
- historical, actual processing data for assessment of new production avenues;
- electricity sourced from hydropower with a minimal carbon footprint
- excellent infrastructure in the form of roads, power, ports, water supply and services; and
- close proximity to major existing and future graphite customers.

The Project currently has environmental/exploitation permission to process 100,000 tonnes (t) of graphitic material per annum (tpa) year, which allows for the production of approximately 10,000 t of graphite concentrate. Given the expected increase in demand for European sourced graphite in the near future, Woxna Graphite has directed efforts towards developing value-add processes for the Woxna Project in order to produce a CSPG suitable for the lithium-ion battery industry. Such production requires an upgrade of the existing Woxna Concentrator to 160,000 tpa and a new value-add production facility (VAP) which includes dry product sizing, spheronizing, high temperature thermal purification, and coating.

In the light of this corporate strategy, LEM, through its subsidiary Woxna Graphite, has commissioned Zenito Limited (Zenito), an independent firm of engineering consultants, to re-scope the Project. Zenito specialises in the development, design, and construction of mining projects and was requested to review, modify, and update the existing:

- Woxna Project Preliminary Economic Assessment (PEA) conducted for the Project by GBM Minerals Engineering Consultants 11 October 2013, noting this report was invalidated by the filing of a subsequent technical report in 2015;
- to upgrade the current design throughputs for the historical Woxna Concentrator to match the higher mill throughput capacity vs existing mining permit;
- to develop a process flow and costing for a VAP;
- to develop process flow sheet, resulting in two specific graphite products, namely CSPG and micronized graphite (as defined in Section 2.4); and
- assess the economic viability of the proposed changes.

Similarly engaged by LEM for this PEA are:

- ReedLeyton Consulting, to re-issue the CIM Mineral Resource estimate by ReedLeyton Consulting 24 March 2015;
- M.Plan International, to reassess the mine plan and production schedule for the Kringel deposit;
- Golder Associates, to examine the environmental aspects of the proposed plan especially the development of two TSFs; and

2.2 Purpose of the Technical Report and use of the term PEA

Woxna Graphite specified that the 2021 PEA be prepared by Zenito to demonstrate the investment case for investors and provide a guide for the company on how to proceed with development of the project. Consequently, the technical report has been prepared in accordance with the requirements of Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) (June 2011).

In terms of NI 43-101 definitions, a PEA means "an economic study, other than a pre-feasibility or feasibility study, that includes analysis of the *potential* viability of mineral resources" (NI 43-101 section 2.3(1)(b) and section 2.3(3)). The specific exemption for PEAs from the normal NI 43-101 prohibition on economic analysis/disclosure that includes/or is based on Inferred Mineral Resources has led to the common usage of the term in public disclosure for early stage exploration projects based on or including Inferred Mineral Resources.

Given that this definition of a PEA as a conceptual study of the potential viability a resource (NI 43-101 (section 3.4(e)), specific cautionary language is mandatory indicating that the economic viability of the mineral resource has not been demonstrated. Any disclosure that implies the PEA has demonstrated economic or technical viability would be contrary to NI 43-101 and the definition of PEA.

The Woxna Project however does not conform to these characteristics and is instead an advanced brownfields project with historical production and with only Measured and Indicated Resources included in the economic assessment. The economic analysis of the Project is preliminary only in the context of the design and costing of the proposed mine, proposed upgrade to the Woxna Concentrator and the VAP.

The Canadian Securities Administrators (CSA) published a staff notice in 2013 (Staff Notice 43-307 Mining Technical Reports – Preliminary Economic Assessments) clarifying its position regarding the use and disclosure of PEAs by issuers in the mining sector) as quoted below. This guidance however has not yet been published as an update to the NI 43-101 Standards of Disclosure:

"CSA broadened the definition of PEA in response to industry concerns that issuers needed to be able to take a step back and re-scope advanced stage projects based on new information or alternative production scenarios. In this context, the revised definition is based on the premise that the issuer is contemplating a significant change in the existing or proposed operation that is materially different from the previous mining study. In most cases, this will also involve considerably different economic parameters and capital investments. Examples of a significant change are a different scale of proposed operation (higher or lower throughput), a different scope of operation (higher or lower grade), the inclusion of other types of mineralisation (oxide vs.sulphide), the use of alternative mining methods (open pit vs. underground), or the use of alternative processing technology".

The Woxna Project clearly conforms to the re-scoping category of an advanced project as described above and the use of the term PEA is therefore appropriate but does not imply an early stage of development.

The accuracy levels of the Woxna Mine design and costing, as well as that of the concentrator and VAP, are appropriate for the PEA level of study and these will be refined and upgraded in a future Preliminary Feasibility Study (PFS)/Definitive Feasibility Study (DFS). The required cautionary statements are therefore applicable to the economic analysis.

2.3 Terms of Reference, scope and purpose of report

The 2021 Zenito PEA has been conducted according to the following terms of reference and specific project criteria as defined by Woxna Graphite:

- the PEA has been conducted according to NI 43-101) (June 2014) disclosure guidelines and the results reported in a technical report compiled and signed-off by various independent specialist consultants:
- the economic analysis in the PEA considers only one of the four graphite deposits held by Woxna Graphite, namely the Kringel deposit. The mineral resource estimates of the remaining two exploitation concessions, have been previously reported (Flinders 2013, 2015) but are excluded from the current economic analysis. The updated Mineral Resources for Gropabo and Mattsmyra deposits are provided and future PFSs or DFSs may include mining and processing of these remaining deposits (Gropabo and Mattsmyra). The Mansberg deposit has only a historic resource
- the valid environmental permit currently held by Woxna Graphite for the Kringel deposits was granted under now defunct legislation. The recommendation is that in order to benefit from the results in the PEA, an application be made for a permit under the new Environment Act that includes permission to extract 160 ktpa (as opposed to the 100ktpa in the current permit), would permit expansion of the open-pit and associated TSF, as well as potentially include the development of the VAP:
- the historical and previous Flinders Mineral Resource estimates were developed without the constraint of an applied mine plan. In the light of more rigorous compliance requirements, the Mineral Resources were reported by ReedLeyton Consultants Limited (ReedLeyton) within the 2021 PEA mine plan and pit shell configuration. No new exploration data was included;

- a mine plan and mine production schedule were to be developed that incorporated the new production target of 160 ktpa. Several different approaches to the mining of the deposit were to be interrogated and the most suitable selected for both economic and production optimisation;
- the historical Woxna Concentrator will be expanded and upgraded to produce approximately 14.7 tpa flotation concentrated graphite for further beneficiation in the VAP. The term Woxna Concentrator refers to this expanded concentrator;
- the Woxna Concentrator is located on the Woxna Mine site whilst the VAP for the purpose of this PEA has been located in an existing brownfield industrial facility in the near-by town Edsbyn;
- 'process plant' describes the Woxna Concentrator + VAP in totality;
- two beneficiated graphite products are to be produced from the VAP. Firstly, approximately 6,604 tpa CSPG at a minimum purity of 99.95% C and an average of approximately 7,749 tpa 93% C micronized graphite (see definitions in Section 2.4);
- the design and costing of the TSF are to be done on the basis that two tailings streams will be produced: one tailings stream being the Non-acid Generating (NAG), while the other is the Potential Acid Generating (PAG) stream. The NAG tailings will represent 90% of the tailings production and will be deposited on the existing tailings area, whilst the PAG tailings represent 10% of the total tailings production and will be deposited in a separate lined TSF adjacent to the current TSF;
- the costing for the Woxna Concentrator and VAP were conducted at ±30% accuracy whilst those for the TSF were estimated at 50% accuracy;

Zenito's primary obligation in preparing technical reports for the public domain is to describe mineral projects in compliance with the reporting codes applicable under the jurisdiction in which the company operates. In this case, it is the NI 43-101. Zenito prepared the technical report and the economic analysis based on the principle of reviewing and interrogating both the work of previous owners and specialist experts who have contributed to the technical information available for the asset.

In the execution of its mandate, Zenito undertook to identify the factors of both a technical and economic nature, which would impact the future viability of the deposits and considered the strategic merits of the deposits utilising the best practice due diligence methodologies.

The technical report is prepared for potential investors and their advisors. The technical report has been compiled in order to incorporate all the available and material information that potential future finance providers and their advisors would reasonably require in order to make balanced and reasoned judgements regarding the techno-economic merits of the mineral asset.

2.4 Terms and conventions

Throughout the technical report the following terms and conventions have been applied:

- where information has been generated or reported by Flinders this name has been retained albeit that the company is the holding company for Woxna Graphite and ultimately changed names to LEM. These three company names have been used throughout and can be considered interchangeable;
- 'graphite' is used in the mineralogical sense and refers to naturally occurring carbon atomically arranged in loosely bound sheets (see section 7.3). It does not refer to a mixture of graphite and gangue material that is mined;
- the mixture of gangue and graphite that constitutes the target material has no geologically defined name. This has therefore been termed 'mineralised material' or 'graphite+gangue'. The graphite+gangue constitutes, and is referred to as, the RoM;
- natural graphite occurs in several coarse and fine grained forms (see Section7.3) 'flake' and 'amorphous' respectively. These terms can apply to the graphite in the mined material or beneficiation product;
- some confusion can be generated from the difference in nomenclature applied to various sources and grades/concentrations of carbon quoted herein and the following are important:

- 'C' refers to elemental carbon and the atomic structure is unspecified;
- analysis of RoM material for carbon content, irrespective of atomical structure/form, results in % carbon figure (% C) quoted in assays of exploration samples and carbon content in the beneficiated material;
- 'Cg%' refers to the percentage of carbon in the form of graphite and is quoted in the Mineral Resource estimates and refers to in situ graphite;
- 'tCpa' refers to tonnes carbon (in the form of graphite) per annum produced and is qualified by the source of the carbon e.g. 'tCpa RoM' refers to the tonnes C in the RoM;
- graphite concentrate produced by flotation in the Woxna Concentrator is simply referred to as 'graphite concentrate' and the grade of graphite is quoted as C%;
- graphite concentrate that has been processed in a spheronizer is simply referred to as 'spherical graphite';
- spheronized, purified, coated graphite product from the VAP, which is a graphite product suitable as the active anode material in lithium-oin batteries, has been termed 'Coated Spherical Purified Graphite' (CSPG). The grade of the carbon in CSPG is quoted as C%;
- an additional VAP product is jet milled, micronized fines from the spheronizing/purification stream which is referred to as 'micronized graphite';

2.5 Primary information sources

This report makes use of the following major primary information sources:

- Historic exploration databases
- Previous modelling information
- Woxna Graphite Restart Project, Preliminary Economic Analysis, GBM Minerals Engineering Consultants Limited, 11 October 2013, (0482-RPT-001 Rev 0).
- Scoping Study for a Demonstration Facility for Thermal Processing of Woxna Graphite, HATCH Ltd., 8 June 2018, (H355799-00000-210-066-0001 Rev 0).
- Benchmark Mineral Intelligence, "Uncoated & Coated Spherical Graphite Market Overview for Leading Edge Materials Corporation," 2020
- A leading European graphite specialist consultancy (name not disclosed due to confidentiality), "Market Information on Micronized Graphite from the Woxna Mine," 2021.
- A proprietary coating technology provider, "Scoping Study Report, Carbon Coating Plant, Leading Edge Battery Anodes Plant," 2021.

Zenito has also used various other information sources which are referenced in Section 27 and where applicable throughout this report.

2.6 Qualified persons

The following independent consultancies and their Qualified Persons (as shown in Table 2-1) are responsible for the information and data interpretation in this PEA. The Qualified Persons are members in good standing of appropriate professional institutions. The technical report is based on and fairly reflects the information and supporting documentation prepared by the Qualified Persons listed as signatories of this document. The Qualified Persons who are signatories to this report have sufficient experience that is relevant to the style of mineralisation and type of deposit and to the activity being undertaken, to qualify as Qualified Persons, as defined by the NI 43-101 and to express their professional opinions on the mineral deposits.

- Zenito Qualified Person is Christopher Stinton, BSc (Hons), CEng MIMMM.
- ReedLeyton Consulting Limited (ReedLeyton)- an independent firm specialising in estimation of mineral resources commissioned to review, validate, and update the PEA Mineral Resource statements. The ReedLeyton Qualified Person is Geoffrey Reed, B App Sc, MAusIMM.
- M.Plan International Limited (MPlan) - an independent firm of engineering consultants commissioned to review and update the PEA mining reserve statements. The MPlan Qualified Person is Mathieu Gosselin, Eng.

- Golder Associates AB, Sweden (Golder) - commissioned to review and update the design of the TSF to accommodate the production from the upgraded Woxna Concentrator and VAP. The Golder Qualified Person is Henning Holmström, M Sc, PhD, MAusIMM. and

2.7 Qualified person site visit

A number of visits by various Qualified Persons and other parties have been carried out to inspect the Project site and verify its characteristics. The following site visits directly applicable to the Project have been carried out by Zenito, Golder, ReedLeyton, M.Plan representatives:

- Chris Stinton, November 2012.
- Chris Stinton, October 2013.
- Representing Golder, Rita Kamera and Romain Girard, accompanied by Peter Young (Woxna), 4th November 2020.
- Henning Holmström has visited the Woxna Site regularly between 2011-2013.
- Geoffrey Reed, Kringel deposits -12th to 13th June 2012 and 17th to 18th June 2014. The remaining three deposits in June 2014
- Mathieu Gosselin, 7th to 8th May 2012 and 14th February 2013.

To be clear, only the 2020 Golder visit was directly related to the activities of this PEA. The Qualified Persons have taken the necessary steps to validate that there is no new material or scientific information about the property since their last site visits.

2.8 Verification

The Qualified Persons have inspected the Project site and carried out due diligence reviews of the information provided to them by Woxna Graphite and previous owners for the preparation of this technical report. The Qualified Persons are satisfied that the information provided was accurate at the time of the estimation and that the interpretations and opinions expressed in them were reasonable and based on the then current understanding of mining and processing techniques and costs, economics, mineralisation processes and the host geologic setting. The Qualified Persons have undertaken high level verification where possible and have made reasonable efforts to verify the accuracy of the data relied on in this report.

Woxna Graphite has reviewed draft copies of this report for factual errors, but edits made as a result of these reviews did not involve any alteration to the conclusions made by the Qualified Persons.

2.9 Independence and financial interest disclaimer

Neither Zenito, Golder, MPlan, ReedLeyton, nor any of their consultants employed in the preparation of this technical report, have or have had any beneficial interest in the deposits of Woxna Graphite or its subsidiaries capable of affecting their ability to give an unbiased opinion and, have not and will not, receive any pecuniary or other benefits in connection with this assignment, other than normal consulting fees. Zenito, Golder, MPlan, and ReedLeyton have been paid fees and will continue to be paid fees for ongoing consultancy input in accordance with normal professional consulting practice.

2.10 Qualified Persons sign-off

The various sections of this technical report were prepared by or under the supervision of the Qualified Person(s) identified in Table 2-1 as shown below:

Table 2-1: Responsible Qualified Persons

Section	Section Title	Qualified Person(s)
1	Summary	Christopher Stinton (Zenito)
2	Introduction	Christopher Stinton (Zenito)
3	Reliance on Other Experts	Christopher Stinton (Zenito)
4	Property Description and Location	Christopher Stinton (Zenito)
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Christopher Stinton (Zenito)
6	History	Christopher Stinton (Zenito)
7	Geological Setting and Mineralisation	Geoffrey Reed (ReedLeyton)
8	Deposit Types	Geoffrey Reed (ReedLeyton)
9	Exploration	Geoffrey Reed (ReedLeyton)
10	Drilling	Geoffrey Reed (ReedLeyton)
11	Sample Preparation, Analyses, And Security	Geoffrey Reed (ReedLeyton)
12	Data Verification	Geoffrey Reed (ReedLeyton)
13	Mineral Processing and Metallurgical Testing	Christopher Stinton (Zenito)
14	Mineral Resource Estimates	Geoffrey Reed (ReedLeyton)
15	Mineral Reserve Estimates	Not Applicable
16	Mining Methods	Mathieu Gosselin (MPlan)
17	Recovery Methods	Christopher Stinton (Zenito)
18	Project Infrastructure	Christopher Stinton (Zenito)
19	Market Studies and Contracts	Christopher Stinton (Zenito)
20	Environmental Studies, Permitting, And Social or Community Impact	Henning Holmström (Golder)
21	Capital and Operating Costs	Christopher Stinton (Zenito)
22	Economic Analysis	Christopher Stinton (Zenito)
23	Adjacent Properties	Geoffrey Reed (ReedLeyton)
24	Other Relevant Data and Information	All Qualified Persons
25	Interpretation and Conclusions	All Qualified Persons
26	Recommendations	All Qualified Persons
27	References	All Qualified Persons

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons have relied on expert opinions and information outwith the Qualified Person's expertise. Such information was provided by Leading Edge Materials Corp. pertained to environmental considerations, taxation and legal aspects including mineral tenure, surface rights and material contracts.

3.1 Property description and location and Adjacent properties

For the purposes of *Section 4 (Property Description and Location*) and Section 23 (Adjacent Properties) of this report the Qualified Person has relied on property ownership data provided by Woxna Graphite. This information is believed to be essentially complete and accurate to the best of the Qualified Person's knowledge and no information has been intentionally withheld that would affect the conclusions made herein. The Qualified Person has not researched the property title or mineral rights for the Project and expresses no legal opinion as to the ownership status of the property.

3.2 Market studies and contracts

For the purposes of *Section 19 (Market Studies and Contracts*) of this report the Qualified Person has relied on information pertaining to market studies and material contracts provided by Leading Edge Materials Corp the sources referenced within the section including:

- Benchmark Mineral Intelligence, "Uncoated & Coated Spherical Graphite Market Overview for Leading Edge Materials Corporation," 2020.
- Leading European Graphite specialist consultancy (Name of consultant withheld due to confidentiality), "Market Information on Micronized Graphite from the Woxna Mine," 2021.

The Qualified Person has reviewed the information provided by Leading Edge Materials Corp believes this information to be accurate and adequate for use in this PEA technical report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

Woxna Graphite, LEM's 100% owned Swedish subsidiary, owns eleven exploration permits (see Section 4.3.1) and four exploitation concessions (see Section 4.3.2) over the Kringel, Gropabo, Mattsmyra and Mansberg deposits which form part of a 40 km mineralisation trend in central Sweden. The relative locations of the four exploitation concessions are illustrated in Figure 1-1. It is important to note that whilst this disclosure contains Mineral Resource estimates for three deposits, the economic analysis for the PEA considers only one of these deposits, namely the Kringel deposit.

The Kringel deposit is located at 61° 26′ 08.94″N and 15° 36′ 16.31″E, approximately 8 km westnorth-west (WNW) of the town of Edsbyn in the Ovanåker Municipality, Gävleborg County, in the Kingdom of Sweden.

4.2 Property, title, and surface rights

Rules and regulations pertaining to mining exploration in Sweden are clearly outlined in the "Guide to Mineral Legislation and Regulations in Sweden" (2000) available from the offices or the website of the Geological Survey (www.sgu.se). The Mining Inspectorate of Sweden provides clear directives, available from the Inspectorate website (www.bergsstaten.se), for conducting exploration.

Under Swedish law land required for exploitation is normally acquired by the mining company through contracts of sale or leases. If there is a contract of sale, a property registration procedure must generally be undertaken through the Land Survey authority in order for registration of title to be granted.

Before any land, inside or outside the exploitation concession area, may be used for the purposes of mining, it has to be designated appropriate by the Mining Inspector (markanvisning). This procedure usually regulates the compensation to be paid to affected landowners, normally on the basis of an agreement between the mining company and the landowners, together with any other parties whose rights may be affected. If no agreement can be achieved, the Mining Act regulates the procedure where the land can forcefully acquired at a premium to market value.

The valid extraction/environmental permit currently held by Woxna Graphite was granted under now defunct legislation. Whilst the permit is valid, the environmental QP has recommended that application be made for a permit under the new Environment Act in order to benefit from the expanded operations demonstrated in the PEA. Such a new application would include permission to extract 160 ktpa (as opposed to the 100 ktpa under the previous permit), would permit expansion of the open-pit and associated TSF, as well as potentially the development of the VAP.

The surface freehold owners are mostly Swedish and international forestry companies. The extent and location of these surface right holders in the vicinity of the Kringel exploitation concession are known to Woxna Graphite and access rights have been verified by Golder Associates AB.

4.3 Permits

4.3.1 Exploration permits

The Minerals Act relates to the exploration and exploitation of certain mineral deposits on land, regardless of the ownership of the land. Applications for permits etc. are made to the Mining Inspectorate (Bergsstaten). The Act defines to which mineral substances its provisions apply; these are known as concession minerals. Concession minerals are divided into three categories, namely traditional ores, certain industrial minerals, and finally oil, gas, and diamonds. Other minerals and other kinds of rock, gravel and sand are excluded from the Act and are normally referred to as landowner minerals.

An exploration permit (undersökningstillstånd) gives access to the land and an exclusive right to explore within the permit area. It does not entitle the holder to undertake exploration work in contravention of any environmental regulations that apply to the area. Applications for exemptions are normally made to the County Administrative Board.

An exploration permit is granted for a specific area where a successful discovery is likely to be made. It should be of a suitable shape and size and no larger than may be expected to be explored by the permit holder in an appropriate manner. A permit is to be granted if there is reason to assume that exploration in the area may lead to the discovery of a concession mineral.

An exploration permit is initially valid for a period of three years, after which it can be extended up to a total of 15-years if special conditions are met.

Compensation must be paid by the permit holder for damage or encroachment caused by exploration work.

When an exploration permit expires without an exploitation concession being granted, the results of the exploration work undertaken must be reported to the Mining Inspector.

4.3.2 Exploitation concessions

An exploitation concession (bearbetningskoncession) gives the holder the exclusive right to exploit a proven, extractable mineral deposit for a period of 25-years, which may be extended. Permits and concessions under the Minerals Act may be transferred with the permission of the Mining Inspector.

An exploitation concession relates to a distinct area, designated on the basis of the location and extent of a proven mineral deposit. A concession may be granted when a mineral deposit is discovered which is probably technically and economically recoverable during the period of the concession, and if the nature and position of the deposit does not make it inappropriate to grant a concession. Special provisions apply to concessions relating to oil and gaseous hydrocarbons.

Under the provisions of the Environmental Code, an application for an exploitation concession is to be accompanied by an environmental impact assessment (EIA). Applications are considered in consultation with the County Administrative Board, taking into account whether the site is acceptable from an environmental point of view.

An exploitation concession does not give the right to commence exploitation since this requires an environmental permit granted by the environmental court and potential other permits.

4.3.3 Permit from the environmental court

Under the rules of the Swedish Environmental Code, a special EIA for the mining operation must always be submitted to the Environmental Court, which examines the impact of the operation on the environment. The Court also stipulates the conditions which the operation is to meet. The conditions that require fulfilment for the current Environmental Permit over the Project and VAP areas are provided in Section 20.2.3.

4.3.4 Woxna Graphite – legal tenure

Woxna Graphite, owns eleven exploration permits and four exploitation concessions over the Kringel, Gropabo, Mattsmyra and Mansberg deposits as summarised in Table 4-1.

Table 4-1: Woxna Graphite legal tenure

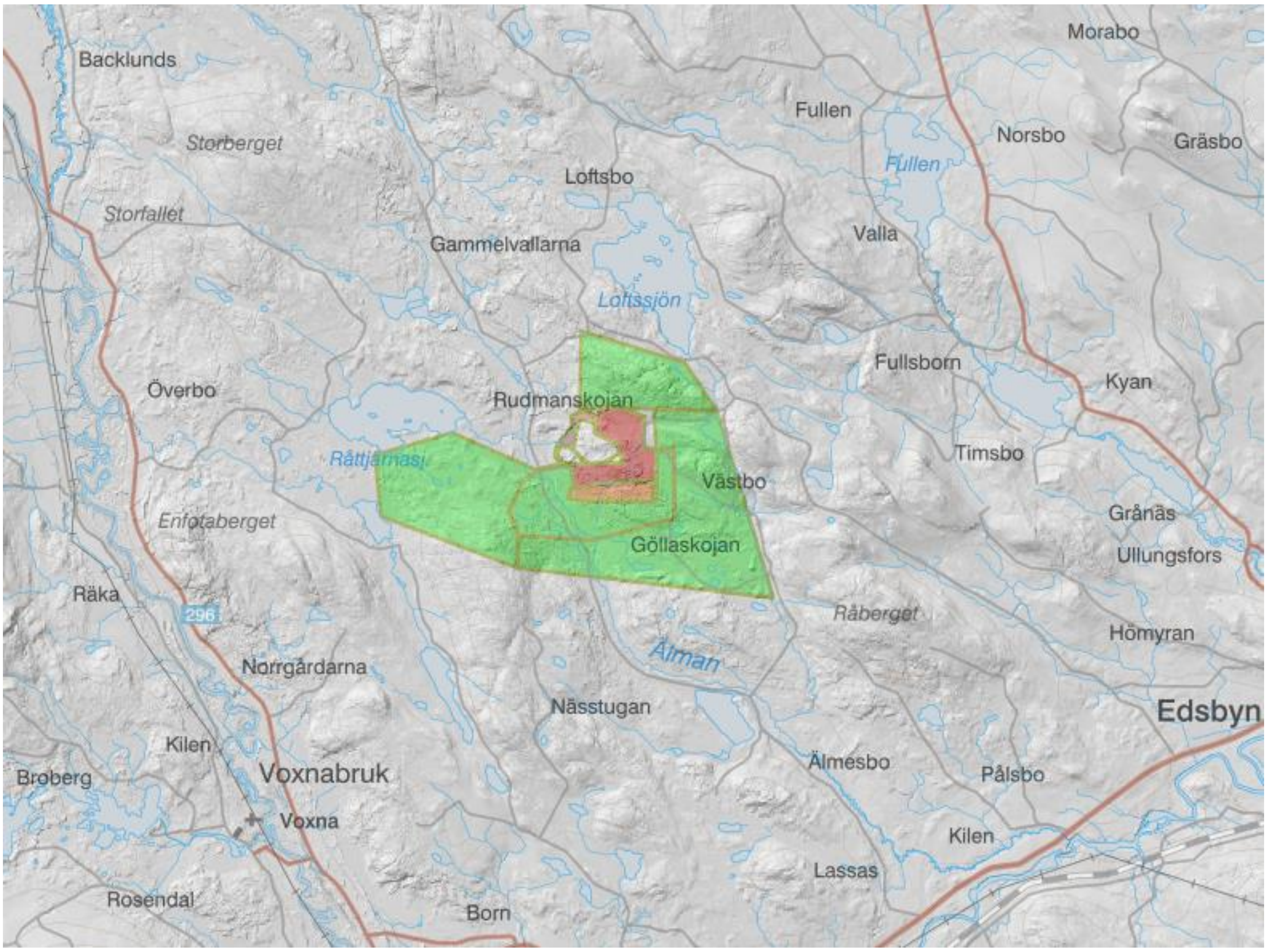
Property	Type of Mineral Tenure	Reference number	Area (ha)	Valid for (years)*	Date of issue or receipt	Expiration date	Conditions/comments
Kringel	Exploration permit	Kringeltjärn Nr 8	157	3+1	29-Nov-19	29-Nov-23	Dnr BS 200-955-2019
		Kringeltjärn Nr 2001	275	3	04-Nov-20	04-Nov-23	Dnr BS 200-446-2020
		Kringeltjärn Nr 2002	413	3	04-Nov-20	04-Nov-23	Dnr BS 200-477-2020
		Kringeltjärn Nr 2003	147	3	04-Nov-20	04-Nov-23	Dnr BS 200-448-2020
	Exploitation concession	Kringelgruvan K Nr 1	30.76	25	03-Nov-16	03-Nov-41	New issue from Mining Inspectorate - Document number Dnr 22-1271-2015
	Environmental permit	Woxna Graphite AB	30.76		Sept/Oct 1992	None	100 kt per annum mineralized material extraction
Gropabo	Exploration permit	Gropabo Nr 5	58.37	3+1	20-Feb-20	20-Feb-24	
	Exploitation concession	Gropabo	18.20	25	21-Feb-00	21-Feb-25	
	Environmental permit	Gropabo 4:1 and Norra Svensbo 14:1	18.2	7	21-Feb-05	21-Mar-12	Mål nr M 3659-02. The condition of issue was that mining would begin within 7 years hence the expiration
Mattsmyra	Exploration permit	Mattsmyra Nr 7	42.18	3+1	20-Feb-20	20-Feb-24	
		Mattsmyra Nr 8	37.54	3+1	23-Oct-19	23-Oct-23	
		Mattsmyra Nr 9	44.19	3+1	23-Oct-19	23-Oct-23	
		Mattsmyra Nr 10	30.26	3+1	21-Nov-19	21-Nov-23	
		Sub-total	154.17				
	Exploitation concession	Mattsmyra	72.97	25	21-Feb-00	21-Feb-25	
	Environmental/extraction permit	Gropabo 4:1 and Norra Svensbo 14:1	72.97	7	21-Mar-05	21-Mar-12	Mål nr M 3659-02. . The condition of issue was that mining would begin within 7 years hence the expiration
Mansberg	Exploration permit	Mansberg Nr 4	65.57	3+1	21-Nov-19	21-Nov-23	
		Mansberg Nr 5	240.86	3+1	21-Nov-19	21-Nov-23	
		Sub-total	306.43				

Property	Type of Mineral Tenure	Reference number	Area (ha)	Valid for (years)*	Date of issue or receipt	Expiration date	Conditions/comments
	Exploitation concession	Mansberg	24.77	25	27-Dec-99	27-Dec-24	
	Environmental/extraction permit	No application filed					

Source: Woxna Graphite 2021

* In June 2020 legislation was introduced that extended existing exploration permits by one year due to the effects of the Covid pandemic.

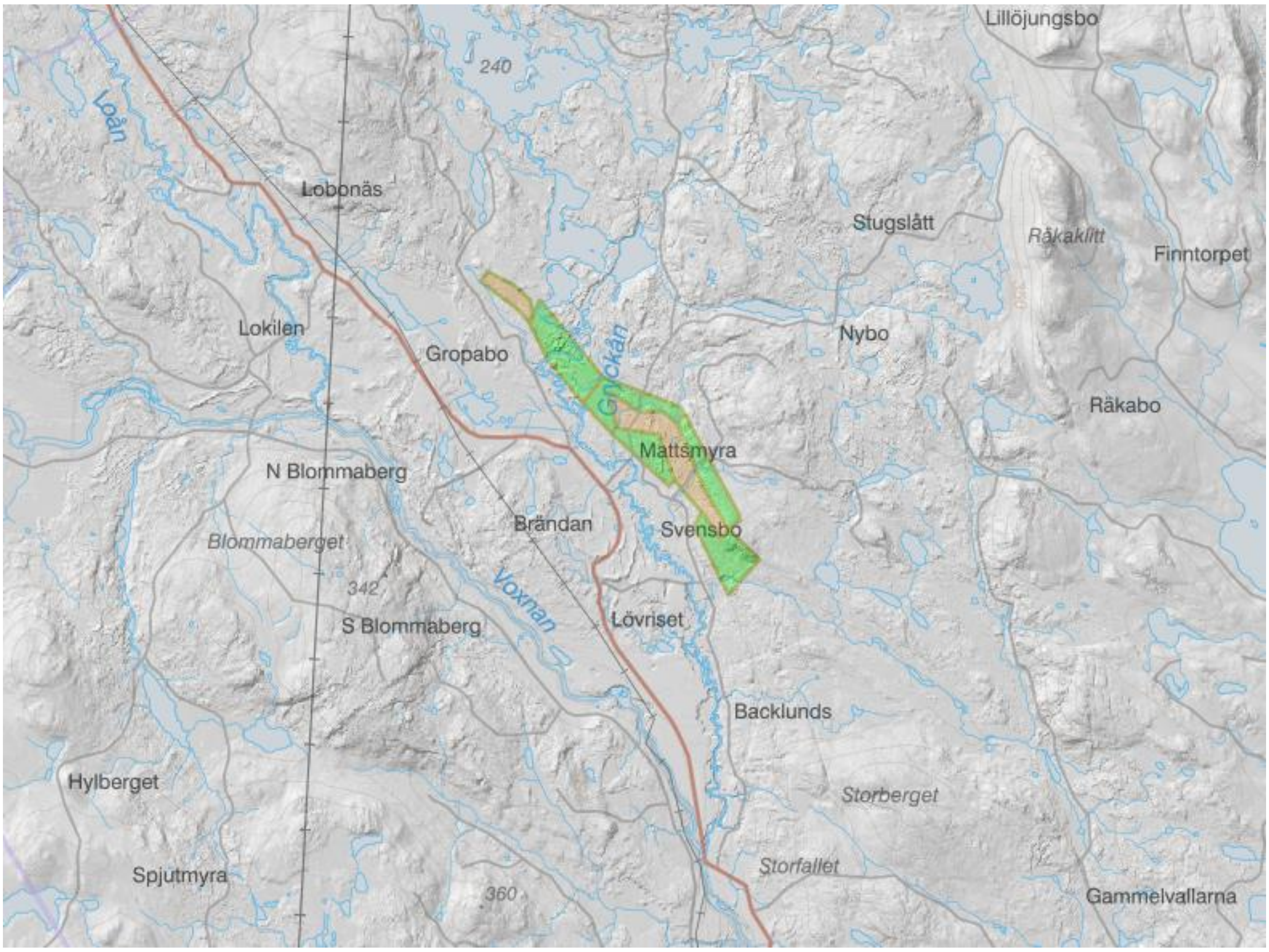
The relative locations and juxtapositions of the exploration permits and exploitation permits are shown in Figure 4-1 to Figure 4-3 and were provided by Woxna Graphite and sourced from the mining inspectorates "Mineral Permits" map viewer.



Source: Woxna Graphite 2021

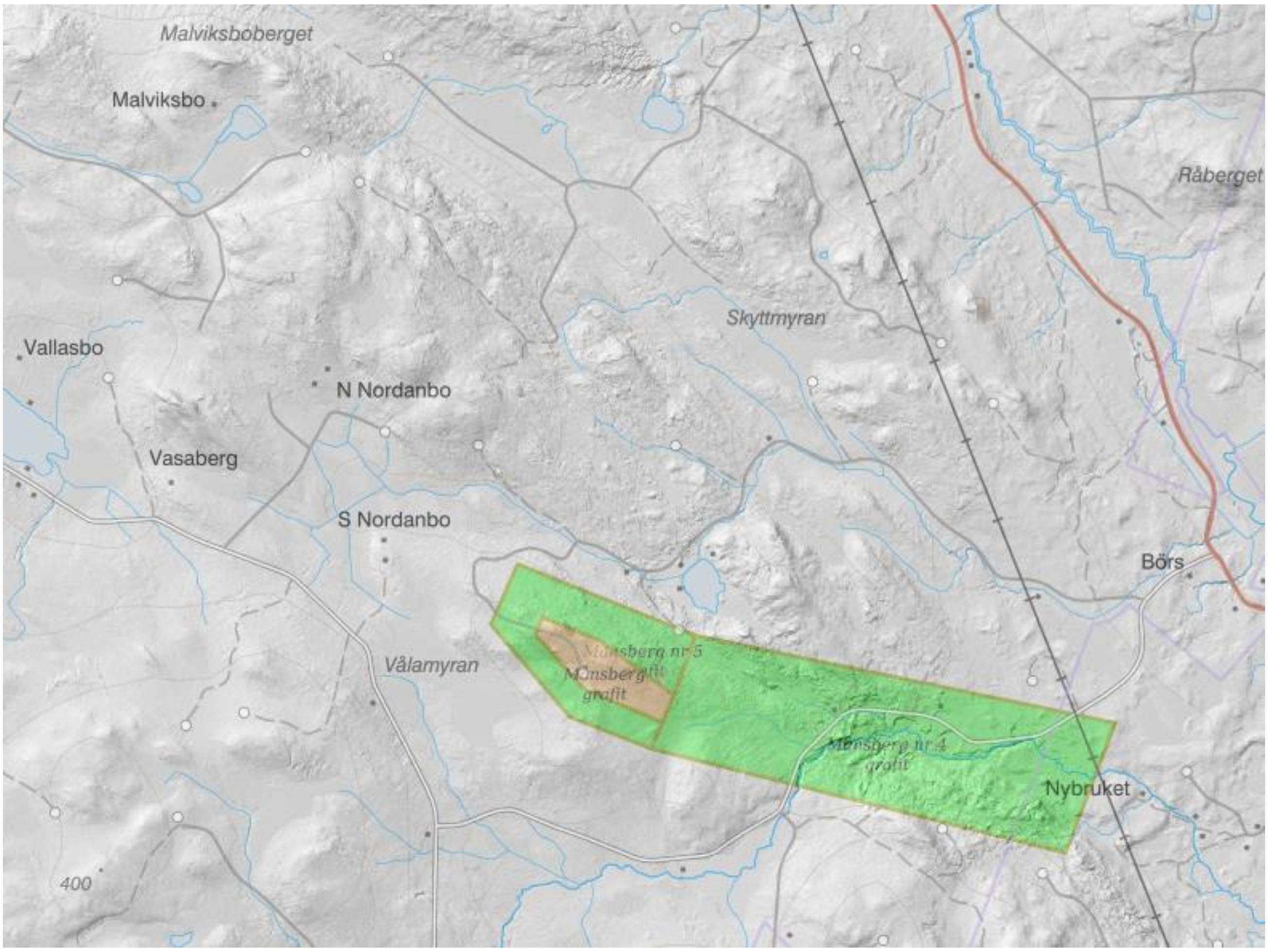
Figure 4-1: Kringel deposit- legal tenure

Legend:
- Exploitation concession.
- Exploration permit, metals, and industrial minerals.
- Land allocation to concession.
- Gallant peat concessions.



Source: Woxna Graphite 2021

Figure 4-2: Gropabo and Mattsmyra deposit - legal tenure



Source: Woxna Graphite 2021

Figure 4-3: Mansberg deposit - legal tenure

The exploitation concession co-ordinates in Swedish reference frame 1999 (SWEREF99TM) and Swedish National Grid RT90 (1990) are provided in Table 4-2; Note that in Swedish convention x and y are interchanged from normal usage elsewhere.

Table 4-2: Exploitation Concession co-ordinates

Deposit	Reference Datum	Point	X Meters	Y Meters
Kringel	SWEREF99TM	1	6,808,741	533,122
		2	6,808,436	533,123
		3	6,808,433	532,191
		4	6,808,477	531,986
		5	6,808,493	531,912
		6	6,808,601	531,937
		7	6,808,628	531,997
		8	6,808,632	532,024
		9	6,808,724	532,590
Mattsmyra	RT90	**Point**	**X Meters**	**Y Meters**

Deposit	Reference Datum	Point	X Meters	Y Meters
		1	6,821,371	1,480,268
		2	6,821,070	1,480.71
		3	6,821,080	1,481,030
		4	6,820, 900	1,481,240
		5	6,820,000	1,481,590
		6	6,819,390	1,481,950
		7	6,819,250	1,481,840
		8	6,819,635	1,481,575
		9	6,820,090	1,481,190
		10	6,820,750	1,480,970
		11	6,820,850	1,480,370
		12	6,821,140	1,480,310
Gropabo	RT90	Point	X Meters	Y Meters
		1	6,823,100	1,478,562
		2	6,822,890	1,478,945
		3	6,822,690	1,479,158
		4	6,822,453	1,479,200
		5	6,822,366	1,479,200
		6	6,822,569	1,479,977
		7	6,822,960	1,478,460
Mansberg	RT90	Point	X Meters	Y Meters
		1	6,793,515	1,512,562
		2	6,793,360	1,512,965
		3	6,792,990	1,512,455
		4	6,792,810	1,512,378
		5	6,793,040	1,512,860
		6	6,793,400	1, 512,545

4.4 Royalties

For exploitation concessions granted after 2005, a royalty is paid on the value of minerals produced at a rate of 0.2%, which is shared between the landholder and the State each receiving 0.15% and 0.05% respectively (www.sgu.se/en/mining-inspectorate/legislation/mineral-act-199145).

Though the Kringel deposit exploitation concession was granted prior to 2005 (see Table 4-1), and therefore this royalty potentially does not apply, the royalty is conservatively assumed applicable in the PEA financial analysis.

4.5 Environmental liabilities

Zenito understands that there are no current outstanding environmental liabilities on any of the exploitation concessions and, as required by Swedish law, all landowners identified by Leading Edge Materials Corp has been informed by the Swedish Inspectorate of Mines (Bergsstaten) that an exploitation concession has been granted in accordance with Chapters 1.1 and 2 of the Mineral Act.

No environmental or planning permitting is required for geological mapping and minor, scattered hand till sampling. However, permits are required from the district authorities for systematic till sampling, trenching, and drilling programmes.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility and infrastructure

The Kringel deposit lies within the Ovanåkers Kommun in the county of Gävleborg, Central Sweden. The property is accessible by road approximately 300 km northwest from Stockholm on highway E4 to Soderhamn, then highway 50/301 to the town of Edsbyn. From Edsbyn highway 301 continues for approximately 10 km before turning off on to a gravel road that accesses the centre of the property, a distance of just over 10 km. Local access to the Kringel exploitation concession area is via unsealed all-weather forestry roads. Refer to Figure 5-1 for illustration of the local topography and site access.

At Kringel, a processing plant on care-and-maintenance, a TSF, office infrastructure and power and water services exist next to a partially exploited open pit.

5.2 Climate and vegetation

The climate is comparatively temperate, considering the Project's northern latitude. The climate is typical of Fennoscandia with cool summers and cold winters. The principal moderating influences are the Gulf Stream and the prevailing westerly winds, which blow in from the relatively warm Atlantic Ocean. In winter, these influences are offset by cold air masses that periodically sweep in from the east.

At Edsbyn, some 10 km to the southeast of the project area, the monthly average minimum temperature ranges from -8°C to +11°C and the range of average maximum monthly temperatures is -1°C to +23°C. Edsbyn receives 30 mm to 70 mm of precipitation per month with fall and winter typically drier and spring and summer typically wetter periods.

The operating season is all year round, with possible short and minor disruptions at the height of winter with snowfall and very low temperatures.

The Project environs comprise of stands of commercial timber. Vegetation cover is a mixture of pine and fern forest, with some localised bogs in low-lying areas. The Kringel concession area, open-pit, concentrator plant and TSF are largely disturbed/brownfields and cleared of local vegetation.

5.3 Availability of local resources

Local services, in terms of machine and engineering plant maintenance, are available in Edsbyn. Regional road, rail, and service infrastructure is well developed.

The local economy has been focussed on forestry and plantation cropping since the 1890s. This is now a largely mechanised enterprise and uses similar types of machinery to that used in mining operations. There are some localised seasonal pasture and very minor cropping. The nearby Woxna River is used for tourism and recreation.

All industrial needs, services, provisions, and supplies are readily and commercially available, of high standard and typical of the Swedish modern industrial economy. Woxna Graphite has excellent access to infrastructure, services, electricity, supplies and a skilled and educated labour force. The town of Bollnas lies about 30 km east of Edsbyn and has a population in excess of 12,000.

5.3.1 Power source

The national power grid extends throughout the region and connected grid power is available at the Woxna Concentrator and open-pit.

Telecommunication, including internet and mobile telephone services, are widely available at the Woxna Mine and processing site.

5.3.2 Water source

Water resources are plentiful. The Kringel site water is supplied via local water source.

5.3.3 Personnel

Sweden has a long history of mining and local and specialised labour is widely available. The local population numbers approximately 11,000 people.

All social needs and services such as accommodation, supplies, communications, etc. are readily and commercially available.

Skilled local labour to resource the operation is expected to be available, some of which may have worked at the Project previously.

5.4 Physiography

The landscape is typical of glacial environments formed during the most recent Quaternary ice age with smoothed highlands, shallow lakes and extensive boggy lowlands. Broad valleys were scoured out in the direction of glacial transport and are flanked by low-lying hills underlain by resistant lithological units. The landscape of Sweden is dominated by low rolling hills (70%) which are covered by glacial moraine and flat lowlands (30%) comprised of bogs and lakes.

The Project elevation ranges from approximately 220 to 280 metres above mean sea level (mamsl) and comprises northwest–southeast (NW-SE) orientated low hills with trellised local stream drainage and numerous freshwater lakes, the latter of which, the Råttjärnasjön and Loftssjön are the largest (see Figure 5-1). The local drainage, in the main, ultimately flows to the Woxnan River which is an incised meandering river to the south of the Kringel exploitation concession. The Woxnan River is the source of hydropower in the district.

The graphite mineralisation of the Kringel deposit is located on the southern slope of the Gräsberget hill (approximately 250 mamsl) and traverses a local drainage system as shown in Figure 5-1.

A topographic terrain model was constructed for the site (Section 20- Figure 20-7) and topographic data from the exploration programmes also provided information for the construction of the geological models, the mine design and the environmental studies.

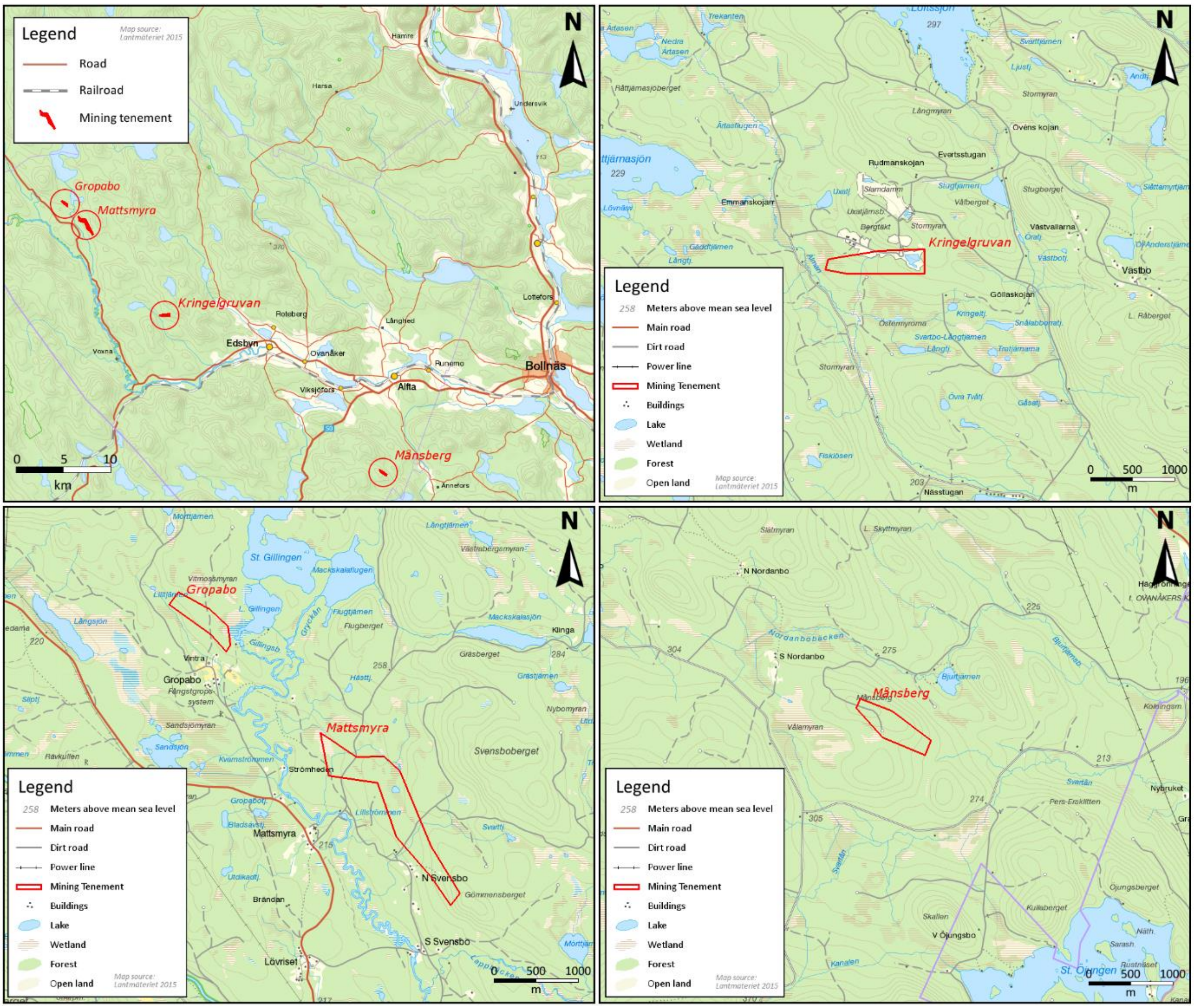


Figure 5-1: Topography and Access, Kringel, Mattsmyra, Gropabo and Månsberg,

6 History

The initial discovery of graphite in the region was made in 1983 by a prospector engaged by the SGU as part of a regional mapping programme tracking large boulders in Quaternary age moraine. The original surveys were directed at uranium detection using airborne radiometric surveys. Following discovery of the deposits in the early 1980s, exploration proceeded under the direction of the precursors to the SGU, namely Sveriges Geologiska AB (SGAB) and Nämnden för statens gruvegendom(NSG) as summarised in Table 6-1.

Table 6-1: Historical ownership and exploration – 'Woxna Graphite Project'

Deposit	Date	Investigating Company	Exploration technique	Ownership
Kringel	1986	Sveriges Geologiska AB (SGAB) Nämnden för statens gruvegendom (NGS)	Slingram geophysical ground survey	Precursors to the Swedish Geological Survey (Sveriges Geologiska Undersökning - SGU)
	1987		Trenching	
	1988		Drilling	
	1989		Second drilling campaign	
	1989		Additional ground geophysics to extend the target area	
	1992			Disposal by SGU to Mineral Resources AB (MIRAB)
	Unknown			Disposal by MIRAB to Tricorona AB
	1996		Brought into production by Tricorona AB	Tricorona AB
	2001		Production ceased	
	2011			Acquisition by Flinders Resources Limited
	2011	Coffey Mining Limited		NI43-101 Mineral Resource disclosure by Flinders for the Kringel deposit
Mattsymyra	1983	SGAB and NGS	Airborne EM	
			First drilling campaign	
	1989		Second phase drilling	
	1992			Disposal by SGU to Mineral Resources AB (MIRAB)
	Unknown			Disposal by MIRAB to Tricorona AB
Gropabo	1983	SGAB and NGS	Discovered by airborne EM geophysical survey	
	1991		First drilling campaign	
	1992		Second phase drilling campaign	
	1992			Disposal by SGU to Mineral Resources AB (MIRAB)
	Unknown			Disposal by MIRAB to Tricorona AB
Mansberg	1983	SGAB and NGS	Discovered by airborne EM geophysical survey	
	1991		First drilling campaign	

Source: ReedLeyton 2015

6.1 Previous/historical exploration and development

In terms of NI 43-101 Guidelines the term “historical estimate” refers to information gathered by previous owners of a property and previous estimates refer to data previously determined or disclosed by the current owners. In this context the data obtained by the SGS, NSG, SGAB and Tricorona is considered historical. However, the data acquired by Flinders is not deemed historical in that the ownership of the properties has not changed albeit that the merger with LEM resulted in a company name change. The disclosure of the Flinders/LEM exploration data is provided in Section 9.

The historical exploration undertaken for the four graphite deposits has been conducted at different levels of intensity with Kringel the most advanced, through Mattsmyra and Gropabo to Mansberg for which only limited historical information is available. The disclosure for the Kringel deposit is therefore reasonably comprehensive whilst that pertaining to the other deposits is less so.

6.1.1 Historical exploration - Kringel

The initial discovery was of a single large, mineralised boulder in Quaternary moraine float on the Kringel deposit area in the mid-1980s by the SGS. The SGS followed the discovery of the graphite mineralisation with a systematic exploration programme from 1985 onwards starting with a trenching programme, a diamond drilling campaign and both regional airborne and local ground-based electromagnetic (EM) surveys.

Electromagnetic exploration methods rely on an electrical or field response being induced within target lithologies and the response type is predicated on host rock mineralogy Geophysical EM surveys, including very low frequency electromagnetics (VLF EM), and magnetometer methods were used to delineate geophysical conductors under the shallow surface layer of recent till. Geophysics proved to be the primary anomaly definer and EM methods proved to be very efficient delineators of conductive zones containing microscopic and coarser blebs of graphite, as well as associated zones with metallic minerals. VLF Slingram methods at 3.6 kilo hertz (kHz) and 60 m coil spacing proved to be the optimal settings and the Kringel deposit was surveyed at 100 × 80 m to 200 × 80 m profile spacing.

In 1988–1989 sufficient diamond drillhole (DD) drilling was undertaken at Kringel and the other three exploitation concession areas to delineate historical mineral resources.

The DD campaign at Kringel comprised 51 DDs on 6 cross-sections in local grid coordinates extending over an approximate strike length of 600 m (see Figure 7-4). The drillhole spacing was a nominal 20 × 50 m with a 35 mm core size diameter over a total of 2,909 m (see Table 6-2). All core was half-sectioned, with one half submitted for analysis (see Section 6.1.1.1 for details). All drillhole core beneath Quaternary age moraine deposits (3 m to 20 m depth) was sampled continuously. All 1988 drilling was at -60º dip and 1989 drilling at -50° to -55° dip. With mineralisation dipping to the SW at 70° to 80°.

Twenty-five samples were submitted to the petrophysical laboratory of Sveriges Geologiska AB for density measurements using the Archimedes (water immersion) method.

No historical information is available on core recovery, but visual inspection of remnant core by Reedleyton indicates that recoveries were <95%.

Table 6-2: Drilling history of the Kringel deposit

Hole type	Year	Number of holes	Metres drilled	Concession
DD	1988	28	1,595	Kringel
DD	1989	23	1,314	Kringel

Source: 2013 PEA ReedLeyton

Note: drilling undertaken by the SGS

6.1.1.1 Historical sampling and chemical analysis - Kringel

Original assay results were compiled into a database by the SGS and subsequently acquired by Woxna Graphite in 1992.

In the case of the Kringel deposit, the half core samples were submitted to the Sveriges Geologiska AB laboratory in Luleå and all samples were analysed using the Leco thermal IR (infrared) methodology, as described in Section 11.

The historical drilling programme resulted in 374 graphite analyses and 52 sulfur assays. Selective samples were analysed by whole rock (ICP) for major and minor elements and loss on ignition (LoI).

Twenty-five samples were submitted to the petrophysical laboratory of Sveriges Geologiska AB for density measurements using the Archimedes (water immersion) method.

No information is currently available as to specific quality assurance, quality assurance (QA) control (QC) protocols used by the SGU in its analytical programme, however it is the opinion of the Qualified Persons that work completed by the SGU was routinely of a high standard and suitable for resource estimation.

6.1.1.2 Historical density determination - Kringel

As indicated in Section 6.1.1.1, 25 density measurements were obtained for the historical drilling programme.

No bulk density determinations conducted on mineralised intervals have been viewed by the Qualified Persons.

6.1.1.3 Historical petrology - Kringel

In total 107 thin sections were prepared from DD cores between 1988 and 1989. The studies mainly focused on the graphite flake size and distribution of the mineralisation.

6.1.1.4 Historical Mineral Resource estimate -Kringel

The Kringel deposit has an historical estimate of Mineral Resources dated 2002 (as provided in the Tasex Capital 2011 Technical Report and Flinders 2013 PEA), which has previously been disclosed as a having been estimated according to a foreign code in accordance with paragraph 2.4(a) of NI 43-101.

The 2002 historical estimates provided in Table 6-3 were performed by Dr L-A Claesson, a Qualified Person as defined NI 43-101. The estimate was undertaken using cross-sectional polygonal interpretations in a simple nearest neighbour, sectional approach and using a density of 2.7 grams per cubic centimetre (g/cm^3) to convert volume to tonnage. The density assumptions are supported by testwork (Claesson 1988, 1989).

The historical Mineral Resource estimate is based on unverifiable information and is not considered by ReedLeyton as NI43-101 compliant. The historical estimate should not be relied upon.

Table 6-3: Historical Mineral Resource statement - Kringel *L-A Claesson, 2002*

Resource	Classification[1]	Tonnes	Grade C	Cut-off	Year	Notes
Kringel	Measured	1.11	11.3%	7%	1988	2
Kringel	Indicated	0.22	11.3%	7%	1988	1
Total		1.33	11.3%	7%	1988	3

Source Tasex Captial 2011

Note: foreign code resources as Part 2 of Companion Policy to NI 43-101

1

6.1.1.5 Historical production - Kringel

Tricorona brought the Project into production in 1996 and ceased operation in 2001. According to information provided in Flinders PEA 2013, the plant had a rated capacity exceeding 10,000 t/a of flake graphite limited by the environmental permit condition restricting mining to 100,000 t/a of graphitic material. Historical processing achieved graphite products ranging in purity between 85% to 94% C.

The historical production data for the Woxna Mine is listed in Table 6-4 below. (SGU, Statistics of the Swedish Mining Industry).

Table 6-4: Historical production - Woxna Mine

Year	Total mined tonnage (t)	RoM (t)	Waste (t)	Beneficiated RoM (t)	Concentrate C grade %	Strip Ratio
1996				500	93	
1997				1,581	93	
1998				3,277	92	
1999	218,101	61,481	156,620	4,504	92	2.5
2000	375,000	79,000	296,000	5,602	91	3.7
2001	70,700	11,525	59,245	1,035	93	5.1
2002				0		

Source: SGU

6.1.1.6 Verification of historical data - Kringel

The exploration was conducted under the guidance of the qualified staff of the SGU. The historical information was reviewed and considered suitable for incorporation into a Mineral Resource estimate by Claesson (2002) and Coffey Mining Limited 2011. In addition, the Qualified Person visited the Woxna Mine site during the 12th and 13th of June 2012 to interrogate the historical exploration information.

Numerous drillhole collars for drillholes completed by historical exploration programmes were located around the Kringel deposit, however some drillhole collars were removed during subsequent mining. In accordance with Swedish regulations, the core has been stored at the core storage warehouse at the mine site under secure conditions since exploration was concluded on the properties.

Assay data in original laboratory sheets from the 1988 and 1989 drilling programme has not been examined by ReedLeyton. The paper records for collar, assay, survey, and geology data were digitised for the 2002 Claesson et al historical Mineral Resource estimate for Kringel and this dataset was compared to the digital data available to Woxna Graphite. Claesson (2002) concluded that the historical data is of sufficient quality and traceable provenance that it is useable as exploration data. The then supervising geologist, who is a QP under current NI 43-101 protocol, also verified the provenance of the data supplied.

A sampling and analytical verification exercise was undertaken by ReedLeyton which included some samples from the historical dataset.

6.1.2 Historical exploration- Mattsmyra

The historical exploration conducted for the Mattsmyra deposit is summarised in Table 6-1.

Four historical drilling campaigns were undertaken by the SGU as summarised below and the location of the drillhole collars is shown in Figure 6-1:

Table 6-5: Historical drilling - Mattsmyra

Concession	Drillhole type	Date	Number of drillholes	Length (m)
Mattsmyra	Diamond drillhole (DD)	1983	4	455
		1989	8	510
		1991	10	963
		1992	11	762
Total			**33**	**2,690**

Source: ReedLeyton 2015

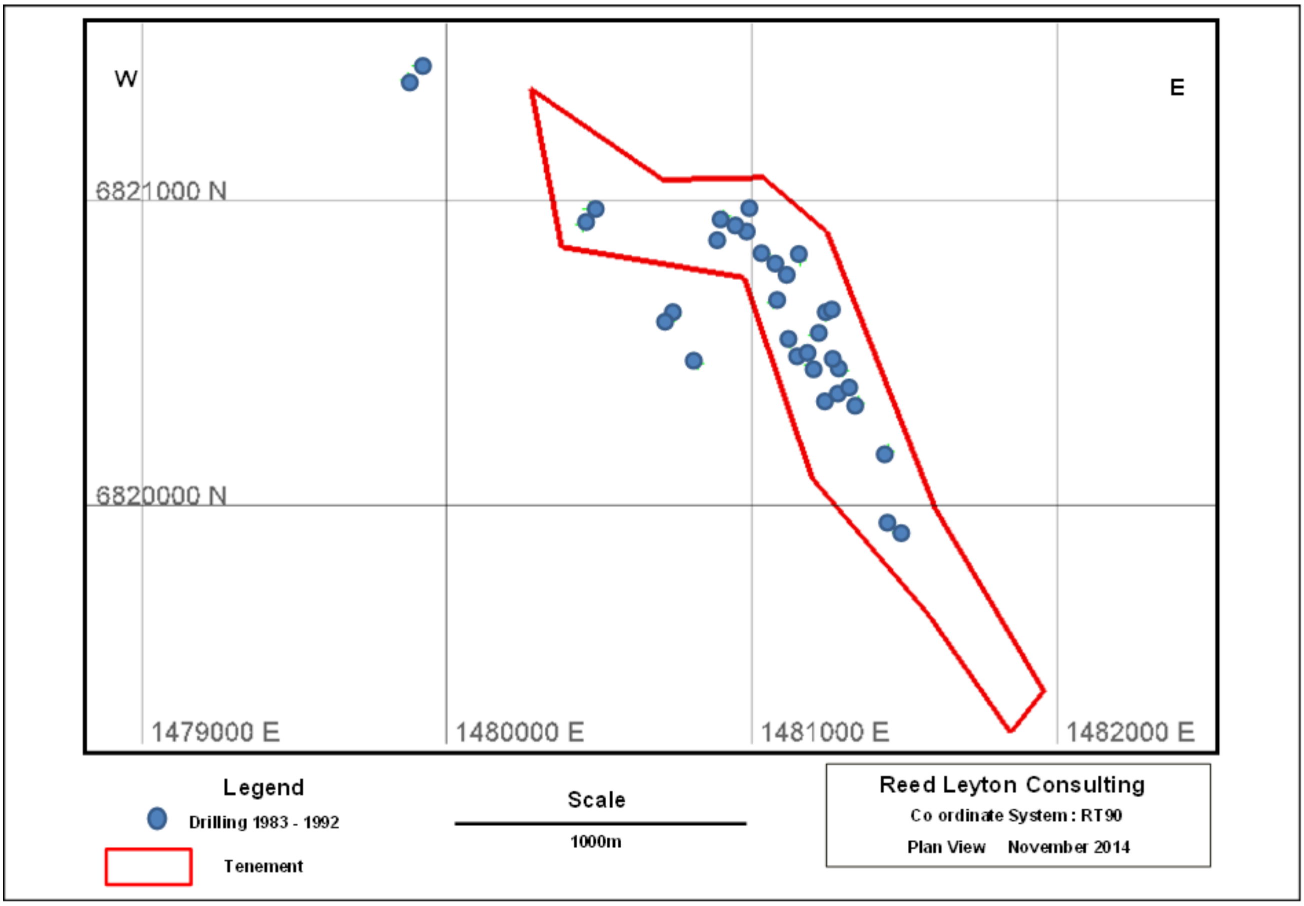


Figure 6-1: Historical drilling campaign - Mattsmyra

All remnant drill core, after sampling, is stored in boxes at the Woxna Mine site.

Sub-cropping graphite mineralisation was the first zone of mineralisation to be discovered by geophysical methods and subsequently developed. At Mattsmyra, graphite mineralisation is associated with schlieren (sheared restite remnants) associated with pegmatite intrusion into Paleoproterozoic age meta-argillites and meta-tuffisites (Claesson et al., 1988; Claesson et al., 1989a; Claesson et al., 1989b). Coarse-, medium- and fine-grained graphite is developed as coarse blebs in monomineralic zones. Parts of the mineralised zone contain wispy pyrrhotite. The combination of both graphite and pyrrhotite is the cause of the strong geophysical response to ground electromagnetic techniques applied during early exploration.

A total of 2,690 m of diamond core was drilled and comprises 33 NQ3 (46 mm) size diamond drillholes. These drillholes occur on 6 cross-sections in local grid coordinates extending over an approximate strike length of 600 m. Drillhole spacing was a nominal 20 m x 50 m. All drillhole core beneath Quaternary age moraine deposits

(3–20 m depth) was sampled continuously. All 1988 drilling was at -60° dip and 1989 drilling at -50 to -55° dip. Mineralisation dips to the SW at 70-80°.

No historical data has been presented on sample recovery, but visual inspection of remnant core indicates that sample recoveries were good (ReedLeyton 2015)

At Mattsmyra and Gropabo the position for the drillhole was originally recorded in a local co-ordinate system. This data converted into coordinates of RT90 and are assumed accurate (ReedLeyton 2021).

Table 6-6: Drillhole collar coordinates (RT 90 grid) - Mattsymara

Drillhole_ ID	Easting (m)	Northing (m)	RL (m)	Total Depth	Dip	Azimuth	Drill Type	Hole Size (mm)
83001	1480443	6820919	216.77	158.89	-50	30	DD	32
83002	1480979	6820902	230.57	116.26	-50	40	DD	32
83003	1481042	6820824	234.87	88.19	-50	40	DD	32
83004	1481283	6820453	235.04	91.25	-50	45	DD	32
89001	1480468	6820974	216.88	81.65	-50	30	DD	32
89002	1480908	6820948	228.56	67.9	-50	40	DD	32
89003	1481116	6820755	232.39	77.7	-50	45	DD	32
89004	1481448	6820177	237.86	79.85	-50	45	DD	32
89005	1481257	6820635	232.99	46.55	-50	45	DD	32
89006	1481241	6820625	232.85	50.9	-50	45	DD	32
89007	1480731	6820604	224.28	56.2	-50	45	DD	32
89008	1480749	6820624	224.83	49.2	-50	45	DD	32
90001	1480991	6820980	229.42	70.6	-60	40	DD	32
90002	1480943	6820922	229.5	148.35	-60	40	DD	32
90003	1480891	6820866	229.88	199.2	-60	40	DD	32
90004	1481082	6820794	232.45	35.75	-60	40	DD	32
90005	1481157	6820805	231.78	39.65	-60	75	DD	32
90006	1481266	6820357	237.3	150	-60	65	DD	32
90007	1481323	6820386	236.95	50.35	-60	70	DD	32
90008	1481209	6820560	233.28	42.75	-60	40	DD	32
90009	1481152	6820490	235.22	148.7	-60	45	DD	32
90010	1480822	6820466	226.35	77.65	-60	40	DD	32
92001	1479911	6821443	214.7	50.73	-50	38	DD	32
92002	1481192	6820458	235.45	126.45	-50	48	DD	32
92003	1481172	6820501	235.84	79.71	-50	56	DD	32
92004	1481127	6820535	238.14	112.8	-50	55	DD	32
92005	1481071	6820667	235.7	59.6	-50	64	DD	32
92006	1481297	6820442	236.06	51.1	-50	62	DD	32
92007	1481297	6820373	236.34	73.86	-50	61	DD	32
92008	1481348	6820335	237.1	53.04	-50	63	DD	32

Drillhole_ ID	Easting (m)	Northing (m)	RL (m)	Total Depth	Dip	Azimuth	Drill Type	Hole Size (mm)
92009	1481486	6819910	234.58	72.66	-50	54	DD	32
92010	1481447	6819942	233.44	72.1	-50	41	DD	32
92011	1479872	6821398	213.8	10	-50	35	DD	32

6.1.2.1 Historical sampling and chemical analysis - Mattsmyra

The historical drilling programme resulted in 381 graphite analyses (carbon by Leco analyser – see Section 11) and 52 sulfur assays (by Leco). Selective samples were analysed by whole rock ICP for major and minor elements and LoI (loss on ignition). Twenty-five samples were submitted to the petrophysical laboratory of SGAB for density measurements using the Archimedes (water immersion) method.

6.1.2.2 Historical Mineral Resource estimate - Mattsmyra

The Mattsmyra deposit has a historical estimate of Mineral Resources dated 2002 (as provided in the Tasex 2011 Technical Report and Flinders 2013 PEA), which has previously been disclosed as a having been estimated according to a foreign code in accordance with paragraph 2.4(a) of NI 43-101.

The estimates provided in Table 6-7 were performed by Dr L-A Claesson, a Qualified Person as defined NI 43-101. The estimate was undertaken using cross-sectional polygonal interpretations in a simple nearest neighbour, sectional approach and using a density of 2.7 g/cm^3 to convert volume to tonnage. The density assumptions are supported by testwork (Claesson 1988, 1989).

The historical Mineral Resource estimate is based on unverifiable information and is not considered by ReedLeyton as NI43-101 compliant. The historical estimate should not be relied upon.

A historical Mineral Resource estimate for the Mattsmyra deposit was undertaken by *L-A Claesson, 2002*

Table 6-7: Historical Mineral Resource estimate- Mattsmyra (1992)

Resource	Classification[2]	Tonnes	Grade C	Cut-off	Year	Notes
Mattsmyra	Indicated	2.17	8.77%	7%	1992	1
Total		2.17	8.77%	7%	1992	1

Source: Tasex Capital 2011

6.1.3 Historical exploration – Gropabo

The mineralisation at the Gropabo deposit was discovered by airborne EM. The first drilling commenced in 1991. A 2nd drilling phase was undertaken the following year. All historical exploration was completed by SGAB and NSG. In 1992 the concession passed to MIRAB as per Table 6-1.

Table 6-8: Historical drilling - Gropabo

Concession	Drillhole type	Date	Number of drillholes	Length (m)
Gropabo	Diamond drillhole (DD)	1991	17	858
		1992	21	930
Total			38	1,788

1

Drilling activity reported is from exploration conducted by previous operators, primarily SGAB. All remnant drill core, after sampling, is stored in boxes at the Woxna Mine site.

Sub-cropping graphite mineralisation was the first zone of mineralisation to be discovered by geophysical methods and subsequently developed. At Gropabo, graphite mineralisation is associated with schlieren (sheared restite remnants) associated with pegmatite intrusion into Paleoproterozoic age meta-argillites and meta-tuffisites (Claesson et al., 1988; Claesson et al., 1989a; Claesson et al., 1989b). The combination of both graphite and pyrrhotite is the cause of the strong geophysical response to ground electromagnetic techniques applied during early exploration.

A total of 1,788 m of DD core was drilled on the concession in 1991 and 1992 and comprises 38 NQ3 (46 mm) size diamond drillholes. These drillholes occur on 6 cross-sections in local grid coordinates extending over an approximate strike length of 600 m. Drill spacing is a nominal 20 x 50 m. All drillhole core beneath Quaternary age moraine deposits (3-20 m depth) was sampled continuously. All 1991 drilling was at -60º dip and 1992 drilling at -50 dip. Mineralisation dips to the SW at 70-80º.

The historic program resulted in 381 graphite analyses (carbon by Leco analyser) and 52 sulfur assays (by Leco). Selective samples were analysed by whole rock ICP for major and minor elements and LOI (loss on ignition). Twenty-five samples were submitted to the petrophysical laboratory of Sveriges Geologiska AB for density measurements using the Archimedes (water immersion) method.

No historic data has been presented on sample recovery, but visual inspection of remnant core indicates that sample recoveries <95%

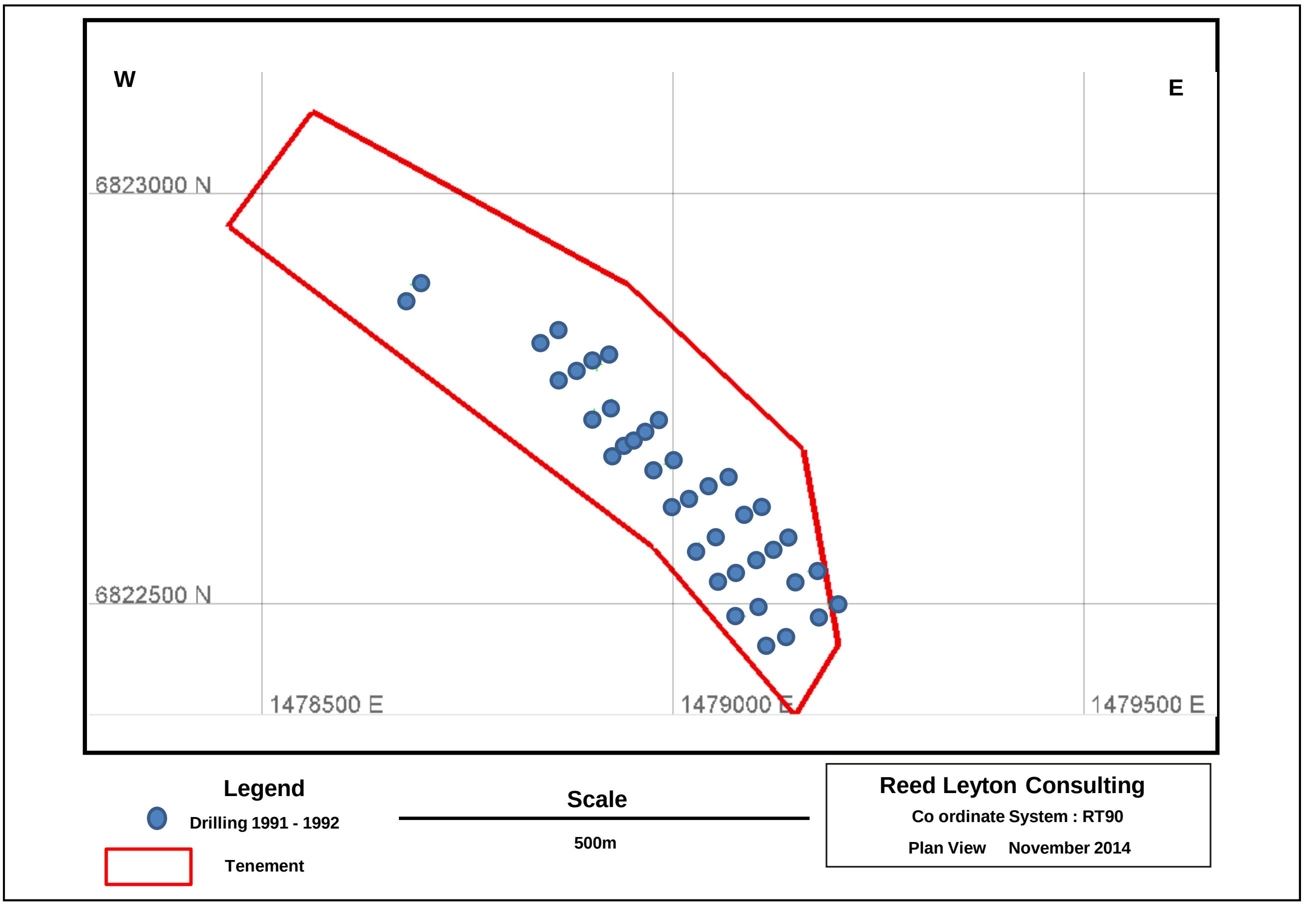


Figure 6-2: Historical drilling - Gropabo

Table 6-9: Drilling Gropabo

Hole_ID	Easting (m)	Northing (m)	RL (m)	Total Depth (m)	Dip	Azimuth	Drill Type	Hole Size (mm)
GRO91001	1479032	6822646	231.64	48.8	-60	63	DD	32
GRO91002	1478989	6822624	231.88	70.4	-60	65	DD	32
GRO91003	1478957	6822712	232.26	50.85	-60	57	DD	32
GRO91004	1478914	6822687	232.38	93.85	-60	67	DD	32
GRO91005	1478894	6822796	235.1	49.4	-60	62	DD	32
GRO91006	1478849	6822774	234.32	65.75	-60	62	DD	32
GRO91007	1479110	6822570	232.52	50	-60	12	DD	32
GRO91008	1479129	6822585	231.77	22.85	-60	61	DD	32
GRO91009	1479087	6822558	234.8	58.25	-60	50	DD	32
GRO91010	1479066	6822544	236.63	50.4	-60	53	DD	32
GRO91011	1479045	6822531	234.89	50.05	-60	53	DD	32
GRO91012	1479053	6822656	231.6	22.6	-60	58	DD	32
GRO91013	1479011	6822635	232.46	53	-60	63	DD	32
GRO91014	1478974	6822724	232.24	21.4	-60	66	DD	32
GRO91015	1478929	6822697	232.22	64.6	-60	43	DD	32
GRO91016	1478912	6822805	234.64	25	-60	64	DD	32
GRO91017	1478872	6822785	234.69	60.9	-60	62	DD	32
GRO92001	1479097	6822625	231.84	18.65	-50	59	DD	32
GRO92002	1479079	6822613	231.76	39.35	-50	56	DD	32
GRO92003	1479040	6822586	232.78	84.86	-50	59	DD	32
GRO92004	1479020	6822572	232.05	72.95	-50	62	DD	32
GRO92005	1479092	6822502	235.82	51.6	-50	56	DD	32
GRO92006	1479070	6822489	234.7	59.35	-50	63	DD	32
GRO92007	1479126	6822465	235.6	49.85	-50	62	DD	32
GRO92008	1479103	6822452	234.01	12	-50	59	DD	32
GRO92009	1478965	6822664	231.16	57.18	-50	63	DD	32
GRO92010	1478985	6822674	231.74	35.37	-50	59	DD	32
GRO92011	1478943	6822705	232.29	53.58	-50	332	DD	32
GRO92012	1478914	6822747	233.2	45.05	-50	57	DD	32
GRO92013	1478893	6822733	232.51	45.36	-50	57	DD	32
GRO92014	1478847	6822835	236.86	30	-50	61	DD	32
GRO92015	1478825	6822823	236.4	50.84	-50	61	DD	32
GRO92016	1478663	6822872	231.52	9	-50	23	DD	32
GRO92017	1478676	6822893	233.61	50.04	-50	23	DD	32
GRO92018	1479139	6822532	234.83	54.75	-50	64	DD	32
GRO92019	1479159	6822543	233.44	36.5	-50	61	DD	32

Hole_ID	Easting (m)	Northing (m)	RL (m)	Total Depth (m)	Dip	Azimuth	Drill Type	Hole Size (mm)
GRO92020	1479168	6822486	235.27	46.8	-50	47	DD	32
GRO92021	1479185	6822503	234.44	27.5	-50	56	DD	32

6.1.3.1 Historical Mineral Resource estimate - Gropabo

The Gropabo deposit has an historical estimate of Mineral Resources dated 2002 (as provided in the Tasex 2011 and Flinders 2013), which has previously been disclosed as a having been estimated according to a foreign code in accordance with paragraph 2.4(a) of NI 43-101.

The estimates provided in Table 6-8 were performed by Dr L-A Claesson, a Qualified Person as defined NI 43-101. The estimate was undertaken using cross-sectional polygonal interpretations in a simple nearest neighbour, sectional approach and using a density of 2.7 g/cm^3 to convert volume to tonnage. The density assumptions are supported by testwork (Claesson 1988, 1989).

The historical Mineral Resource estimate is based on unverifiable information and is not considered by ReedLeyton as NI43-101 compliant. The historical estimate should not be relied upon.

An historical Mineral Resource estimate for the Gropabo deposit was undertaken by *L-A Claesson, 2002*

Table 6-8: Historical Mineral Resource estimate- Gropabo (1991)

Resource	Classification[3]	Tonnes	Grade C	Cut-off	Year	Notes
Gropabo	Indicated	2.08	6.89%	7%	1991	1
Total		2.08	6.89%	7%	1991	3

Source Tasex Capital 2011

6.1.4 Historical exploration - Mansberg

The Mansberg deposit has an historical estimate of Mineral Resources dated 2002 (as provided in the Tasex 2011 Technical Report and Flinders 2013 PEA), which has previously been disclosed as a having been estimated according to a foreign code in accordance with paragraph 2.4(a) of NI 43-101.

The estimates provided in Table 6-9 were performed by Dr L-A Claesson, a Qualified Person as defined NI 43-101. The estimate was undertaken using cross-sectional polygonal interpretations in a simple nearest neighbour, sectional approach and using a density of 2.7 g/cm^3 to convert volume to tonnage. The density assumptions are supported by testwork (Claesson 1988, 1989).

The historical Mineral Resource estimate is based on unverifiable information and is not considered by ReedLeyton as NI43-101 compliant. The historical estimate should not be relied upon.

An historical Mineral Resource estimate for the Mansberg deposit was undertaken by *L-A Claesson, 2002*

Table 6-9: Historical Mineral Resource estimate- Mansberg (1991)

Resource	Classification[4]	Tonnes	Grade C	Cut-off	Year	Notes
Mansberg	Inferred	1.35	9.44%	7%	1993	1
Total		1.35	9.44%	7%	1993	1

1

1

Source Tasex Capital 2011

6.2 Conclusion

The historical data for Kringel, Mattsmyra and Gropabo was independently reviewed by Coffey Mining Limited for a Mineral Resource estimate published in 2011 (as shown in Table 6-1) and was considered appropriate for use in a resource estimate. ReedLeyton concurs with this conclusion based on its own verification of the data for the pre-2012 Kringel, Mattsmyra and Gropabo information.

Table 6-10: Summary Historical Mineral Resource estimate

Concession	Classification	Tonnes (Mt)	Grade C (%)	Cut-off Grade (C%)	Date
Kringel	Measured	0.22	11.4	7	1988
	Indicated	1.11	11.3	7	1988
Gropabo	Indicated	2.08	6.89	7	1991
Mattsmyra	Indicated	2.17	8.77	7	1992
Mansberg	Inferred	1.35	9.44	7	1993
Total		**6.93**	**8.82**		2002

Disclosed as historical resources Companion policy NI 43-101

Foreign Resource as provided for in Companion Policy NI43-101

Source Tasex Capital 2011

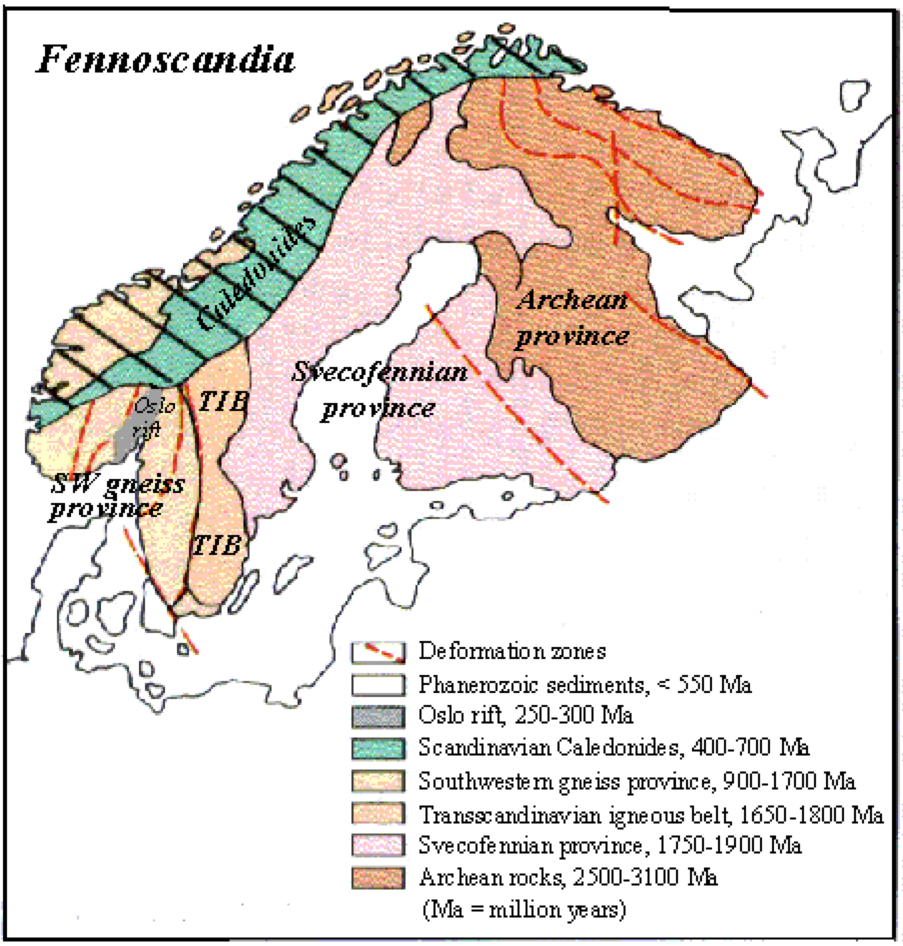
7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional geology

The geological history of Sweden is highly complex and includes at least four periods of cratonic stability with sedimentary basin development separated by at least six orogenic, mountain building events and associated magmatic activity. To date no simple unified framework for the classification of the various geological domains has been proposed and the summary below is based on tectonic events rather than lithology.

However, broadly the geology of Sweden consists of the following main components (see Figure 7-1):

- *Precambrian basement* of gneisses and greenstone belt which forms the western part of the Fennoscandanavian stable craton (>2.5 giga annum (Ga)) in the extreme north of the country. This Archaean basement was affected by the oldest identified orogenies namely the Archaean Lopian (2.8 Ga to 2.6 Ga) and Sveco-Karelian (2.0 Ga to 1.8 Ga) mountain building events;
- *younger metasedimentary and metavolcanic sequences* which are the metamorphosed remnants younger sedimentary and magmatic (2.0 Ga and 1.65 Ga) sequences deposited on top and within this ancient crust. The sedimentary units are grouped as the Svecofennian Province which occur in central Sweden. Numerous magmatic complexes also intruded into the stable craton, such as the Transcandanavian Igneous Belt (TIB) located in the west of the country. Apart from the TIB granitoids, these sequences were affected by Paleo-Mesoproterozic mountain building process known as the Gothian (1.7 Ga to 1.5 Ga); Hallandian (1.5 Ga to 1.4 Ga) and Sveconorwegian (1.1 Ga to 0.9 Ga) orogenies;
- *remnants of Scandinavian Caledonides*, a sequence of metasediments and metavolcanics deposited in the Lapetus Ocean (the predecessor of the present-day Atlantic Ocean) dated. 700 Ma to 400 Ma which occur as nappes thrust onto the Fennoscandanavian craton;
- Phanerozoic sedimentary sequences of shales, limestone and sandstone occur in southernmost Sweden and are remnants of Cambro-Silurian cover aged between 540 Ma and 420 Ma.
- *Tertiary age sediments* formed between 250 Ma and 55 Ma occur in the most southerly and southwestern parts of Sweden (Skåne).



Source: Naturhistoriska riksmuseet www.nrm.se/faktaomnaturenochrymden/geologi

Figure 7-1: Regional geology of Sweden

7.2 Local geology

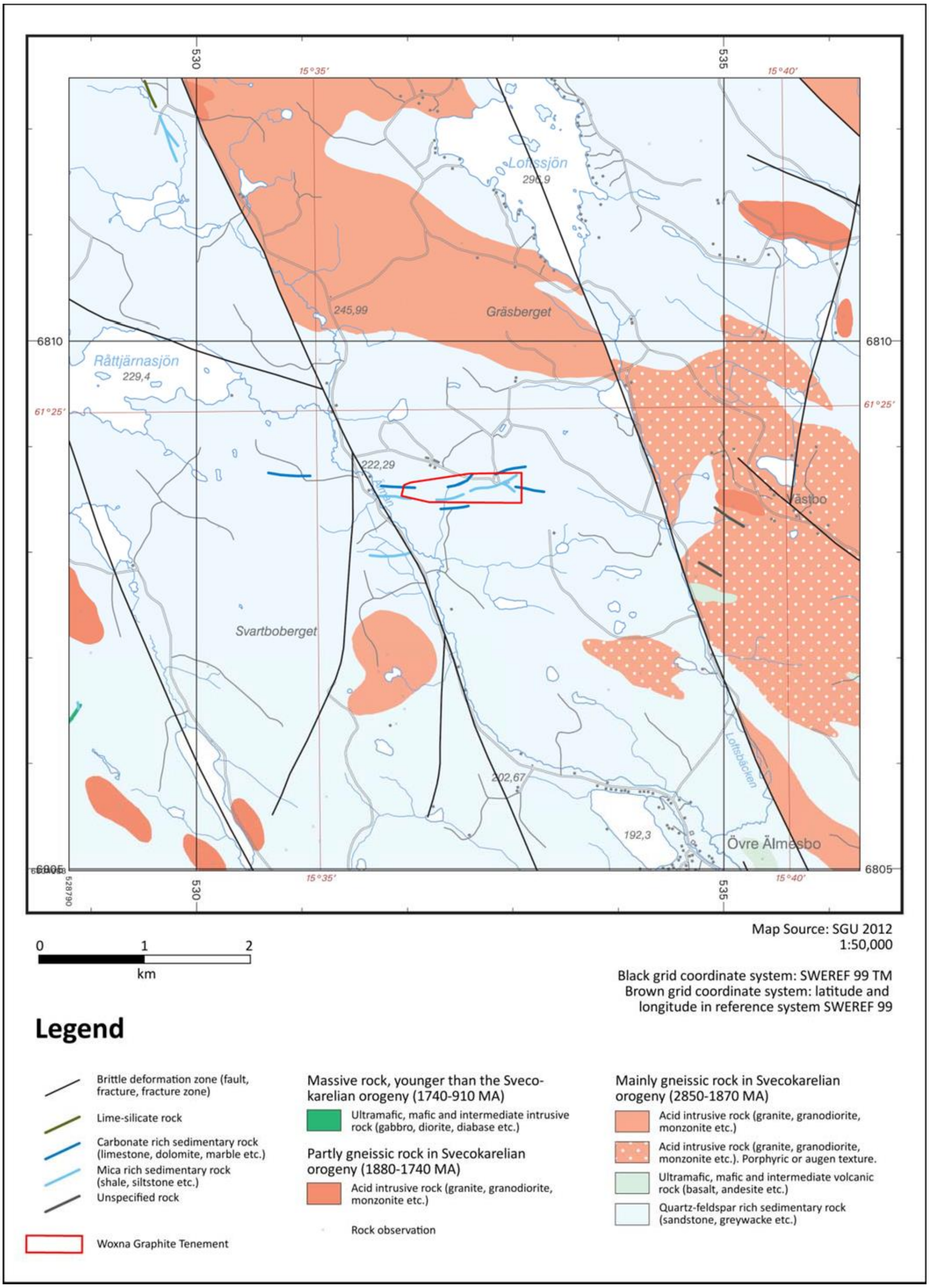
7.2.1 Local geology - Kringel

The Kringel graphite deposit is hosted by a sequence of steeply dipping metasedimentary and metavolcanic units as shown in Figure 7-2 and Figure 7-3. The host units are metamorphosed to sillimanite grade and are intruded by igneous units ranging from alkali pegmatite to granite. The local geology within the exploitation area is dominated by steeply-dipping, calcareous quartz-rich meta-tuff, with interbedded metasedimentary units and cross-cutting pegmatite.

As discussed in Section 7.3 and Section 8, trace to massive graphite mineralisation in two discrete tabular zones is developed in association with pegmatitic intrusions. The graphite is considered hydrothermal/metasomatic in nature and trace pyrrhotite is associated with the mineralised zone, in its foot wall and hanging wall. The mineral assemblage includes accessory prehnite and zoisite and the ubiquitous quartz-feldspar-chlorite-sericite metasomatic assemblage which, as indicated above, indicates a low grade of metamorphism.

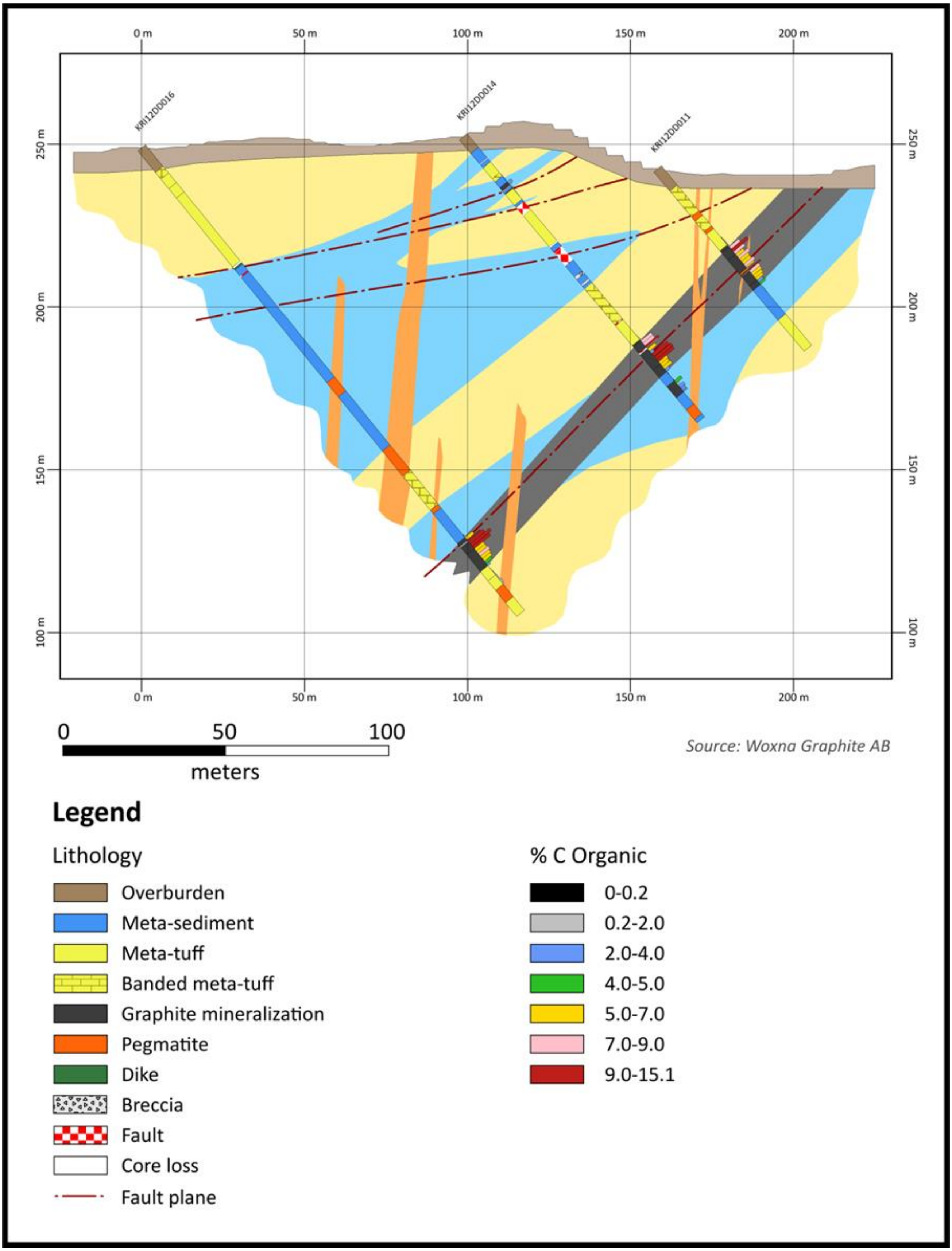
The mineralisation is tabular in shape, and occurs late in the structural history, postdating and cross-cutting any remnant tectonised and metamorphosed lithologies.

The Kringel deposit area has variable cover of 2 m to 15 m of Quaternary age glacial moraine.



Source: ReedLeyton 2013

Figure 7-2: Local geology of the Kringel deposit and environs



Source: ReedLeyton 2013

Figure 7-3: Cross section of the Kringel deposit

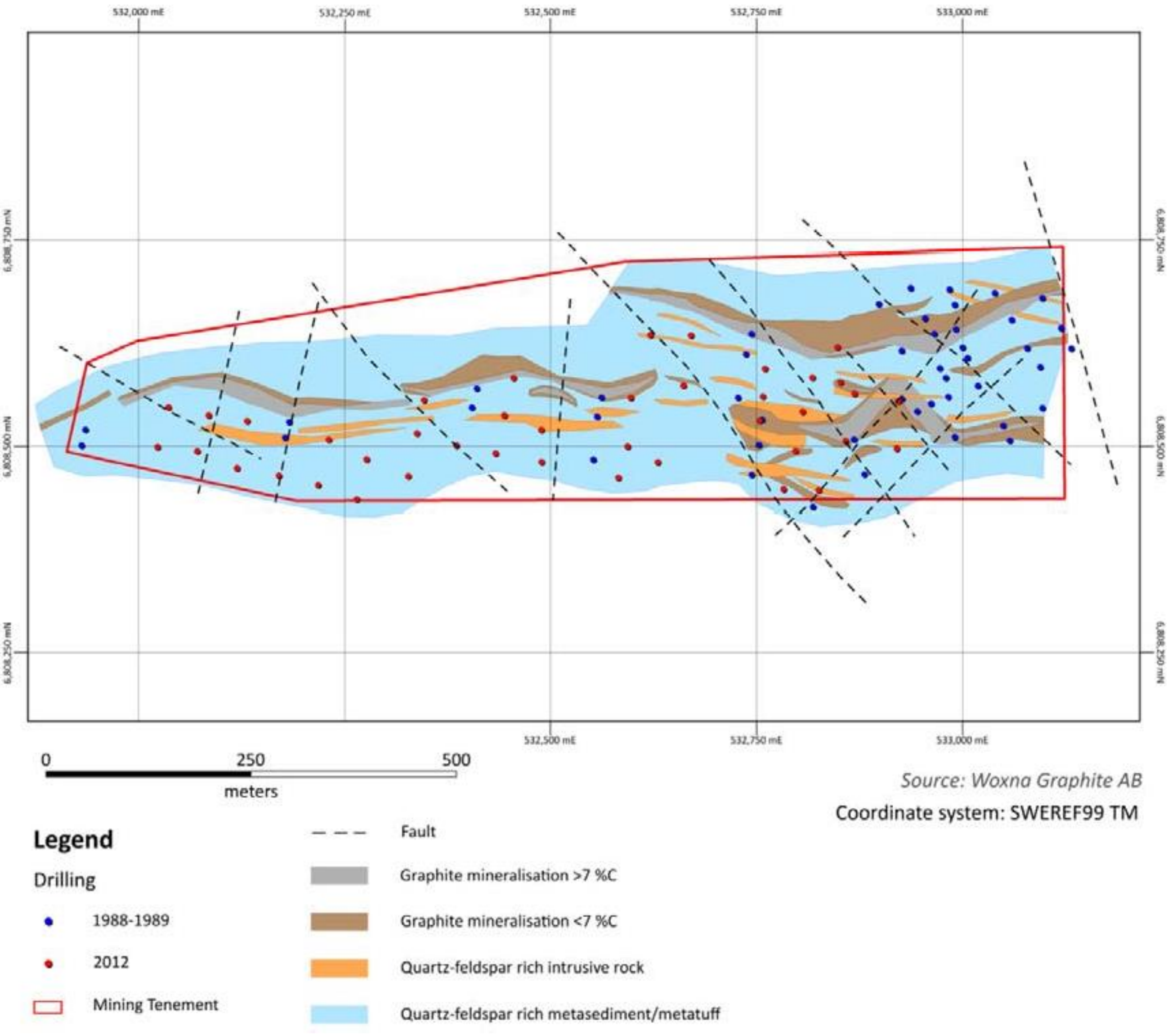


Figure 7-4: Local geology of the Kringel concession

7.2.2 Local geology - Mattsmyra

The local geology of the Mattsmyra deposit dominated by steeply to moderately dipping porphyroblastic metavolcanic and meta-argillic lithologies with common intrusive alkali pegmatites (see Figure 7-5, Figure 7-6 and Figure 7-7) . Bedrock mapping and geophysical interpretation indicate the presence of an offset off a regional-scale shear fault with dextral sense of motion.

The graphite mineralisation is broken up into several discrete domains with lower-order faulting normal to this large fault zone. Geophysical data has demonstrated continuity over 1,200 m of strike (Claesson et al.,1992). The Mattsmyra deposit seems to have higher grade metamorphism present, with prograde metamorphism to sillimanite grade and later retrograde metamorphism to chlorite grade, with chlorite, epidote, and phlogophite present in iron- and magnesium-rich lithologies.

Graphite mineralisation occurs in prehnite-bearing meta-tuffs, garnetiferous meta-argillites and pegmatitic gneiss in at least three discontinuous, stratiform graphite-pyrrhotite horizons.

Three types of mineralisation have been distinguished:

- medium- to coarse-grained, with most grains and aggregates 0.7–1.5 mm in length;
- fine-grained with pyrrhotite; most grains are <0.5 mm in length; and
- very fine-grained impregnations associated with magnetite; most grains are <0.3 mm in length.

The mineralisation has been explored to 40-50 m depth and is open at depth. The explored zone extends for about 2.5 km along strike and is open laterally. Based on unexplored geophysical conductors, the unexplored favourable horizon extends for 5 km to the southeast. Folding and faulting have cut the mineralised zone into several bodies, particularly in the north. In central and southern areas, multiple mineralised horizons occur and have been attributed to structural repetition (folding). Individual bodies of mineralisation have a thickness of 3-30 m, but, due to structural repetition, may attain true thicknesses of up to 55 m.

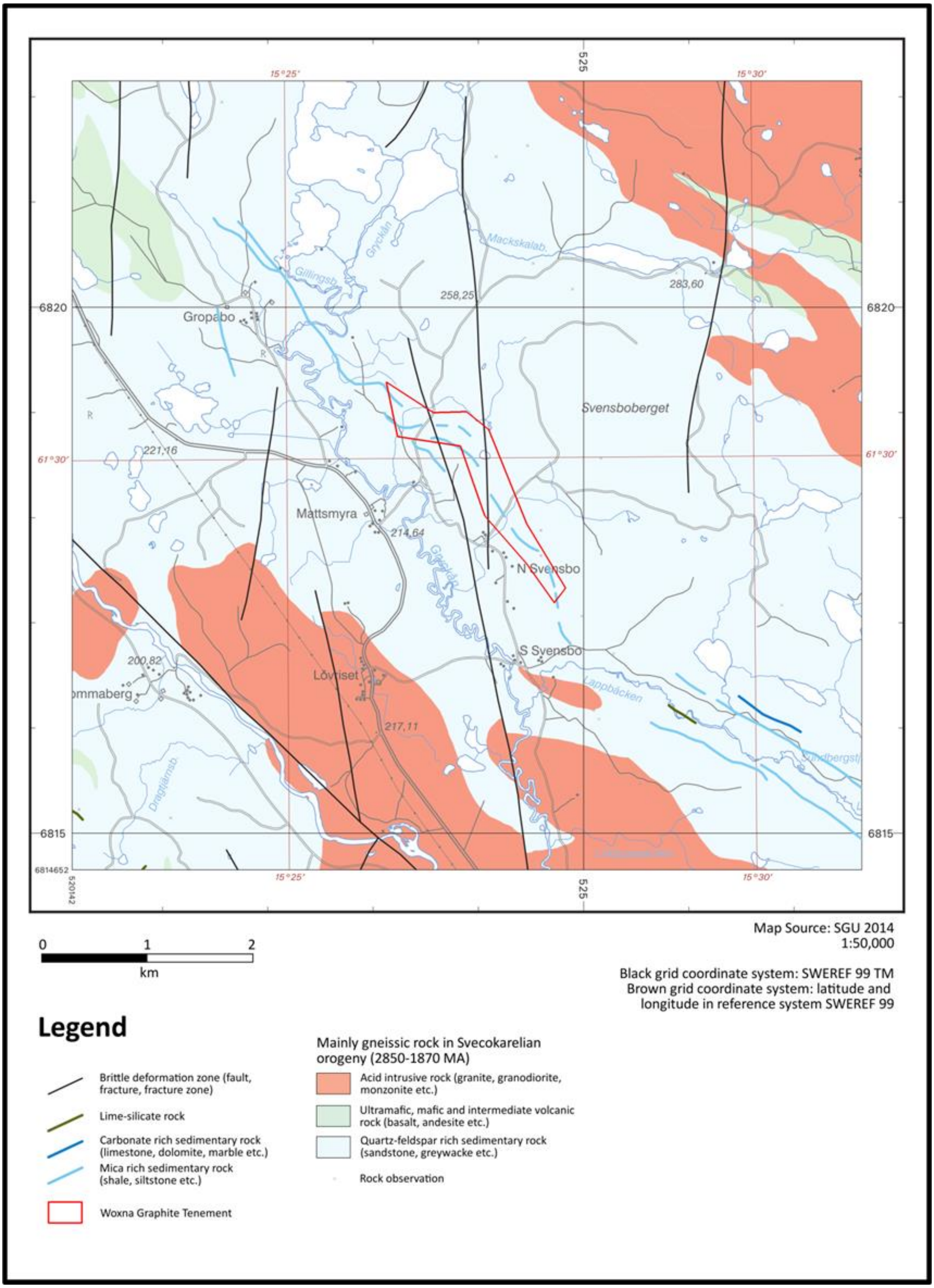
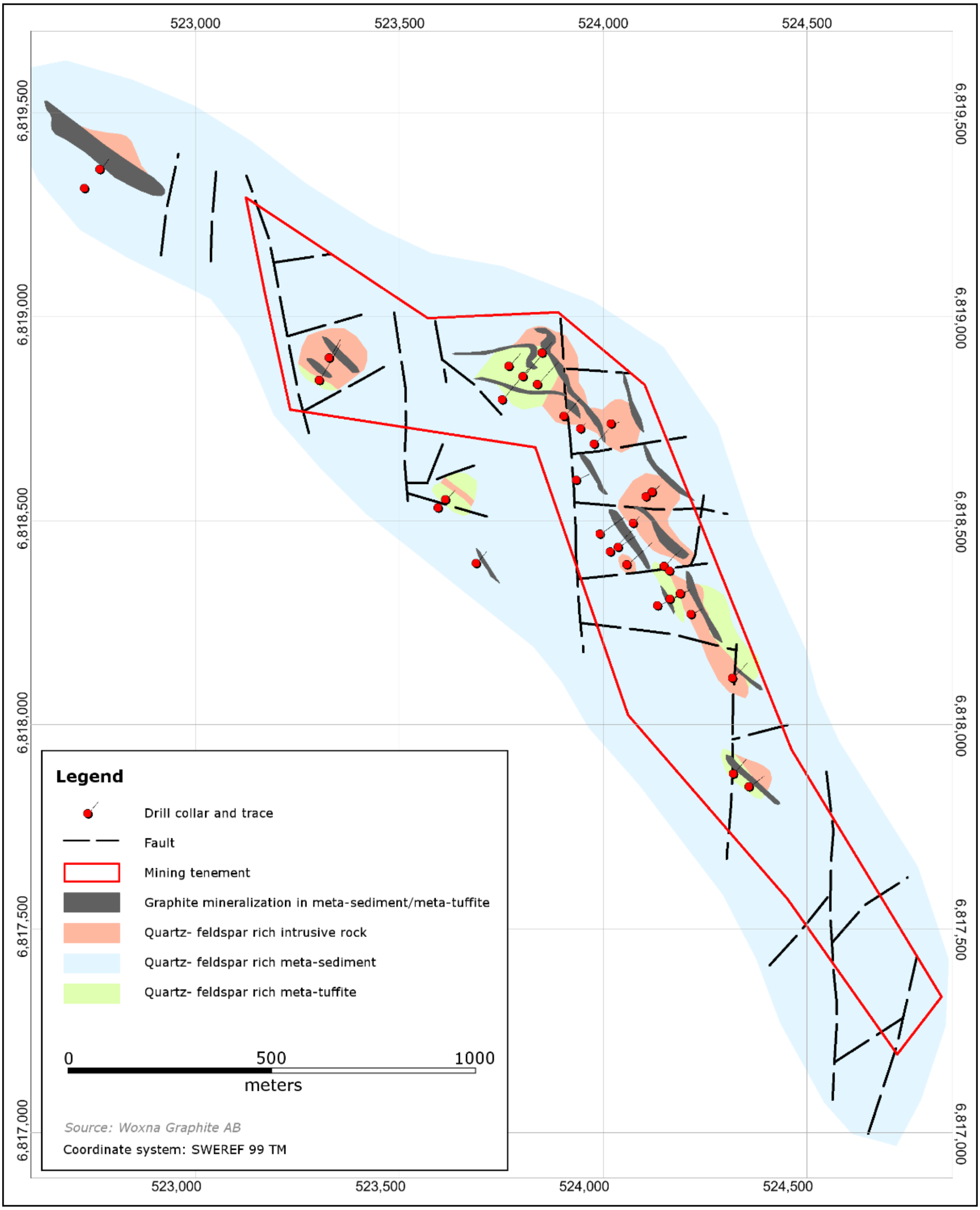
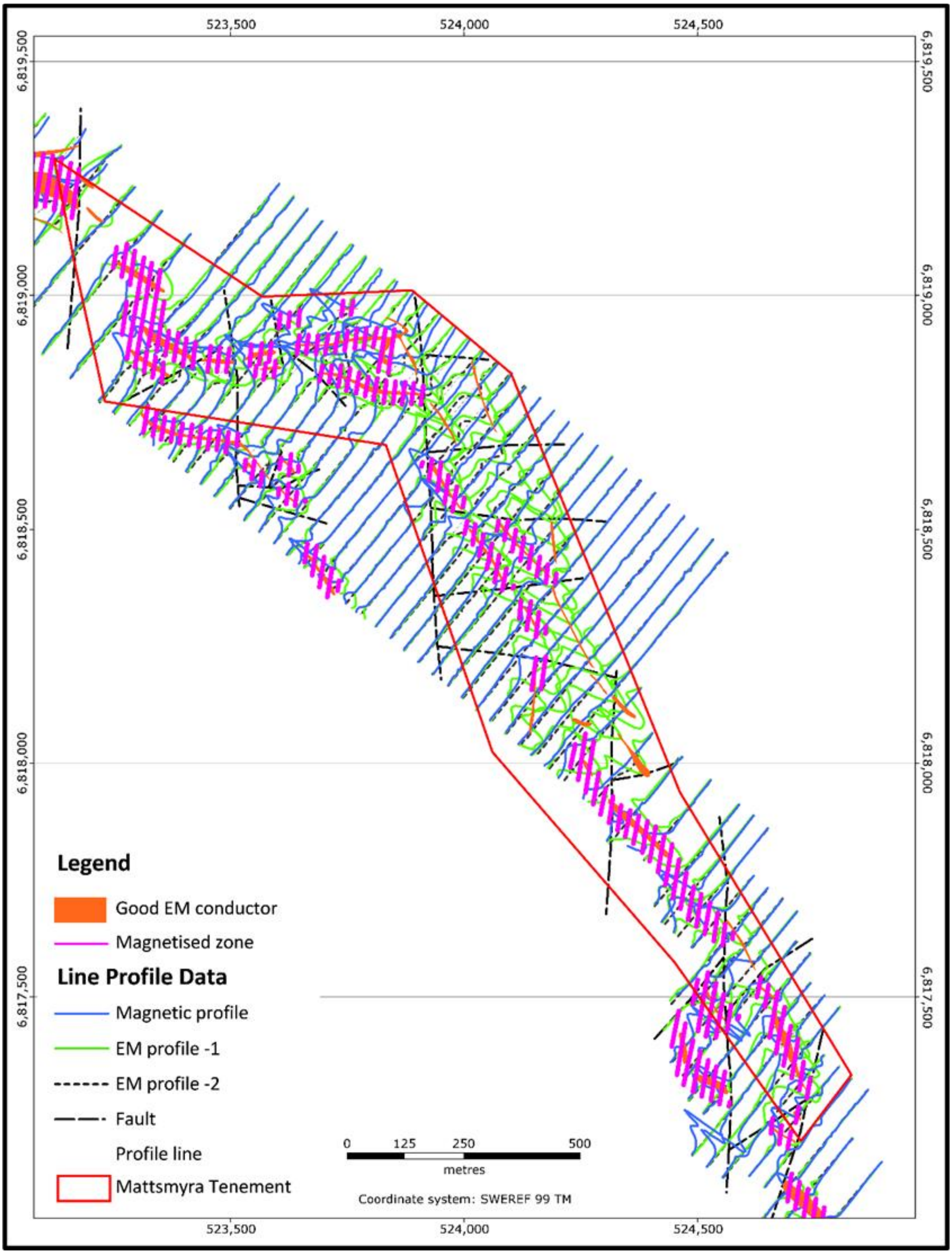


Figure 7-5: Local geology of the Mattsmyra deposit environs



Source: ReedLeyton 2013

Figure 7-6: Local geology of the Mattsmyra concession



Source: ReedLeyton 2013

Figure 7-7: Bedrock mapping and geophysical interpretation of the Mattsmyra deposit

7.2.3 Local geology - Gropabo

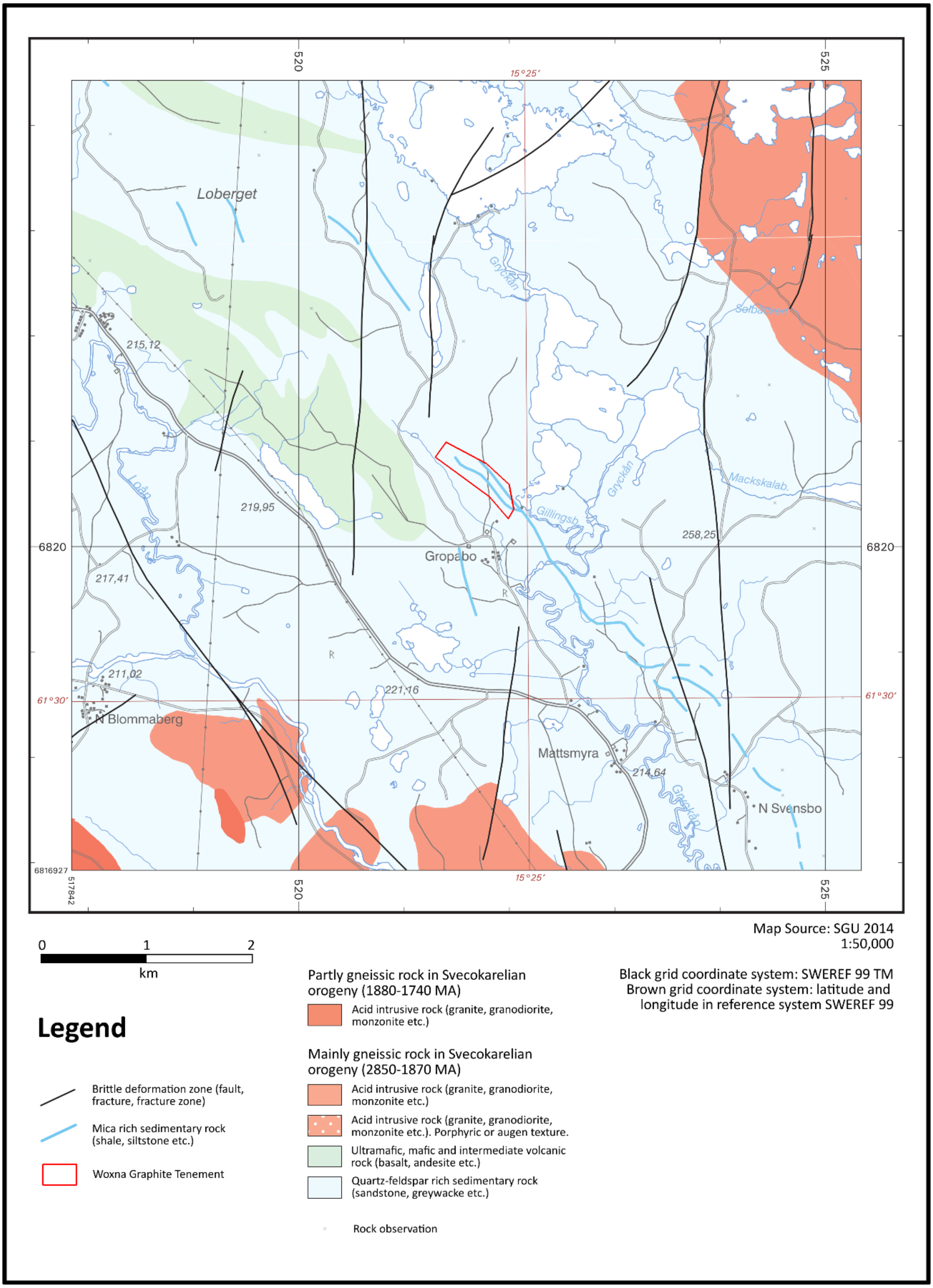
The local geology of the Gropabo is dominated by steeply to moderately dipping porphyroblastic metavolcanic and meta-argillic rocks with common intrusive alkali pegmatites (Figure 7-8 and Figure 7-9). Bedrock mapping and geophysical interpretation (Figure 7-10) indicate the presence of an offset off a regional-scale shear fault with dextral sense of motion.

The graphite mineralisation is broken up into several discrete domains with lower-order faulting normal to this large fault zone. Geophysical data have demonstrated continuity over 1,200 m of strike (Claesson et al.,1992). Gropabo seems to have higher grade metamorphism present, with prograde metamorphism to sillimanite grade and later retrograde metamorphism to chlorite grade, with chlorite, epidote, and phlogophite present in iron- and magnesium-rich lithologies.

Graphite mineralisation occurs in prehnite-bearing meta-tuffs, garnetiferous meta-argillites and pegmatitic gneiss in at least three discontinuous, stratiform graphite-pyrrhotite horizons.
Three types of mineralisation have been distinguished:

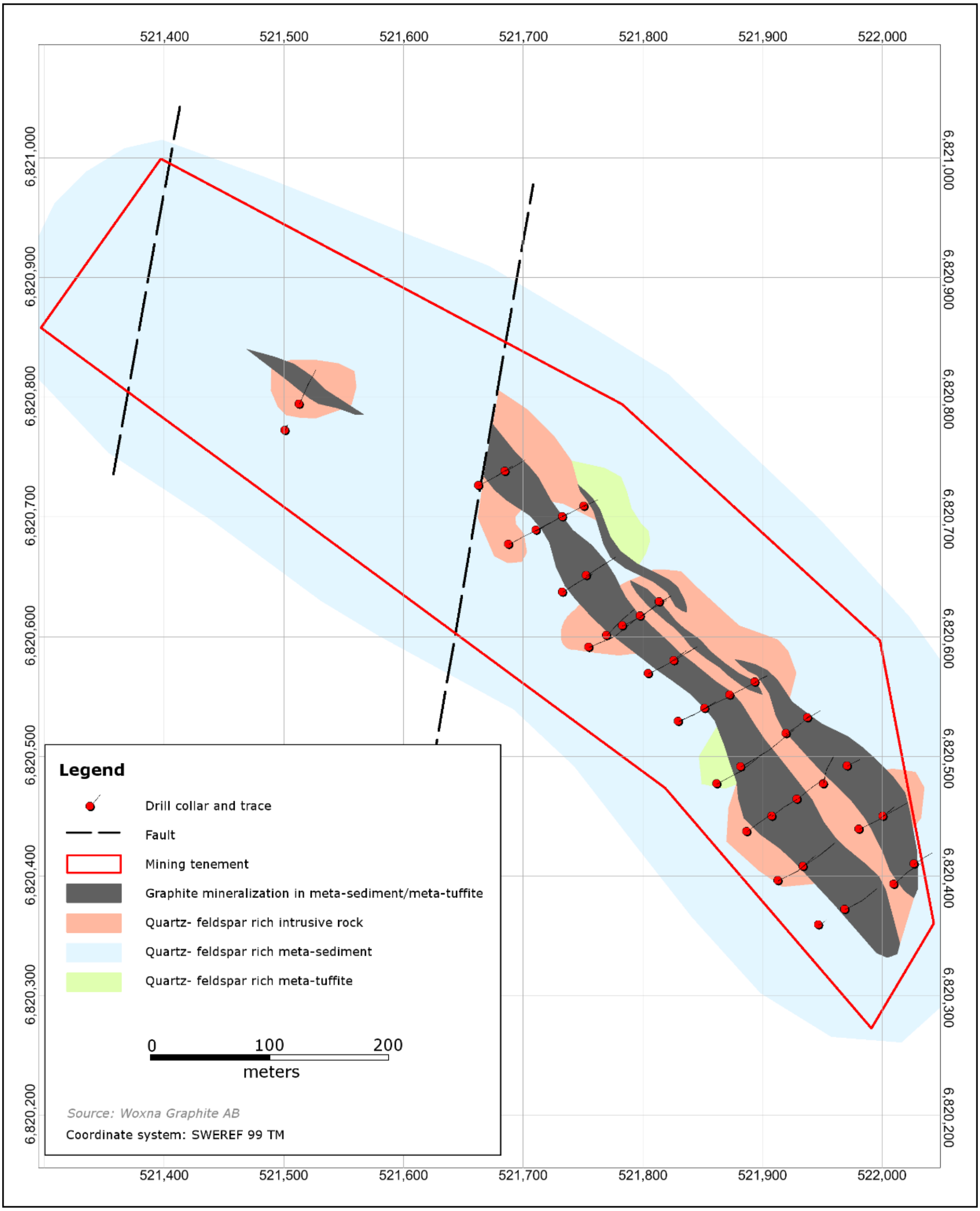
- medium- to coarse-grained, with most grains and aggregates 0.7-1.5 mm in length;
- fine-grained with pyrrhotite; most grains are <0.5 mm in length;
- very fine-grained impregnations associated with magnetite; most grains are <0.3 mm in length.

The mineralisation has been explored to 40-50 m depth and is open at depth. The explored zone extends for about 2.5 km along strike and is open laterally. Based on unexplored geophysical conductors, the unexplored favourable horizon extends for 5 km to the southeast. Folding and faulting have cut the mineralised zone into several bodies, particularly in the north. In central and southern areas, multiple mineralised horizons occur and have been attributed to structural repetition (folding). Individual bodies of mineralisation have a thickness of 3-30 m, but, due to structural repetition, may attain true thicknesses of up to 55 m.



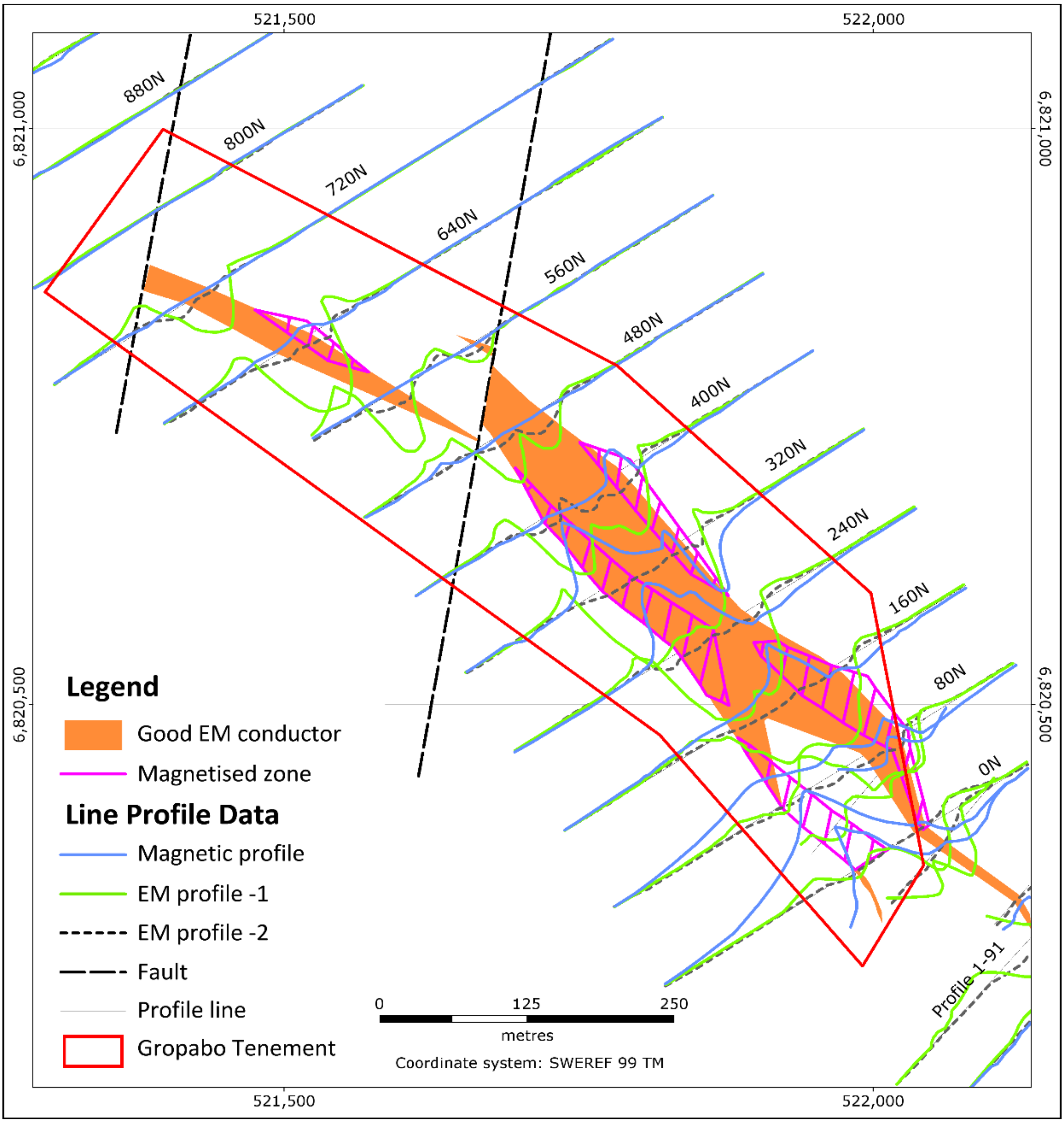
Source: ReedLeyton 2013

Figure 7-8: Local geology of the Gropabo deposit and environs



Source: ReedLeyton 2013

Figure 7-9: Local geology of the Gropabo concession



Source: ReedLeyton 2013

Figure 7-10: Bedrock mapping and geophysical interpretation - Gropabo

7.2.4 Local geology - Mansberg

The local geological setting for the Mansberg deposit is provided in the diagrams below. Exploration has been limited but serves to confirm the presence of mineralisation and to justify additional investigation.

Drilling was completed in 1993. The geology is dominated by migmatised metasedimentary and metavolcanic lithologies traversed by cross-cutting alkali pegmatite. All bedrock geology is beneath 3-12 m of Quaternary age moraine. Graphite mineralisation occurs in migmatised metasedimentary rocks containing feldspar porphyrobalsts and pegmatitic schlieren. Ill-defined banding of phlogopite-, quartz- and graphite-rich layers may be present. The mineralised zone is interpreted to dip about 60ºSW (Claesson, 1992). The graphite is relatively coarse-grained along shears, but is generally medium- to coarse-grained flake (typically 0.3-0.6 mm in length, but up to 2.5 mm) and intergrown with silicates including prehnite; it may be associated with variable amounts of magnetite, pyrrhotite, pyrite, chalcopyrite and galena.

The mineralised zone has been drilled along a geophysically conductive zone on two sections 200 m apart and is open to the southeast and, possibly, the northwest. Geophysical interpretation suggests that this conductor is about 50 m wide. The best drill intersection is 55 m wide, possibly due to structural repetition. Mineralisation is open at depth.

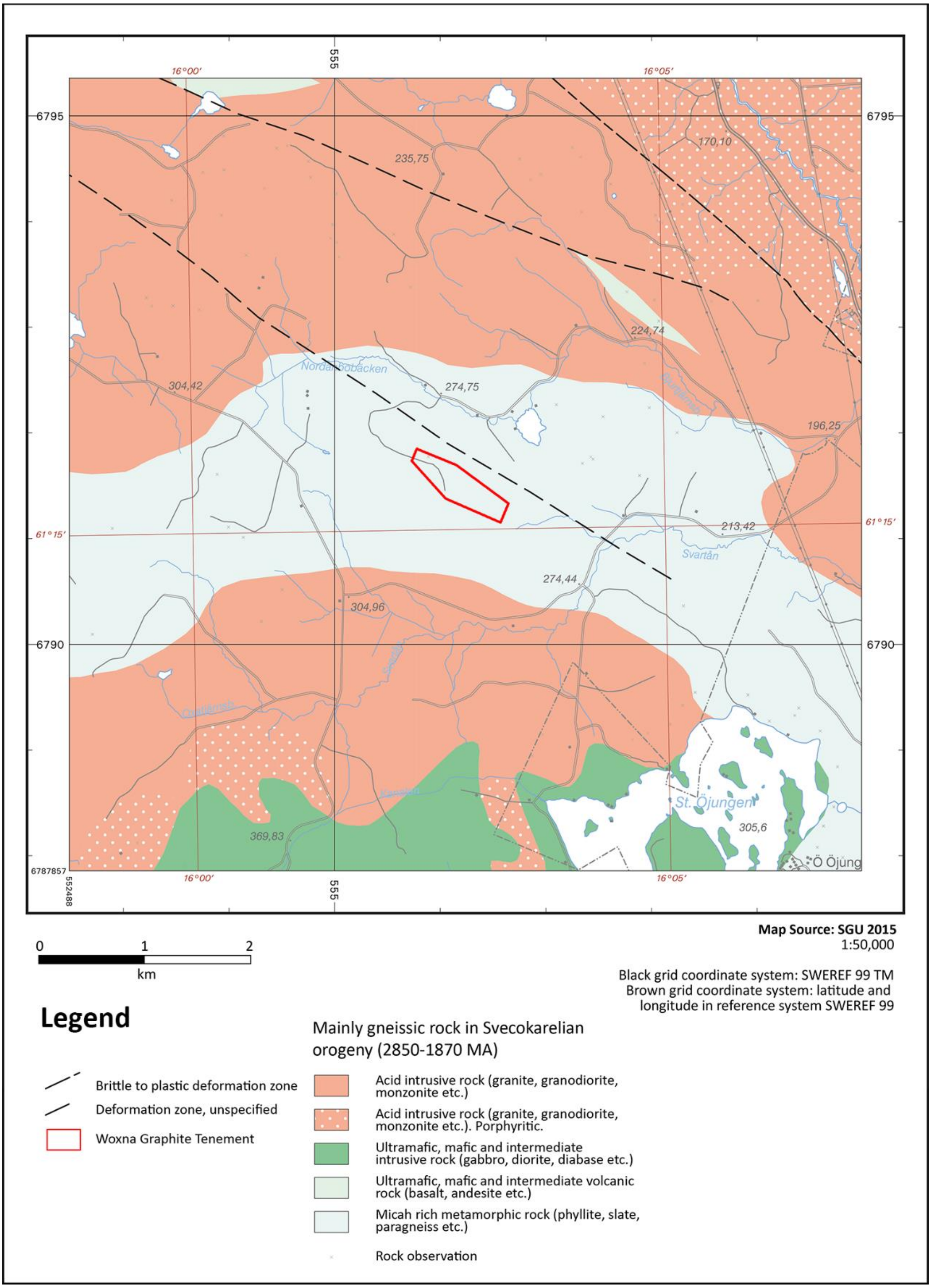


Figure 7-11: Local geology of the Mansberg deposit and environs

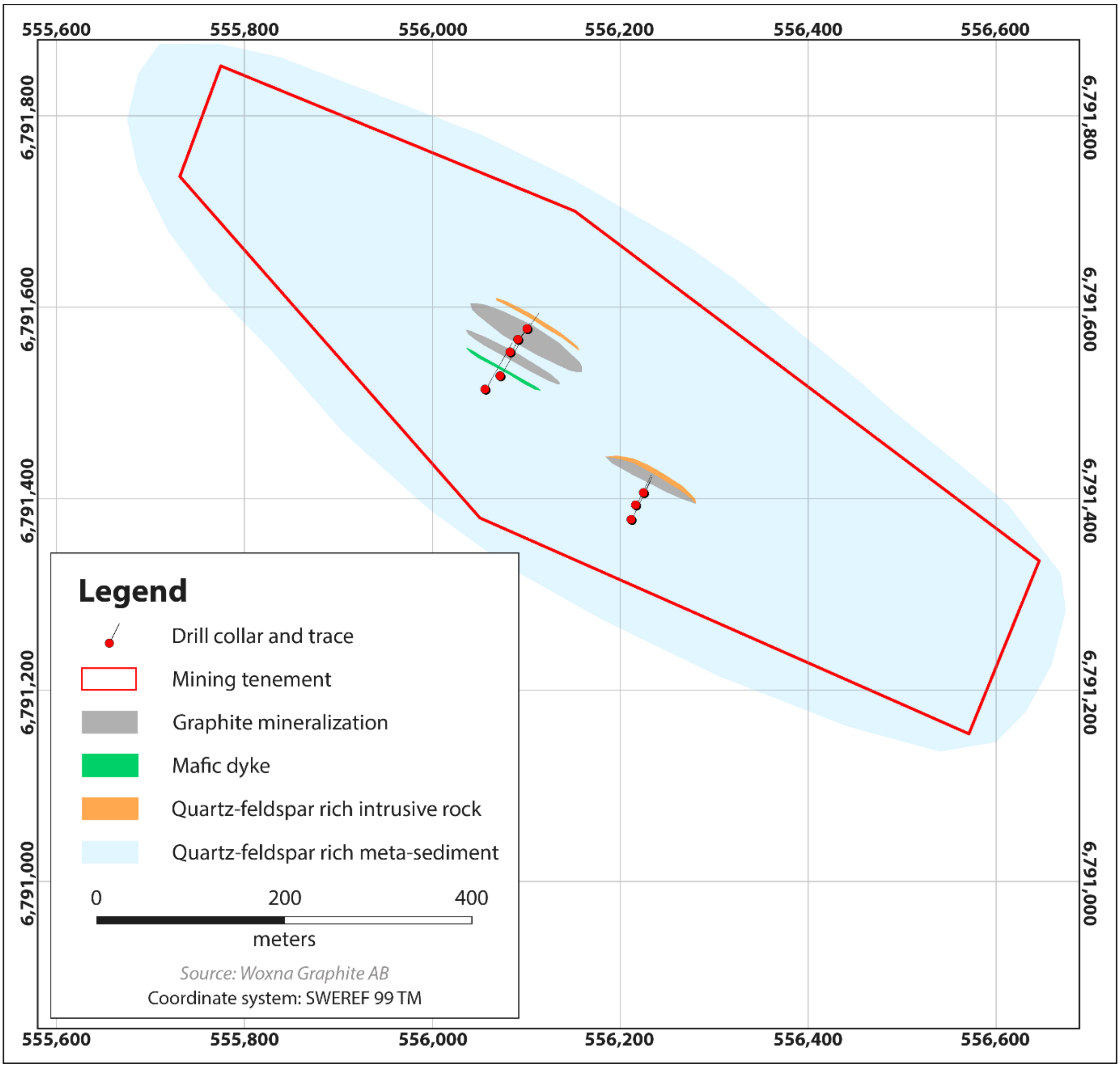


Figure 7-10: Local Geology of the Månsberg concession

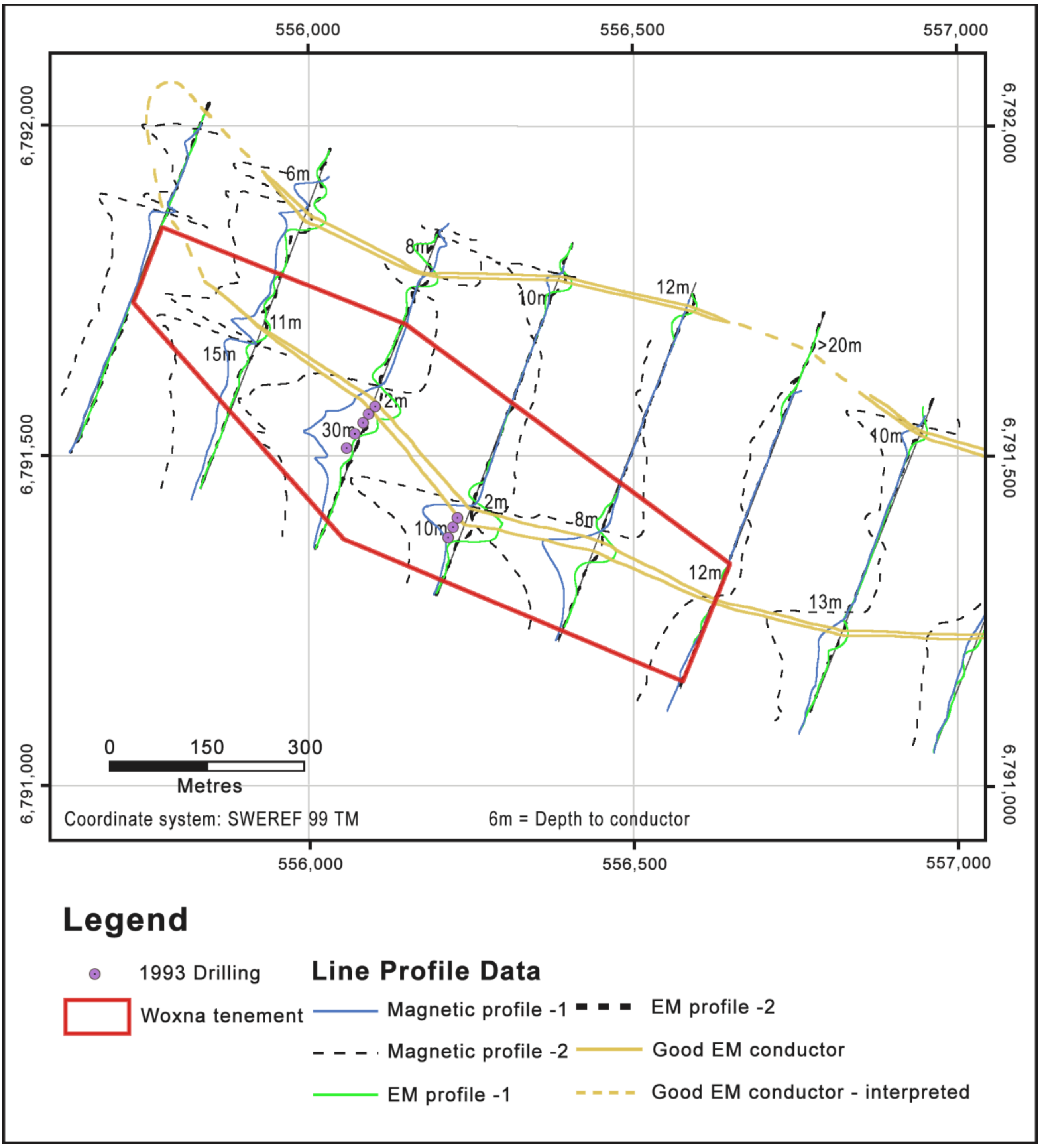


Figure 7-11: Bedrock mapping and geophysical interpretation of the Månsberg Deposit

7.3 Mineralisation

Graphite is a naturally occurring allotrope of carbon produced by the metamorphism of organic material originally deposited as sediment or mixed with sediment. As organic material is metamorphosed, hydrogen and oxygen are driven off as water, leaving the carbon behind to form graphite.

Graphite is an opaque mineral with six-sided form and crystallises in the hexagonal system with rhombohedral symmetry although well-formed crystals of graphite are quite rare in nature, and most graphite occurs in its massive form.

On an atomic level, graphite has a sheet-like structure within which carbon atoms are arranged in sheets and held together by strong, covalent chemical bonds. These carbon sheets are only weakly bonded together with those above and below by surface attractions (Van der Waals forces). It has a perfect basal cleavage and thus presents as flat flakes (>70 µm) or as a finer-grained amorphous, microcrystalline type.

The following types of graphite are found in nature:-

- *Flake graphite:- crystalline small flakes of graphite* which occurs as isolated, flat, plate-like particles;
- *amorphous graphite*:- very fine flake graphite is sometimes called amorphous;
- *lump graphite (or vein graphite)*:- occurs in fissure veins or fractures and appears as massive platy intergrowths of fibrous or acicular crystalline aggregates, and is probably hydrothermal in origin;

Graphite occurs mainly in five rock associations (Taylor, 2006) and these are:

- amorphous deposits formed by the thermal metamorphism of coal or carbon-rich sedimentary rocks;
- disseminated in marble - metamorphosed dolomite or a calcareous protolith;
- veins filling fractures fissures and cavities in country rock;
- disseminated in metamorphosed silica-rich metasedimentary rocks such as quartzites;
- contact metasomatic or hydrothermal deposits in metamorphosed calcareous sedimentary or volcaniclastic protoliths.

7.3.1 Mineralisation - Kringel

Generally, graphite is developed as an accessory mineral as laminated aggregates dispersed through schistose and siliceous metamorphic rocks (see Figure 8-1 and Figure 8-2)

The genesis of graphite mineralisation identified and historically exploited at the Project area is predominantly considered hydrothermal and/or metasomatic. The mineralisation is associated with pegmatite intrusions that are interpreted to be the heat and metasomatising fluid source during contact metamorphism of the Paleo-Proterozoic age host meta-argillites and meta-tuffites (Claesson et al., 1988; Claesson et al., 1989a; Claesson et al., 1989b).

The nature of the graphite mineralisation at Kringel is summarised below:

- the Kringel mineralisation was intersected on all the drilling sections suggestive of continuous mineralisation over the concession area;
- the mineralisation is tabular and conformable with steeply dipping host metasediments and metavolcanics;
- the mineralisation is known to extend to at least a depth of 150 m below the surface;
- the mineralisation strikes eastwest (E-W), and dip varies between 60 ^{0}and 80^{0} to the south;
- grade distribution varies both laterally and vertically;
- based on grade distribution six main higher grade Type A zones have been identified with a cut-off grade of 7% Cg (see Section 14);
- outer, lower grade Type B domains (<7% Cg cut-off grade) were identified within which 11 small mineralised envelopes/bodies exist;
- faulting of the orebodies is apparent (Figure 7-3);
- the mineralised envelopes/bodies vary in width 5 m to 15 m (averaging 10 m);
- coarse, medium and fine-grained graphite is developed as blebs in monomineralic zones (Figure 8-1, Figure 8-2). Parts of the mineralised zone contain wispy pyrrhotite (FeS_2). The combination of both graphite and pyrrhotite is the cause of the strong geophysical response to ground electromagnetic techniques applied during early exploration;

7.3.2 Mineralisation - Mattsmyra

At Mattsmyra, Graphite mineralisation occurs in prehnite-bearing meta-tuffs, garnetiferous meta-argillites and pegmatitic gneiss in at least three discontinuous, stratiform graphite-pyrrhotite horizons. Three types of mineralisation have been distinguished:

- Medium- to coarse-grained, with most grains and aggregates 0.7–1.5 mm in length;
- Fine-grained with pyrrhotite; most grains are <0.5 mm in length;
- Very fine-grained impregnations associated with magnetite; most grains are <0.3 mm in length.

7.3.3 Mineralisation – Gropabo

At Gropabo, Graphite mineralisation is present in two discrete zones and is developed over 480 m of strike length and varies up to 100 m in width. The graphitic horizons are separated by argillic metasedimentary units and pegmatite intrusions. The grade of the mineralised horizons is strongly dependent on the degree of pegmatite intrusion. Some of the pegmatites contain graphite, but not in economic quantities.

7.3.4 Mineralisation - Mansberg

At Mansberg, Graphite mineralisation occurs in migmatised metasedimentary rocks containing feldspar porphyrobalsts and pegmatitic schlieren. Ill-defined banding of phlogopite-, quartz- and graphite-rich layers may be present. The mineralised zone is interpreted to dip about 60ºSW (Claesson, 1992). The graphite is relatively coarse-grained along shears, but is generally medium- to coarse-grained flake (typically 0.3-0.6 mm in length, but up to 2.5 mm) and intergrown with silicates including prehnite; it may be associated with variable amounts of magnetite, pyrrhotite, pyrite, chalcopyrite and galena.

7.3.5 Conclusions

It is the opinion of the Reedleyton that the nature and genesis of the mineralisation at all 4 deposits is adequately understood and that the exploration programmes conducted were appropriate to the style of the deposit (with the addition of the geophysical data) and the genetic geological model. The exploration programme has proved successful and can be adapted to other graphite deposits in the region.

8 DEPOSIT TYPES

As discussed in Section 7.3 the mineralisation at the four concession sites comprises of metasomatically/hydrothermally formed graphite in association with prominent pegmatitic intrusions into steeply-dipping, calcareous quartz-rich meta-tuff, with interbedded metasedimentary material. Essentially the morphology of the mineralisation conforms to that of the host units resulting in tabular orebodies in the vicinity of cross-cutting pegmatites as shown in Figure 7-3, Figure 7-6, Figure 7-9 and Figure 7-11.

The overburden is composed of Quarternary aged glacial moraine ranging in thickness from 0.5 m to 20 m over Kringel area with an average thickness of 3.5 m. The moraine comprises micrite, chert and limestone.

The host material comprises variously highly strained sedimentary and volcaniclastic proto-lithologies, which have undergone brittle fracturing (see Figure 7-3). This host material forms the waste rock in the mining plan and comprises calcareous quartz-rich meta sediment. In some areas the graphite mineralisation is almost cropping out on surface.

Figure 8-1: Graphite mineralisation of the Kringel deposit

Figure 8-2: Graphite mineralisation of the Kringel deposit

Figure 8-3: Graphite mineralisation of drillhole 90006 - Mattsmyra

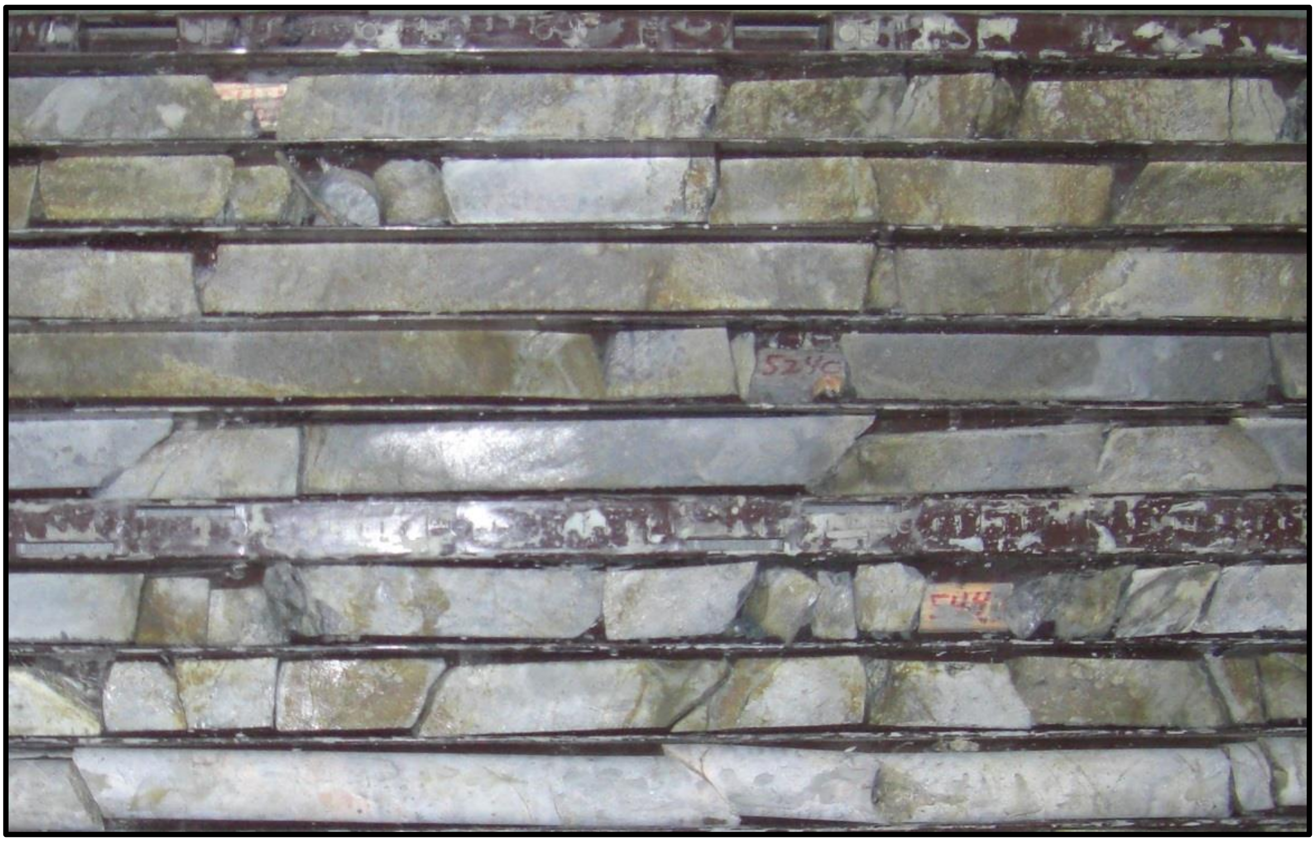


Figure 8-4: Graphite mineralisation of drillhole 92004 - Mattsmyra

Figure 8-5: Graphite mineralisation of drillhole 91013 - Gropabo deposit

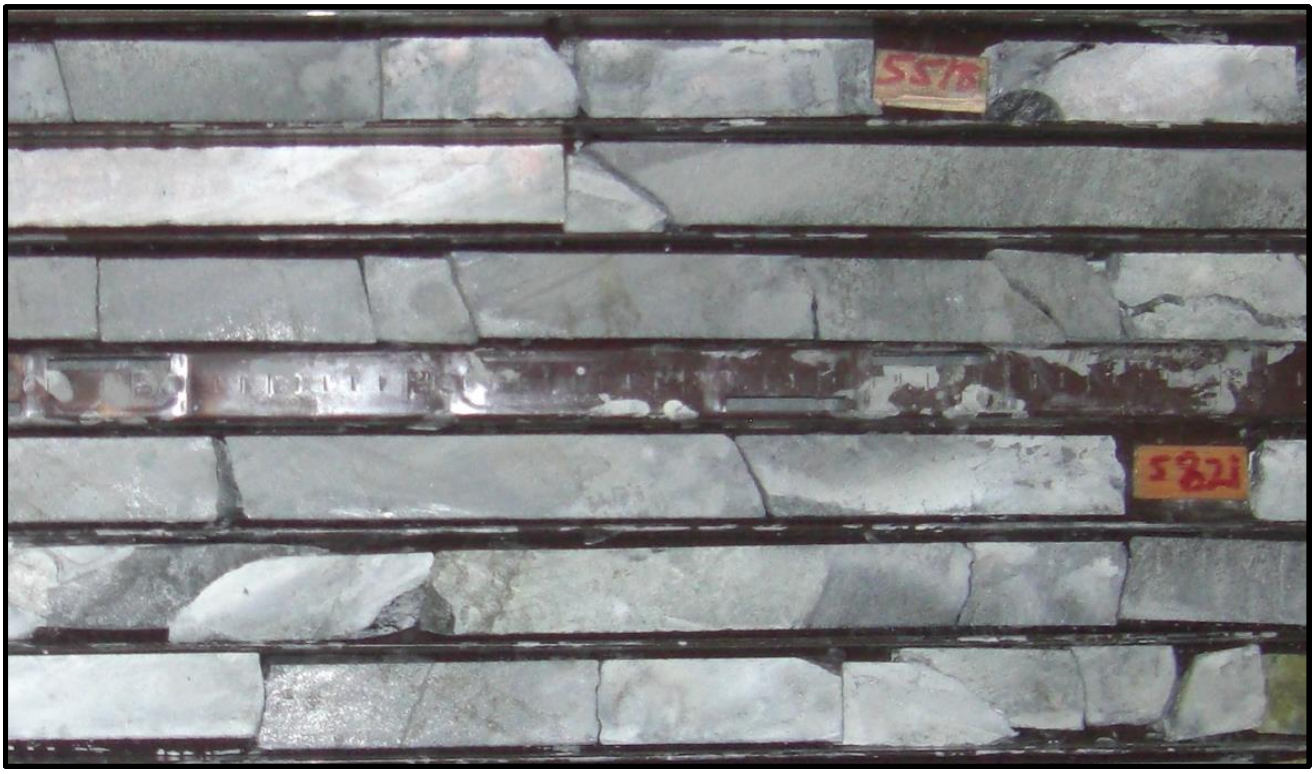


Figure 8-6: Graphite Mineralisation of Drill hole 92003, Gropabo deposit

9 EXPLORATION

The historical exploration conducted by the SGU is discussed in the 2013 and 2015 Technical Reports quoted in Section 1. This exploration was conducted by historical owners and in terms of NI43-101 guidelines has been classified as historical and has therefore been reported in Section 6. The exploration disclosure in Section 6 includes geophysical surveys as well as drilling and sampling programmes.

For the purposes of this 2021 PEA the exploration undertaken by Flinders is not considered historical and is reported below. The exploration conducted comprised drilling campaigns on the Kringel deposit with associated sampling and analytical programmes as reported in detail in Section10 and Section 11.

No additional exploration was conducted for the Mattsmyra, Gropabo and Mansberg deposits but verification of the historical drilling campaigns and a resampling exercise was undertaken in 17th and 18th of June 2014 by ReedLeyton. The results of these exercises are reported below.

The 2012 drilling campaign was conducted under the guidance of the qualified staff and contractors of Flinders. In accordance with Swedish regulations, the drillhole core has been stored at the core storage warehouse at the Woxna Mine site under secure conditions since exploration was concluded on the properties.

ReedLeyton visited the Woxna Mine site during the 12th and 13th of June 2012 to examine company records and conduct field checks of the Flinders exploration programmes. Numerous drillhole logs were studied, and drillhole core from twelve drillholes (representative of 13% of the dataset) were examined in their entirety. A sampling and analysis verification exercise was undertaken on 59 samples of drillhole core (or 4% of the sample data) matching assayed intervals from the 2012 campaign.).

10 Drilling

Drilling activity from historical exploration conducted by previous operators, primarily SGU, has been provided in Section 6.1.

The drilling conducted by current owners LEM (previously Flinders) is disclosed below.

All remnant drillhole core, after sampling, is securely stored in wooden core boxes at the Woxna Mine site.

10.1 Drilling campaign - Kringel

Drilling conducted by Flinders (now LEM) was carried out in 2012 over the area shown in Figure 10-1 The campaign was designed to:

- to infill the historical SGS 1988-1989 campaign, and
- to the extend the historical drilling grid on a nominal 50 m x 50 m spacing.

During the spring and summer of 2012 Flinders completed 3,673 m of drilling over 41 diamond drillholes at the Kringel deposit (see Figure 10-1). Sixteen north – south orientated profiles were drilled across the Kringel deposit at 50 m spacing. Three of these profiles were infill type drilling on profiles existing from the historical drilling programme. Of the 3,673 m of drilling, 270 m was overburden drilling, the remaining 3,403 m being core. Drilling was completed by independent contractor Ludvika Borrteknik AB using a GM100 rig and BGM size rods producing a core with diameter 42 mm.

Originally it was planned to drill 36 drillholes for a total of 3,000 m, but as initial results were favourable it was decided to extend the programme. Five additional drillholes were drilled resulting in the 41-holes (Table 9-1). The location of the 2012 drillholes is shown as red dots on Figure 10-1.

Table 10-1: Flinders drilling of the Kringel deposit project

Hole Type	Year	Drillhole Number	Meters (m)	Concession
DD	2012	41	3,673	Kringel

Source: ReedLeyton 2012

Most drillholes dip 50° and drillhole lengths are typically 100 m, resulting in a vertical depth test of approximately 80 m. Shorter drillholes were drilled where the graphite was intersected close to surface. Drillhole numbering starts with the abbreviation KRI followed by the year (12) and ends with a continuous drillhole no from KRI12001 to KRI12041.

Twelve drillholes have been deviation surveyed to date. The start azimuth was measured using the Reflex Azimuth Pointing System (APS), which is a GPS based compass that measures true north azimuth and is not affected by magnetic disturbance. Any uncertainty in drillhole trend caused by the lack of survey is considered by the Qualified Person to be of minor significance at the spacing of the drillholes and relatively short drillhole length in relation to a scale of the ore body.

Drillholes were laid out with the aid of a GPS, with spacing confirmed by tape and compass. At the end of the drilling programme, an independent Swedish company conducted a differential global positioning survey (DGPS) during which the location of 40 drillholes were measured with an accuracy of between 0.01 m and 0.2 m (XYZ).

Additional DGPS surveying was conducted by Tyréns in January 2013. Where possible, Flinders has surveyed all drillhole collars by DGPS. The exception is the drillhole collars now located within the boundaries of the open-pit which were removed during mining. Position for these drillholes has been calculated by converting the historical local grid into coordinates of SWEREF99 and are assumed accurate.

The rock competence in drillholes viewed by Qualified Person ReedLeyton was in general very good. Fractured rock was encountered only locally.

All DD drilled at Kringel were sampled with an average of 1 m intervals and sampled continuously based on the logged geology. Core recovery from Flinders drilling campaign was generally >95%.

Combined with the historical data, the total number of drillholes is 92 for a total length of 6,581 m. Data and plans relating to the collar locations, drillhole collar orientations and drillhole surveys for both the historical SGU data and the 2012 Flinders campaign were examined by the Qualified Person and most drillhole collars were located in the field.

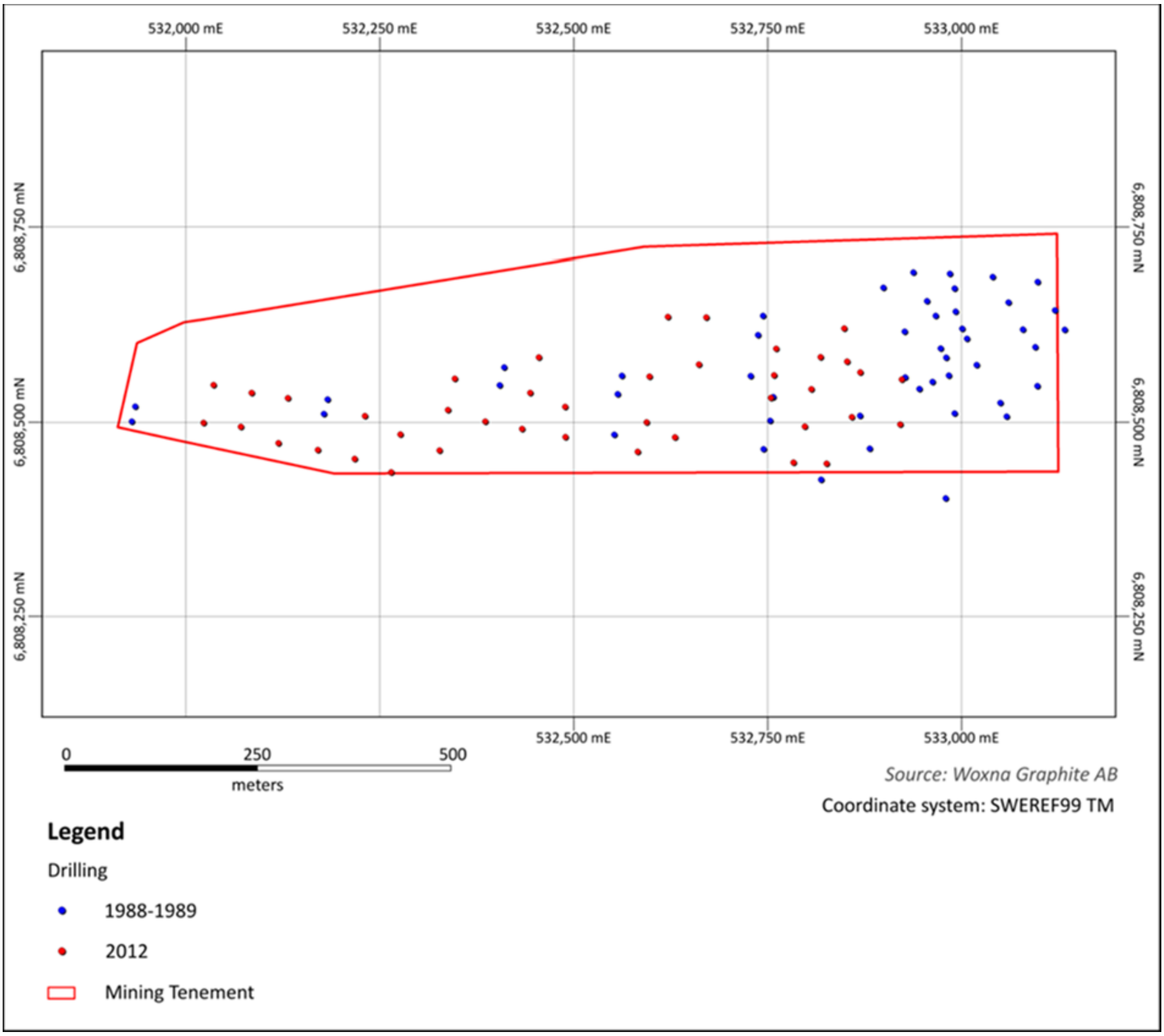


Figure 10-1: Drilling at the Kringel deposit

Table 10-2: Kringel drillhole collar co-ordinates (SWEREF99 TM Grid)

Hole_ID	Easting (m)	Northing (m)	RL (m)	Total Depth	Dip (°)	Azimuth (°)	Drill Type	Hole Size
KRIN88001	533,050	6,808,524	254.28	52.15	-60	349	DD	35 mm
KRIN88002	533,058	6,808,507	252.52	48.80	-60	349	DD	35 mm
KRIN88003	533,098	6,808,546	256.47	35.75	-60	345	DD	35 mm
KRIN88004	532,963	6,808,551	251.34	37.30	-60	65	DD	35 mm
KRIN88005	532,946	6,808,542	250.82	46.3	-60	65	DD	35 mm
KRIN88006	532,980	6,808,583	255.10	25.05	-60	345	DD	35 mm
KRIN88007	532,983	6,808,560	253.36	99.6	-60	349	DD	35 mm
KRIN88008	533,007	6,808,607	257.50	34.15	-60	349	DD	35 mm
KRIN88009	532,938	6,808,692	266.06	34.25	-60	345	DD	35 mm
KRIN88010	532,955	6,808,655	260.42	48.85	-60	345	DD	35 mm
KRIN88011	532,966	6,808,636	257.73	57.65	-60	345	DD	35 mm
KRIN88012	532,985	6,808,691	265.32	30.35	-60	349	DD	35 mm
KRIN88013	532,991	6,808,672	263.38	48.7	-60	349	DD	35 mm
KRIN88014	532,992	6,808,642	259.04	43.1	-60	349	DD	35 mm
KRIN88015	533,040	6,808,686	268.74	35.75	-60	349	DD	35 mm
KRIN88016	533,060	6,808,654	263.05	56.15	-60	345	DD	35 mm
KRIN88017	532,991	6,808,511	250.38	94.15	-60	358	DD	35 mm
KRIN88018	532,899	6,808,673	261.82	39.8	-60	349	DD	35 mm
KRIN88019	532,973	6,808,594	255.14	72.7	-60	334	DD	35 mm
KRIN88020	533,001	6,808,620	256.09	66.8	-60	349	DD	35 mm
KRIN88021	533,079	6,808,619	255.19	81.85	-60	334	DD	35 mm
KRIN88022	533,095	6,808,596	258.22	100.85	-60	334	DD	35 mm
KRIN88023	533,098	6,808,680	267.29	48.7	-60	345	DD	35 mm
KRIN88024	533,120	6,808,643	264.08	76.8	-55	345	DD	35 mm
KRIN88025	533,133	6,808,618	261.96	86.6	-50	345	DD	35 mm
KRIN88026	533,019	6,808,573	255.83	98.8	-50	345	DD	35 mm
KRIN88027	532,927	6,808,557	248.14	48.05	-55	349	DD	35 mm
KRIN88028	532,869	6,808,508	252.12	45.75	-50	341	DD	35 mm
KRIN89001	531,936	6,808,520	222.67	37.1	-50	16	DD	35 mm
KRIN89002	531,931	6,808,500	222.50	59.25	-51	12	DD	35 mm
KRIN89003	532,183	6,808,529	226.77	39.9	-49	11	DD	35 mm
KRIN89004	532,179	6,808,510	226.41	46.9	-51	21	DD	35 mm
KRIN89005	532,411	6,808,570	233.03	44.35	-47	20	DD	35 mm
KRIN89006	532,405	6,808,547	233.02	69.2	-42	14	DD	35 mm
KRIN89007	531,302	6,808,230	237.65	47.55	-50	10	DD	35 mm
KRIN89008	532,562	6,808,559	245.95	45.5	-60	22	DD	35 mm
KRIN89009	531,298	6,808,211	237.13	62.5	-50	10	DD	35 mm

Hole_ID	Easting (m)	Northing (m)	RL (m)	Total Depth	Dip (°)	Azimuth (°)	Drill Type	Hole Size
KRIN89010	532,557	6,808,536	244.02	59.65	-55	18	DD	35 mm
KRIN89011	531,977	6,808,092	217.95	40.45	-50	15	DD	35 mm
KRIN89012	532,744	6,808,636	248.02	55.25	-53	18	DD	35 mm
KRIN89013	531,971	6,808,073	218.50	61.8	-50	15	DD	35 mm
KRIN89014	532,738	6,808,612	254.73	60.3	-50	15	DD	35 mm
KRIN89015	532,757	6,808,532	250.70	41.1	-49	14	DD	35 mm
KRIN89016	532,753	6,808,501	249.01	46	-60	12	DD	35 mm
KRIN89017	532,819	6,808,426	246.01	44.85	-60	11	DD	35 mm
KRIN89018	532,745	6,808,465	247.25	67.4	-58	13	DD	35 mm
KRIN89019	532,881	6,808,465	250.84	67.95	-60	348	DD	35 mm
KRIN89020	532,979	6,808,402	247.34	38.6	-60	8	DD	35 mm
KRIN89021	532,927	6,808,616	256.23	63.2	-60	342	DD	35 mm
KRIN89022	532,728	6,808,559	252.34	101.65	-49	8	DD	35 mm
KRIN89023	532,553	6,808,483	240.70	113.1	-60	13	DD	35 mm
KRI12DD001	532,849	6,808,620	258.00	48.9	-50	358	DD	42 mm
KRI12DD002	532,853	6,808,578	253.77	77.55	-50	347	DD	42 mm
KRI12DD003	532,921	6,808,497	251.11	119.65	-50	340	DD	42 mm
KRI12DD004	532,826	6,808,446	247.95	80.6	-50	23	DD	42 mm
KRI12DD005	532,783	6,808,447	246.24	49.3	-50	18	DD	42 mm
KRI12DD006	532,798	6,808,494	248.93	127.35	-50	29	DD	42 mm
KRI12DD007	532,807	6,808,542	252.09	124.2	-50	349	DD	42 mm
KRI12DD008	532,758	6,808,560	253.82	101.6	-49	12	DD	42 mm
KRI12DD009	532,761	6,808,594	257.38	68.7	-55	19	DD	42 mm
KRI12DD010	532,818	6,808,583	255.52	94	-50	7	DD	42 mm
KRI12DD011	532,671	6,808,634	240.34	71.9	-50	20	DD	42 mm
KRI12DD012	532,622	6,808,635	240.28	54.1	-50	15	DD	42 mm
KRI12DD013	532,859	6,808,506	251.99	137.45	-50	13	DD	42 mm
KRI12DD014	532,662	6,808,574	252.41	113.5	-50	15	DD	42 mm
KRI12DD015	532,598	6,808,558	246.94	113.35	-50	15	DD	42 mm
KRI12DD016	532,631	6,808,480	248.69	184.5	-50	22	DD	42 mm
KRI12DD017	532,594	6,808,499	247.53	87.5	-50	17	DD	42 mm
KRI12DD018	532,583	6,808,461	243.66	145.1	-50	7	DD	42 mm
KRI12DD019	532,490	6,808,480	235.36	115.1	-50	8	DD	42 mm
KRI12DD020	532,489	6,808,519	238.74	92.6	-49	19	DD	42 mm
KRI12DD021	532,444	6,808,537	233.11	77.7	-50	13	DD	42 mm
KRI12DD022	532,434	6,808,491	232.18	122.15	-50	37	DD	42 mm
KRI12DD023	532,328	6,808,463	229.97	84.45	-50	22	DD	42 mm
KRI12DD024	532,338	6,808,515	230.76	80.4	-50	15	DD	42 mm

Hole_ID	Easting (m)	Northing (m)	RL (m)	Total Depth	Dip (°)	Azimuth (°)	Drill Type	Hole Size
KRI12DD025	532,347	6,808,556	233.41	48.45	-50	16	DD	42 mm
KRI12DD026	532,277	6,808,484	227.80	79.2	-50	11	DD	42 mm
KRI12DD027	532,265	6,808,435	230.32	118.2	-50	17	DD	42 mm
KRI12DD028	532,231	6,808,507	227.40	62.9	-50	14	DD	42 mm
KRI12DD029	532,218	6,808,452	226.72	86.8	-50	15	DD	42 mm
KRI12DD030	532,171	6,808,464	226.56	83.25	-50	16	DD	42 mm
KRI12DD031	532,120	6,808,473	226.33	83.6	-50	15	DD	42 mm
KRI12DD032	532,132	6,808,530	225.68	40.4	-51	13	DD	42 mm
KRI12DD033	532,086	6,808,537	224.88	49.4	-49	16	DD	42 mm
KRI12DD034	532,072	6,808,493	225.23	74.55	-49	13	DD	42 mm
KRI12DD035	532,037	6,808,547	224.00	41.9	-49	15	DD	42 mm
KRI12DD036	532,024	6,808,499	225.96	71.6	-50	14	DD	42 mm
KRI12DD037	532,386	6,808,500	231.31	98.55	-49	13	DD	42 mm
KRI12DD038	532,456	6,808,583	233.41	41.5	-46	30	DD	42 mm
KRI12DD039	532,754	6,808,531	250.64	111.9	-50	14	DD	42 mm

Source: ReedLeyton 2012

10.2 Drillhole verification - Kringel

Twelve drillholes were examined in detail, by Geoffrey Reed of ReedLeyton, on 13 June 2012, at the Kringel site, refer Table 10-3 below.

Table 10-3: Drillholes and core examined by ReedLeyton, Kringel Deposit

Project	Hole number	Approximate meters examined
Kringel	KRIN88014	40
Kringel	KRIN88015	30
Kringel	KRIN89015	35
Kringel	KRIN89021	60
Kringel	KRIN89022	95
Kringel	KRIN89023	100
Kringel	KRI12DD001	42
Kringel	KRI12DD003	110
Kringel	KRI12DD007	120
Kringel	KRI12DD008	90
Kringel	KRI12DD009	60
Kringel	KRI12DD010	90

Source: ReedLeyton 2012

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling methodology, sample preparation and analysis

The disclosure for the Mattsmyra, Gropabo and Mansberg deposits is considered historical and is provided in Section 6. The following information pertains to the 2012 Flinders drilling campaign for the Kringel deposit. Some additional check analyses were undertaken to verify the historical data for Mattsmyra and Gropabo and the results are reported in this section.

For the Flinders exploration campaign, a core samples were half-sectioned and submitted for analysis in geologically meaningful lengths. Physical records of these are available to Woxna Graphite. All lengths quoted are down hole and not “true” widths.

Drill core from the 2012 program was logged at the Woxna Mine site. Once geologically logged, RQD measurements were taken. Core was then photographed, and magnetically measured prior to storage on pallets at the site. Regular batches of samples were then sent via independent contractor for analysis. Flinders geologists Lars Dahlenborg, Elin Eyösä and Janne Kinnunen supervised sampling of all drillholes drilled in 2012.

Field preparation involved the marking of sample intervals were marked on the core and the core tray. Each interval was given a unique sample number. The sample numbers were taken from unique sample ticket booklets made for Flinders. One part of the sample ticket was placed in the bag together with the cut core. The sample numbers ranged from 26700 to 28300.

A total of 92 blank samples were inserted at a rate of approximately 1 in 15, resulting in approximately 7% of the submitted samples being blanks.

The core was then shipped by truck to ALS Chemex preparation laboratory in Piteå for core cutting, where core was split by diamond saw under supervision by Flinders staff. The cutting of core at ALS Chemex laboratory in Sweden is in keeping with industry practice, and security of the delivery chain is more than adequate (Flinders 2015). One half was retained as a check sample and the remainder bagged for analysis. The core was cut in consideration of the main foliation/banding of the rock. The retained core was examined by the Qualified Person in 2013 and was considered fresh with little or no secondary mineralisation. The retained core was stored at the Woxna Mine site which is a key access only facility, and there is no evidence that samples have been disturbed in any way since cutting.

The laboratory that completed analysis of the Kringel samples was the Government owned SGAB ANALYS, (Box 801, Luleå, Sweden 95128). The laboratories that carried out the sample preparation and analysis were independent of the Woxna Graphite. No details of certification by any standards associations and the particulars of any certification are known, however the laboratory was well regarded and applied best practice of the day.

The Kringel drilling programme resulted in 1,344 graphite analyses. Every 6th sample (17%) was assayed for sulfur up until drillhole KRI12015. Flinders was advised to assay every 2nd sample for sulfur, starting with drill hole KRI12016. On average, every 3rd sample (33%) was assayed for sulfur. In total, 441 samples were assayed for sulfur and every 12th sample (8%) was assayed by ICP-MS for major and minor elements.

The Leco analysis methodology is digitally controlled and designed to measure the carbon and sulfur content in a wide variety of organic materials, as well as inorganic samples including soil, cement, and limestone. Analysis begins by weighing out a sample into a combustion crucible. On analysis, the sample is typically combusted at >1,350^{0}°C in a pure oxygen environment. All sample materials contained in the crucible go through an oxidative reduction process which causes carbon-bearing compounds to break down, producing elemental carbon, which oxidises to form CO_2. From the combustion chamber, the gases flow through two Anhydrone ($MgClO_4$) tubes to remove moisture, through a flow controller (3.5 l/min) then through to an infrared (IR) detection cell. The IR cell measures the concentration of carbon dioxide gas present. The LECO analysers have an inherent manufacturer specified accuracy of ±1% carbon present.

Density measurements were conducted by Flinders staff using the Archimedes method. In total 1,424 measurements were made covering each assay interval as well as the lithologies in the foot and hanging walls.

11.2 Kringel - quality control and assurance

The majority of the Kringel core examined by the Qualified Person in 2013 had been previously cut by diamond saw into halves, and in some cases, quartered. The core was re-sampled in a verification exercise as close to the original sample lengths as possible for direct comparison. The 2012 Kringel re-sampling exercise included 59 samples prepared at ALS Chemex. The check analysis batches included both crusher duplicates and pulp duplicates at a rate of >10%.

Interrogation of the verification samples is provided in detail in Flinders 2013 and summarised in Section 12.1.1.3. Paired plots (Figure 12-1, Figure 12-2, Figure 12-3) show a high degree of correlation between parent data and duplicate data. The exercise has shown that the procedure has a high degree of reproducibility.

12 Data Verification

12.1 Kringel - database verification by qualified person

Assay data in original laboratory sheets from the 1988 and 1989 drilling programme has not been examined by ReedLeyton. The paper records for collar, assay, survey, and geology data were digitised for the 2002 Claesson et al historical Mineral Resource estimate for Kringel and this dataset was compared to the digital data available to Woxna Graphite. Claesson (2002) concluded that the historical data is of sufficient quality and traceable provenance that it is useable as exploration data. The then supervising geologist, who is a QP under current NI 43-101 protocol, also verified the provenance of the data supplied.

Following the acquisition of the historical database by Flinders, the capture of 2012 drilling campaign digital data was completed by Flinders staff. Hard copy data has been verified and all data is stored in a database and managed by Woxna Graphite. The digital data has been both randomly and systematically checked by ReedLeyton and shown to be correct using a number of checks listed below:

- the digital data was compiled directly into Microsoft Excel™ and validated in Microsoft Access™ and exported into a csv format; and
- the database was then imported into Maptek Vulcan software in the csv format and checked in sections and visually;

ReedLeyton also viewed the Kringel paper records located in the Geology office of the Woxna Mine site and compared 5% of the records with the Flinders Digital database in June 2012.

The historical data for Kringel, Mattsmyra and Gropabo was independently reviewed by Coffey Mining Limited for a Mineral Resource estimate published in 2011 (as shown in Table 6-1) and was considered appropriate for use in a resource estimate. ReedLeyton concurs with this conclusion based on its own verification of the data for Kringel, Mattsmyra and Gropabo.

ReedLeyton has not viewed the data for Månsberg and although the Månsberg data was previously reviewed by Coffey Mining Limited, the Månsberg data remains historical by nature and should not be relied upon.

12.1.1 Check sampling - Kringel

A sampling verification exercise was conducted on the 2012 drilling samples which was summarised in Section 11.2 and described below.

The Kringel re-sampling included 59 samples (Table 12-1). The core trays selected for re-sampling were taken to the Flinders core saw facility by a Flinders field assistant, and quarter-core (as appropriate and available) sections were re-sawn for the check samples.

ReedLeyton checked that the correct samples were taken, sawn, and the resulting sample were placed in individual sample bags with an identifying tag. The bags were sealed with a plastic tie. The bags were retained under ReedLeyton's supervision, and personally delivered to the ALS Chemex laboratory manager (Tony Ökvist) at Öjebyn (Sweden) for further processing and transport.

Table 12-1: Check sample intervals by ReedLeyton representative

Project	Drillhole Number	Check Sample From (m)	Check Sample To (m)	Check Sample Interval (m)	Check Sample Number
Kringel	KRIN88014	4.75	6.75	2	27,796
Kringel	KRIN88014	8.75	10.75	2	27,797

Project	Drillhole Number	Check Sample From (m)	Check Sample To (m)	Check Sample Interval (m)	Check Sample Number
Kringel	KRIN88014	13.75	15.75	2	27,798
Kringel	KRIN88014	15.75	17.75	2	27,799
Kringel	KRIN88014	19.75	21.75	2	27,801
Kringel	KRIN88014	21.75	24.05	2.3	27,802
Kringel	KRIN88015	8.05	10.05	2	27,803
Kringel	KRIN88015	10.05	12.05	2	27,804
Kringel	KRIN88015	14.3	15.35	1.05	27,805
Kringel	KRIN88015	21	22.4	1.4	27,806
Kringel	KRIN88015	22.4	24.6	2.2	27,807
Kringel	KRIN89015	7.45	8.45	1	27,808
Kringel	KRIN89015	9.05	10.4	1.35	27,809
Kringel	KRIN89015	19.25	20.4	1.15	27,810
Kringel	KRIN89015	21.85	23.85	2	27,812
Kringel	KRIN89021	30.4	32.4	2	27,813
Kringel	KRIN89021	32.4	34.4	2	27,814
Kringel	KRIN89021	38.7	40.7	2	27,815
Kringel	KRIN89021	40.7	42.7	2	27,816
Kringel	KRIN89022	10	11.2	1.2	27,817
Kringel	KRIN89022	12.2	14.2	2	27,818
Kringel	KRIN89022	72.4	74.4	2	27,819
Kringel	KRIN89022	74.4	76.4	2	27,820
Kringel	KRIN89022	79.35	79.9	0.55	27,822
Kringel	KRIN89022	82	84	2	27,823
Kringel	KRIN89023	87.2	88.3	1.1	27,824
Kringel	KRIN89023	88.3	90.3	2	27,825
Kringel	KRIN89023	96	98	2	27,826
Kringel	KRIN89023	98	100	2	27,827
Kringel	KRI12DD001	14	15	1	27,828
Kringel	KRI12DD001	23.4	24.2	0.8	27,829
Kringel	KRI12DD001	43.5	44.5	1	27,830
Kringel	KRI12DD003	49	50	1	27,831
Kringel	KRI12DD003	57	58	1	27,860
Kringel	KRI12DD003	93.5	94.5	1	27,833
Kringel	KRI12DD003	94.5	95.5	1	27,834
Kringel	KRI12DD003	103.8	104.8	1	27,835
Kringel	KRI12DD003	104.8	105.8	1	27,836
Kringel	KRI12DD007	68.1	69.1	1	27,837
Kringel	KRI12DD007	74.6	75.6	1	27,838

Project	Drillhole Number	Check Sample From (m)	Check Sample To (m)	Check Sample Interval (m)	Check Sample Number
Kringel	KRI12DD007	79.4	80.4	1	27,839
Kringel	KRI12DD007	113.6	114.6	1	27,841
Kringel	KRI12DD007	116.6	117.6	1	27,842
Kringel	KRI12DD008	59.75	60.75	1	27,843
Kringel	KRI12DD008	62.75	63.75	1	27,844
Kringel	KRI12DD008	70.1	71.1	1	27,845
Kringel	KRI12DD008	72	73	1	27,846
Kringel	KRI12DD008	80.75	81.75	1	27,847
Kringel	KRI12DD009	43.8	44.8	1	27,848
Kringel	KRI12DD009	48.8	49.8	1	27,849
Kringel	KRI12DD009	49.8	50.8	1	27,851
Kringel	KRI12DD009	52.8	53.8	1	27,852
Kringel	KRI12DD009	57.8	58.4	0.6	27,853
Kringel	KRI12DD010	35.15	36.15	1	27,854
Kringel	KRI12DD010	40.4	41.4	1	27,855
Kringel	KRI12DD010	43.4	44.4	1	27,856
Kringel	KRI12DD010	53.5	54.5	1	27,857
Kringel	KRI12DD010	59.7	60.7	1	27,858
Kringel	KRI12DD010	82.6	83.6	1	27,859

Source: ReedLeyton 2012

12.1.1.1 Check sampling - Mattsmyra

The Mattsmyra re-sampling included 26 samples (Table 12-5). The core trays selected for re-sampling were taken to the Flinders core saw facility by a Flinders field assistant, and quarter-core (as appropriate and available) sections were re-sawn for the check samples.

Table 12-2: Check sample intervals- Mattsmyra deposit

Deposit	Hole Number	Check Sample From (m)	Check Sample To (m)	Check Sample Interval (m)	Check Sample Number
Mattsmyra	MAT90006	25.5	26.55	1.05	28542
Mattsmyra	MAT90006	30.2	33.5	3.3	28543
Mattsmyra	MAT90006	99.95	100.3	0.35	28544
Mattsmyra	MAT90006	99.95	100.3	0.35	28546
Mattsmyra	MAT90006	105.6	108.9	3.3	28548
Mattsmyra	MAT90006	112.1	115.4	3.3	28549
Mattsmyra	MAT90006	118.5	121	2.5	28550
Mattsmyra	MAT90006	124.2	126.9	2.7	28551

Deposit	Hole Number	Check Sample From (m)	Check Sample To (m)	Check Sample Interval (m)	Check Sample Number
Mattsmyra	MAT90006	129.6	132.3	2.7	28552
Mattsmyra	MAT90009	92.85	95.25	2.4	28553
Mattsmyra	MAT90009	98.45	101.2	2.75	28554
Mattsmyra	MAT90009	98.45	101.2	2.75	28556
Mattsmyra	MAT90009	107.9	110.3	2.4	28558
Mattsmyra	MAT90009	112.8	114.7	1.9	28559
Mattsmyra	MAT90009	122.1	123.2	1.1	28560
Mattsmyra	MAT90009	133.25	135.15	1.9	28561
Mattsmyra	MAT92004	41.5	43.5	2	28562
Mattsmyra	MAT92004	45.5	48.5	3	28563
Mattsmyra	MAT92004	51.5	54.1	2.6	28564
Mattsmyra	MAT92004	57.15	59.15	2	28565
Mattsmyra	MAT92004	62.15	65.15	3	28566
Mattsmyra	MAT92004	62.15	65.15	3	28568
Mattsmyra	MAT92004	68.3	69.55	1.25	28570
Mattsmyra	MAT92004	72.5	74.5	2	28571
Mattsmyra	MAT92004	76.1	79.1	3	28572
Mattsmyra	MAT92004	81.1	83.2	2.1	28573

Source: ReedLeyton 2015

12.1.1.2 Check sampling – Gropabo deposit

The Gropabo re-sampling included 26 samples (Table 12-3). The core trays selected for re-sampling were taken to the Flinders core saw facility by a Flinders field assistant, and quarter-core (as appropriate and available) sections were re-sawn for the check samples.

Final results were received via direct email from ALS Chemex on 5 November 2014 Both the raw data and the analysis certificate were received. Table 12-3 shows the analysis values for Cg only, comparing the original sample interval and value versus the check sample interval and value. There is extremely good agreement between the individual samples.

Table 12-3: Check sample intervals by ReedLeyton, Gropabo Deposit

Deposit	Drillhole Number	Check Sample From (m)	Check Sample To (m)	Check Sample Interval (m)	Original Data Cg (%)	Check Sample Number	Check Data Cg (%)
Gropabo	GRO91013	5.5	7.5	2	8.4	28516	8.57
Gropabo	GRO91013	9.5	11.2	1.7	4.9	28517	4.99
Gropabo	GRO91013	13.2	15.6	2.4	5.5	28518	6.08
Gropabo	GRO91013	17.6	19.05	1.45	7.3	28522	6.27
Gropabo	GRO91013	24.45	26.45	2	6	28523	6.04
Gropabo	GRO91013	28.45	30.45	2	10.7	28524	11.15

Deposit	Drillhole Number	Check Sample From (m)	Check Sample To (m)	Check Sample Interval (m)	Original Data Cg (%)	Check Sample Number	Check Data Cg (%)
Gropabo	GRO91013	38.4	41.4	3	0.95	28525	0.77
Gropabo	GRO91013	50.25	50.45	0.2	11.8	28526	11.3
Gropabo	GRO92003	12.15	14.85	2.7	7.7	28527	6.78
Gropabo	GRO92003	18.2	21.2	3	10.2	28528	10.75
Gropabo	GRO92003	25.85	27.85	2	10	28529	9.12
Gropabo	GRO92003	31.1	32.4	1.3	6.8	28533	6.96
Gropabo	GRO92003	48.9	50.9	2	1.9	28534	1.71
Gropabo	GRO92003	52.65	55	2.35	8.3	28535	7.84
Gropabo	GRO92003	56.05	58.05	2	10.6	28536	10.25
Gropabo	GRO92003	60.05	62.05	2	12.6	28537	12.5
Gropabo	GRO92003	64.05	66.2	2.15	12.2	28538	12.25
Gropabo	GRO92003	68.3	70.7	2.4	1.7	28539	1.8
Gropabo	GRO92003	72.7	74.7	2	13	28540	11.65
Gropabo	GRO92003	77.3	79.3	2	2.7	28541	2.52

Source: ReedLeyton 2015

12.1.1.3 Analysis of sampling check data

Check sampling by ReedLeyton of the Kringel core consisted of crush duplicates and pulp duplicates (see Section11.2). Greater than 10% of each batch sent to ALS Chemex has been composed of these duplicates.

In the tables below the term Cg refers to carbon content in the form of graphite.

All QA/QC data for the 2012 database has been deemed acceptable for the purposes of the Mineral Resource estimation.

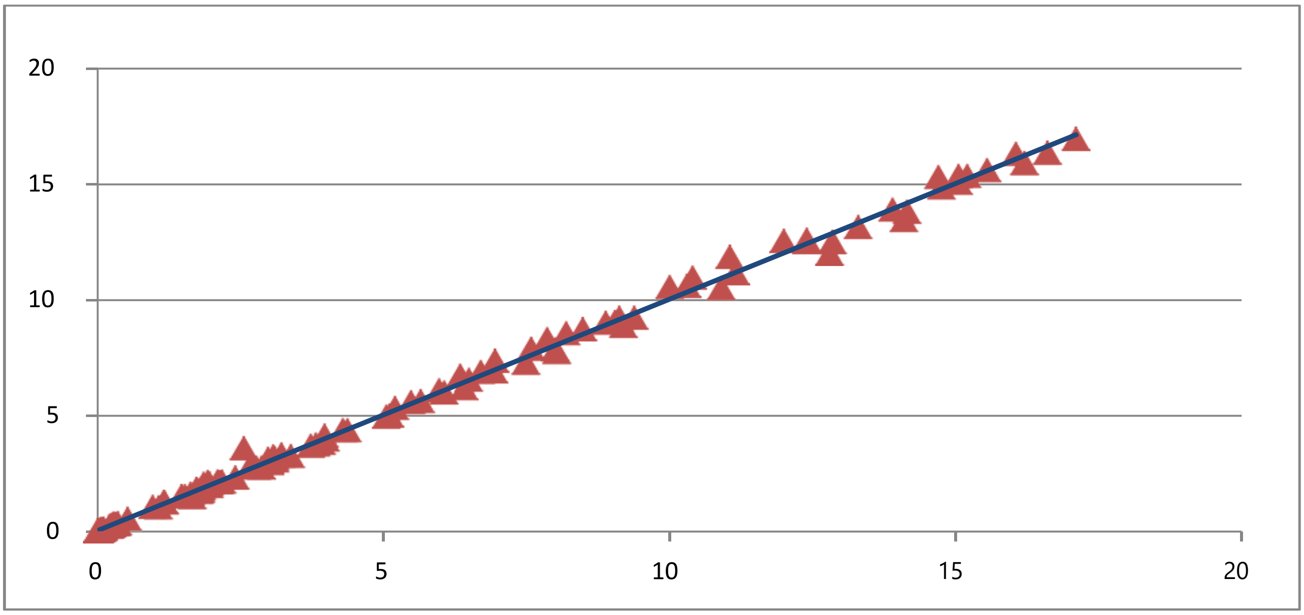


Figure 12-1: Coarse duplicate data for Cg%, Kringel deposit (Source: ReedLeyton 2012)

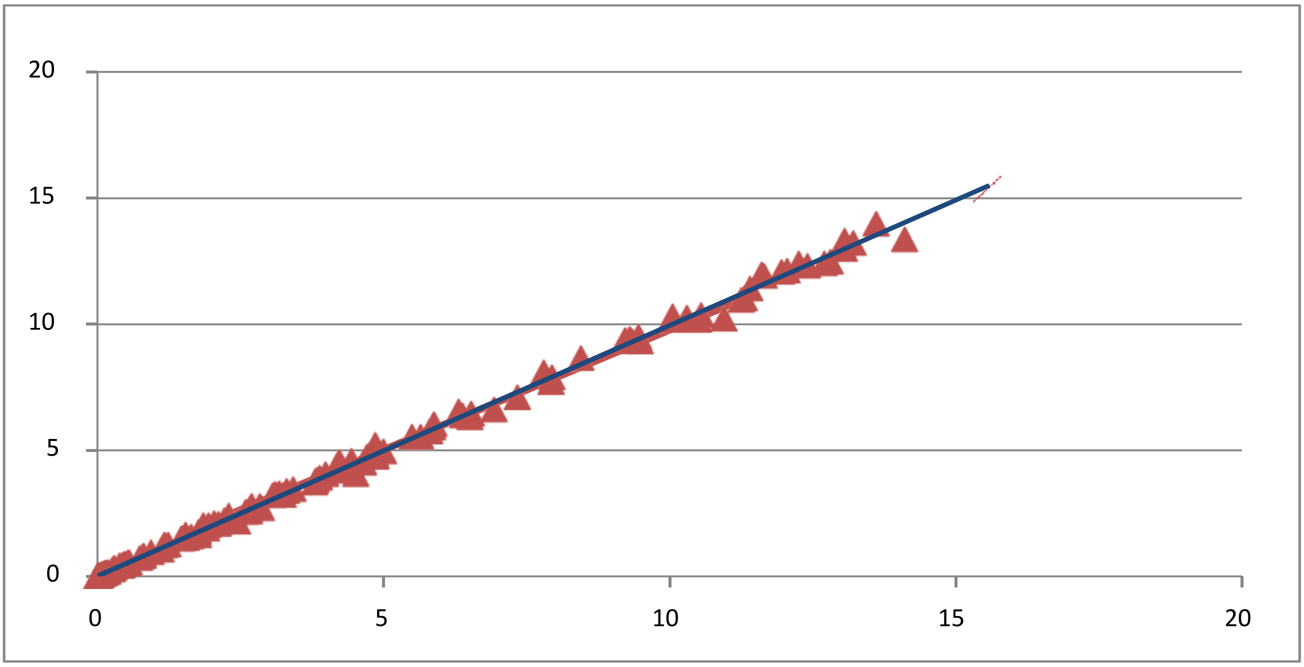


Figure 12-2: Pulp duplicate data for Cg%, Kringel deposit (Source: ReedLeyton 2012)

Figure 12-2 demonstrates a high degree of correlation between Parent data (Vertical axis) and duplicate data (Horizontal axis). Blue lines provide a 1 to 1 correlation trend.

12.1.1.4 Results

Analytical results were received via direct email from ALS Chemex on 18 June 2012. Both the raw data and the analysis certificate were received. A comparison of the original data interval as compared to the check samples is shown in Table 12-4. There is extremely good agreement between the individual samples from both datasets.

Table 12-4: Drillhole core re-sampled for check analysis, matched with original assays

Project	Hole Number	From (m)	To (m)	Original Data Interval (m)	Historical Data Cg (%)	Check Sample Number	Check Data Cg (%)
Kringel	KRIN88014	4.75	6.75	2.00	16.30	27,796	12.25
Kringel	KRIN88014	8.75	10.75	2.00	15.70	27,797	13.05
Kringel	KRIN88014	13.75	15.75	2.00	13.70	27,798	10.35
Kringel	KRIN88014	15.75	17.75	2.00	10.20	27,799	8.79
Kringel	KRIN88014	19.75	21.75	2.00	6.00	27,801	5.31
Kringel	KRIN88014	21.75	24.05	2.30	4.30	27,802	3.49
Kringel	KRIN88015	8.05	10.05	2.00	13.10	27,803	10.95
Kringel	KRIN88015	10.05	12.05	2.00	5.10	27,804	5.72
Kringel	KRIN88015	14.30	15.35	1.05	7.50	27,805	4.95
Kringel	KRIN88015	21.00	22.40	1.40	5.10	27,806	4.29
Kringel	KRIN88015	22.40	24.60	2.20	5.00	27,807	4.66
Kringel	KRIN89015	7.45	8.45	1.00	3.50	27,808	2.87

Project	Hole Number	From (m)	To (m)	Original Data Interval (m)	Historical Data Cg (%)	Check Sample Number	Check Data Cg (%)
Kringel	KRIN89015	9.05	10.40	1.35	4.80	27,809	4.73
Kringel	KRIN89015	19.25	20.40	1.15	3.60	27,810	2.67
Kringel	KRIN89015	21.85	23.85	2.00	4.60	27,812	4.17
Kringel	KRIN89021	30.40	32.40	2.00	12.50	27,813	12.25
Kringel	KRIN89021	32.40	34.40	2.00	13.50	27,814	10.55
Kringel	KRIN89021	38.70	40.70	2.00	5.70	27,815	4.76
Kringel	KRIN89021	40.70	42.70	2.00	6.10	27,816	5.09
Kringel	KRIN89022	10.00	11.20	1.20	3.40	27,817	3.06
Kringel	KRIN89022	12.20	14.20	2.00	5.50	27,818	5.13
Kringel	KRIN89022	72.40	74.40	2.00	9.40	27,819	8.87
Kringel	KRIN89022	74.40	76.40	2.00	10.80	27,820	9.24
Kringel	KRIN89022	79.35	79.90	0.55	-	27,822	1.10
Kringel	KRIN89022	82.00	84.00	2.00	8.60	27,823	7.86
Kringel	KRIN89023	87.20	88.30	1.10	6.10	27,824	5.17
Kringel	KRIN89023	88.30	90.30	2.00	9.90	27,825	8.35
Kringel	KRIN89023	96.00	98.00	2.00	7.50	27,826	6.25
Kringel	KRIN89023	98.00	100.00	2.00	9.90	27,827	9.55
Kringel	KRI12DD001	14.00	15.00	1.00	8.77	27,828	8.51
Kringel	KRI12DD001	23.40	24.20	0.80	2.68	27,829	2.52
Kringel	KRI12DD001	43.50	44.50	1.00	1.77	27,830	1.41
Kringel	KRI12DD003	49.00	50.00	1.00	5.54	27,831	5.77
Kringel	KRI12DD003	57.00	58.00	1.00	14.15	27,860	14.65
Kringel	KRI12DD003	93.50	94.50	1.00	9.28	27,833	8.80
Kringel	KRI12DD003	94.50	95.50	1.00	4.72	27,834	5.12
Kringel	KRI12DD003	103.80	104.80	1.00	3.88	27,835	4.47
Kringel	KRI12DD003	104.80	105.80	1.00	5.34	27,836	5.98
Kringel	KRI12DD007	68.10	69.10	1.00	12.40	27,837	12.65
Kringel	KRI12DD007	74.60	75.60	1.00	0.05	27,838	<0.01
Kringel	KRI12DD007	79.40	80.40	1.00	6.80	27,839	8.17
Kringel	KRI12DD007	113.60	114.60	1.00	2.22	27,841	2.05
Kringel	KRI12DD007	116.60	117.60	1.00	4.62	27,842	4.61
Kringel	KRI12DD008	59.75	60.75	1.00	6.44	27,843	7.75
Kringel	KRI12DD008	62.75	63.75	1.00	7.98	27,844	8.79
Kringel	KRI12DD008	70.10	71.10	1.00	2.08	27,845	2.79
Kringel	KRI12DD008	72.00	73.00	1.00	7.93	27,846	7.99
Kringel	KRI12DD008	80.75	81.75	1.00	4.13	27,847	4.57
Kringel	KRI12DD009	43.80	44.80	1.00	7.60	27,848	7.49
Kringel	KRI12DD009	48.80	49.80	1.00	11.60	27,849	12.80

Project	Hole Number	From (m)	To (m)	Original Data Interval (m)	Historical Data Cg (%)	Check Sample Number	Check Data Cg (%)
Kringel	KRI12DD009	49.80	50.80	1.00	8.65	27,851	9.56
Kringel	KRI12DD009	52.80	53.80	1.00	5.66	27,852	5.77
Kringel	KRI12DD009	57.80	58.40	0.60	3.97	27,853	4.03
Kringel	KRI12DD010	35.15	36.15	1.00	11.70	27,854	13.10
Kringel	KRI12DD010	40.40	41.40	1.00	7.64	27,855	7.39
Kringel	KRI12DD010	43.40	44.40	1.00	4.05	27,856	4.16
Kringel	KRI12DD010	53.50	54.50	1.00	6.49	27,857	6.95
Kringel	KRI12DD010	59.70	60.70	1.00	3.82	27,858	4.21
Kringel	KRI12DD010	82.60	83.60	1.00	3.34	27,859	3.43

Source: ReedLeyton 2012

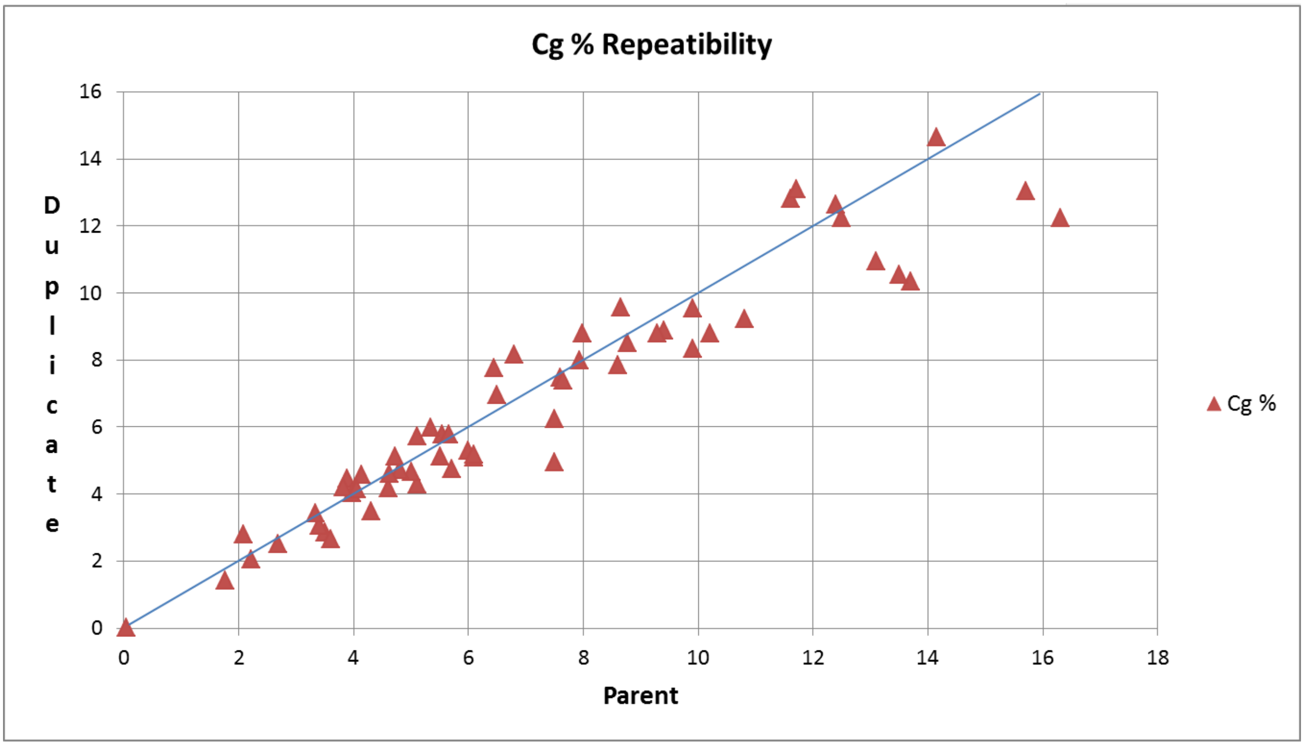


Figure 12-3: Paired historical and modern analytical data for Cg (Source: ReedLeyton 2012)

The paired plots in Figure 12-3 above, demonstrate high degree of correlation between historical data (Horizontal axis) and modern data (Vertical axis). Blue lines provide a 1 to 1 correlation trend.

12.2 Mattsmyra and Gropabo deposit – check sampling

The samples taken and re-assayed for the Mattsmyra and Gropabo deposits are shown in Table 12-2 and Table 12-3. There is extremely good agreement between the individual samples. The paired plots demonstrate a high degree of correlation between historical data (Vertical axis) and modern data (Horizontal axis). Blue lines provide 1 to 1 correlation trend.

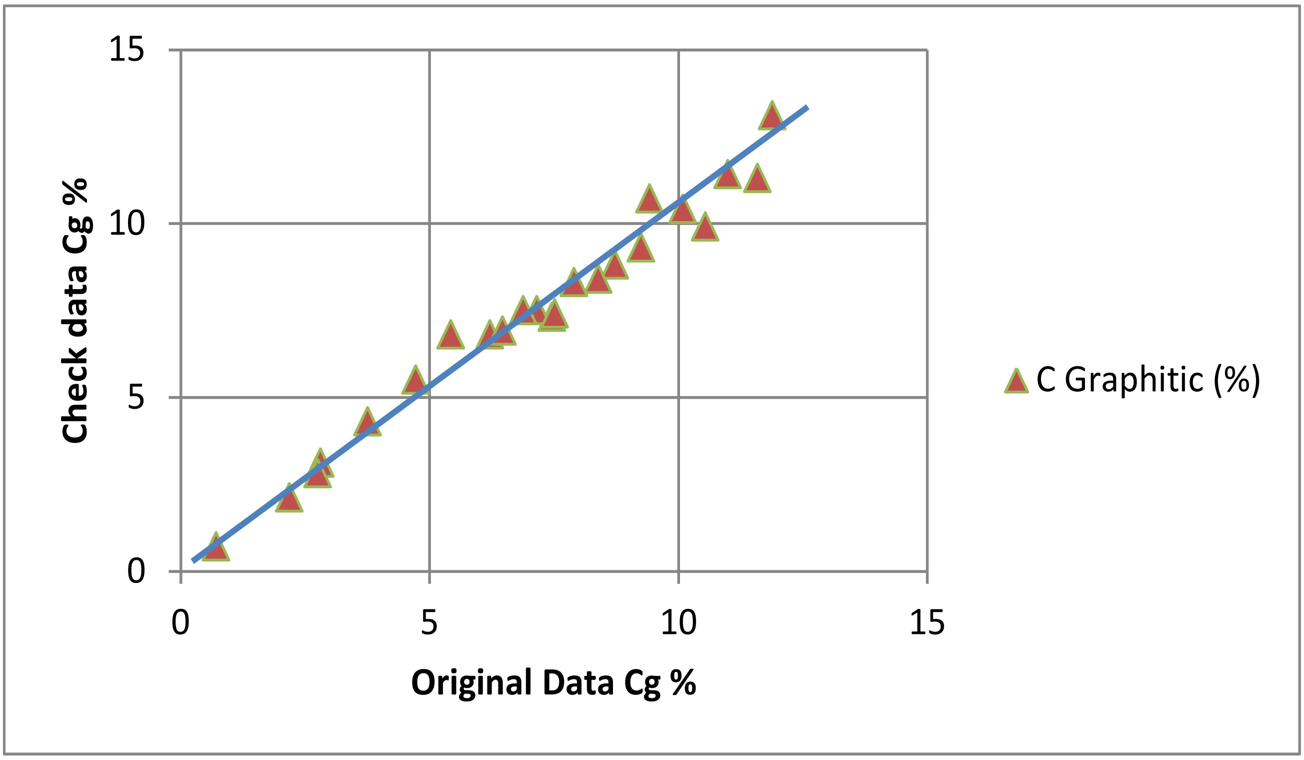


Figure 12-5: Check data Cg% compared to original date Cg%, Mattsmyra deposit (Source: ReedLeyton 2015)

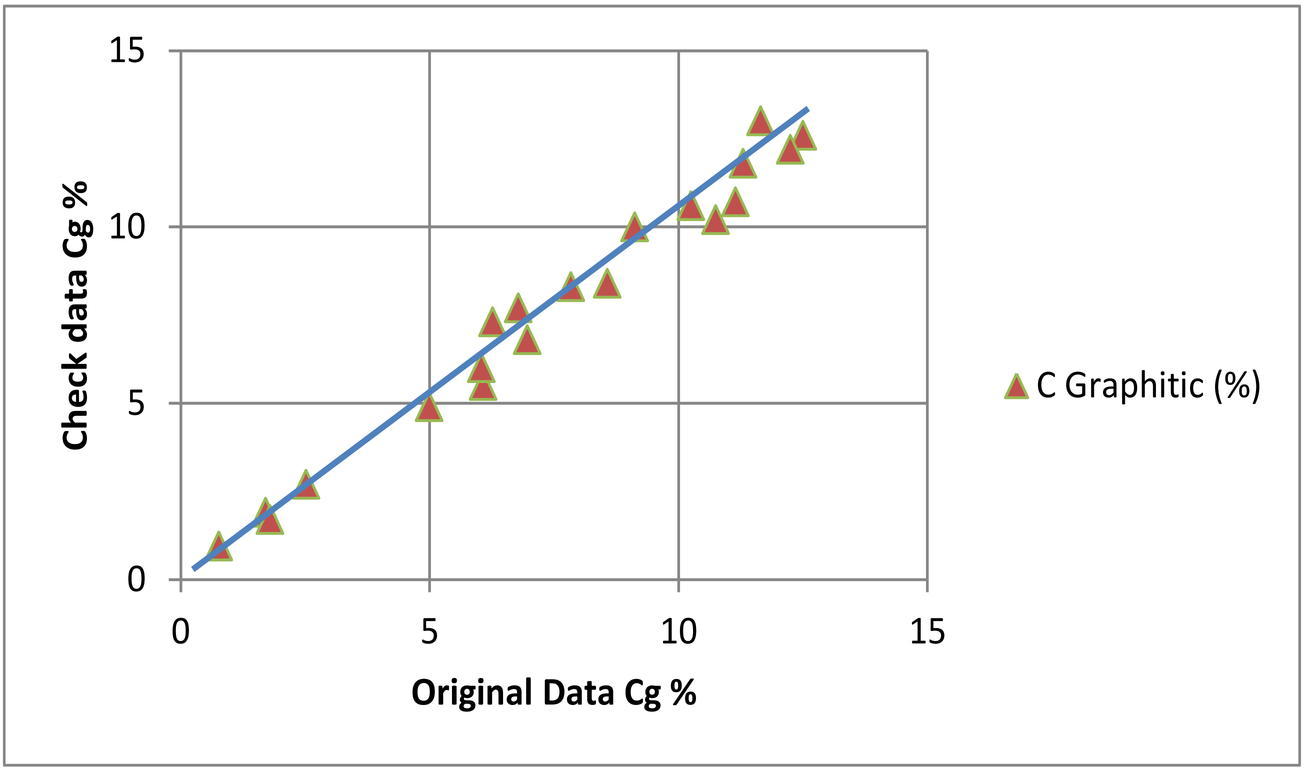


Figure 12-4: Check data Cg% compared to original date Cg%, Gropabo deposit (Source: ReedLeyton 2015)

12.3 Density

12.3.1 Density - Kringel

A total of 1,423 density determinations have been completed and determinations were calculated for wet and dry weight volumes.

The density measurements were performed on a representative 10 cm piece of drillhole core. The 10 cm sample was weighed dry on a balance then it was weighed while suspended in water. From the data, the density is calculated as follows:

Density = Weight of sample (g)/(Weight in air (g) – Weight in water (g))

The densities range between 2.35 t/m^3 and 3.67 t/m^3 with the majority of the determinations limited to the range 2.6 t/m^3 to 2.8 t/m^3. ReedLeyton divided the 1,423 density determination by the geological domains identified in the geological block models (see Section 14) and histogram plots confirm that the majority of the determinations average 2.7 t/m^3.

The average for the waste rock determinations was 2.7 t/m^3.

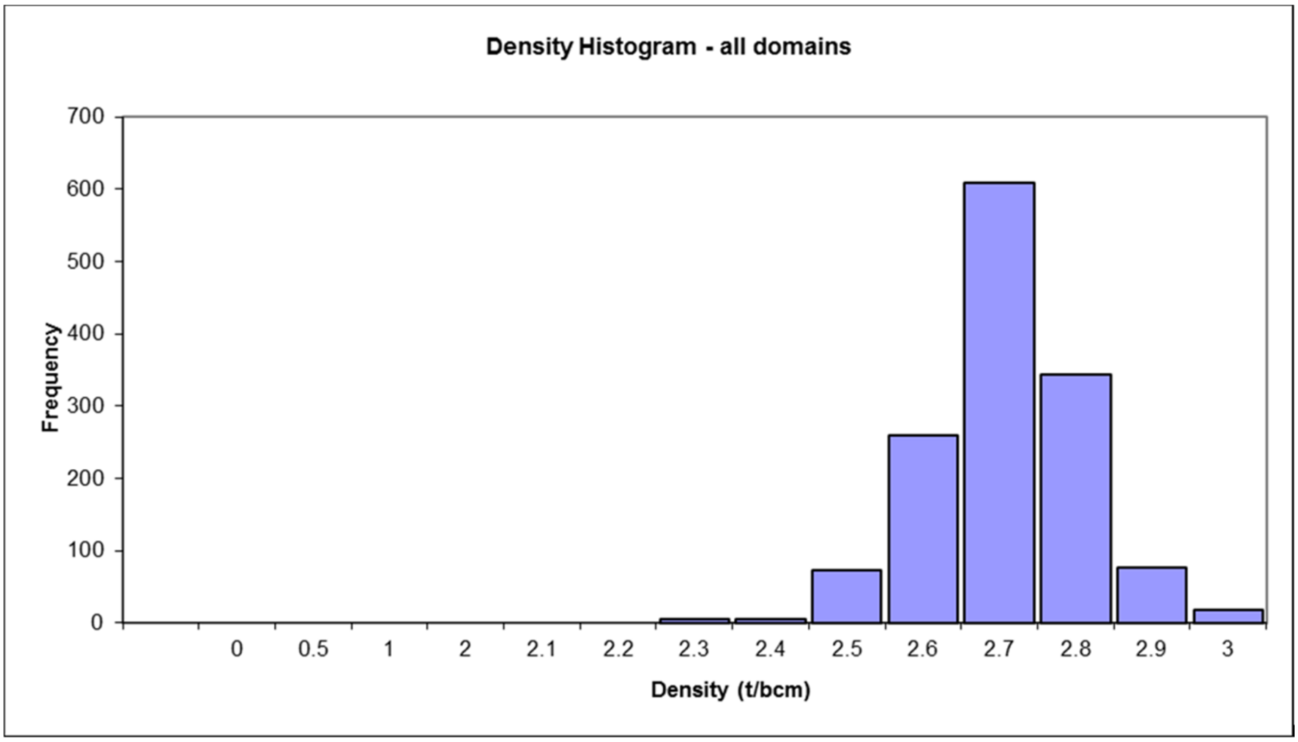


Figure 12-5: All 1,424 density determinations, Kringel Deposit

12.3.2 Density -Mattsmyra

A total of 458 density determinations have been completed with a range of values between 2.48 g/cm^3 and 3.86 g/cm^3 . The majority of determinations range from 2.6 g/cm^{3}to 2.9 g/cm^3. ReedLeyton has also divided the 458 bulk density determination by rock type in Figure 12-9 and 12-10. The density determinations were calculated wet and dry weight volume determinations. The majority of the determinations average 2.84 g/cm^3.

The average for the waste rock determinations was 2.7 g/cm^3.

The density value 2.82 for Type A Graphite and the value 2.86 for Type B Graphite was run according to density testwork by Flinders previously attributed to various assays within the geology database.

12.3.3 Gropabo Density

A total of 402 bulk density determinations have been completed with a range of values between 2.39 g/cm^3 and 3.05 t/m^3. The majority of determinations range from 2.6 g/cm^3 to 2.9 g/cm^3 with the average 2.79 g/cm^3. The density determinations were calculated as wet and dry weight volume determinations.

The average for the waste rock determinations was 2.7 g/cm^3.

The density value 2.81 for Type A Graphite and the value 2.83 for Type B Graphite was run according to density testwork by Flinders previously attributed to various assays within the geology database.

12.4 Limitations or failure to conduct verification

ReedLeyton checked that the correct samples were taken, sawn, and the resulting sample bulks were placed in individual sample bags with an identifying tag. The bags were sealed with a plastic tie. The bags were retained under ReedLeyton’s supervision, and personally delivered to the ALS Chemex laboratory manager (Tony Ökvist) at Öjebyn (Sweden) for further processing and transport.

12.5 Adequacy of the data

Cutting of core and dispatch to the ALS Chemex laboratory in Sweden is in keeping with industry practice, and security of the delivery chain is more than adequate. LEM believes that the sampling procedures and handling in the field, sample preparation, sample and data security, and the analytical procedures were sufficient to maintain the integrity of the samples as representative of the material sampled.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Testwork methodology

Several series of testwork have been carried out on samples of mineralised material and flotation concentrates since 2012 by various laboratories and specialist consultants. These testwork programmes included the following:

- graphite flotation including locked cycle flotation tests;
- dewatering;
- upgrading the flotation concentrates by spheronizing;
- hydrometallurgy; and
- pyrometallurgy.

A summary of the post-2012 programmes and the laboratories which undertook the testwork are provided in Table 13-1 below:

Table 13-1: Post 2012 Testwork Programmes

Date	Tests
Aminpro 13/07/13	• Metallurgical Testwork • Front End Engineering
BGRIMM 01/06/16	• Flotation • Hydo-purification on concentrate
2016	• Grinding • spheronization
2016	• Grinding • spheronization
15/11/16	• Flake Analysis
02/03/17	• Standard Sample Analysis • Spheronization
08/05/17	• Standard Sample Analysis • Spheronization • Purification • Cell Assembly and Testing
05/01/17	• Comminution & shaping
10/10/18	• Micronizing • spheronization
02/06/20	• Produce d50 of 4µm
30/01/19	• 17 heat treatments
30/01/19	• Purification
09/02/18	• High gradient mag sep to remove iron containing minerals

Date	Tests
	• Modal composition • Elemental comp • Elemental distribution • Mineral association • Mineral locking • Particle size distribution • Mineral grain distribution
29/05/20	• Waste Characterization • Waste Classification
27/12/12	• Dewatering concentrate

All samples for the tests listed in Table 13-1 were selected and taken by Woxna Graphite, apart from those requested by Amelunxen Mineral Processing Ltd (Aminpro) which specified the core required for its test programme and the samples were then taken by Woxna Graphite personnel.

Concentrate dewatering testwork was commissioned to support the dewatering plant design. The sample was a single concentrate sample which was created using a fine graphite concentrate product sample from the historical product which is stockpiled at the Woxna Concentrator site.

Aminpro was commissioned to complete a comprehensive testwork programme with the objectives of increasing the coarse graphite flake recovery, overall plant recovery and product grade. A single point blast sample, which assayed at 3.3% C, was used at the start of the test programme until a better drillhole core sample was supplied for final testwork. The drillhole core was considered representative of the RoM but actually assayed 12.2% C which is higher than average grades even when taking into account treating the higher grade Type A. Detailed testwork was done on this sample.

BGRIMM carried out flotation testwork to produce a concentrate grade for hydrometallurgical tests. The sample used by BGRIMM was a new mining sample obtained from the 2015 test mining campaign.

The spheronization of flotation concentrate at different particle sizes (grind sizes) was assessed. The sample was a split from the sample from a previous test , a +80 µm flotation concentrate, having a d_{50} of 285 µm, and grade 92.98% C, 3.25% Si and 1.26% Al. It had a grade and particle size similar to that which potentially would be suitable for upgrading by spheronizing.

Several series of tests assessed spheronization, purification and the product's characteristics for batteries. The grade and particle size were similar to that which potentially would be suitable for upgrading by spheronizing.

Fine milling and spheronization was looked at. The sample was a concentrate which had been produced by re-floating a production concentrate sample to increase the graphite grade. The particle size distribution was 26,3% less than 63 µm, 49,9% less than 100 µm and 99,9% less than 500 µm.

Magnetic separation tests assessed whether magnetic separation could increase the grade of the concentrates. Two flotation concentrate samples were tested, one +75 µm and the other -75 µm.

Tests regarding micronizing a flotation concentrate and then spheronizing the products were performed. The prime sample was a +40 µm flotation concentrate which had been produced by re-floating an actual production concentrate sample to increase the graphite grade.

Tests at different temperatures to assess the effect on the graphite purity were carried out. Small scale tests used graphite samples from Woxna Graphite AB, and larger sample tests were carried out on one sample of a milled and shaped product from Hosokawa and two large milled and shaped samples from other tests.

The analysis of flotation concentrate, which had been upgraded by additional flotation, was carried out to assess its composition. Two samples were examined, a sample of a flotation concentrate which had been upgraded by re-floating which was then split into +38 μm and -38 μm concentrate samples.

The waste characterisation of several different types of waste from Woxna including tailings, especially regarding the potential for acid generation were assessed. The only sample used which has relevance to the process was a sample of tailings. There is no description of where this sample was taken.

Jet micronizing tests on one flotation concentrate fines sample and two spheronizing rejects to mill to d_{50} of 4 μm were carried out. The flotation concentrate was -75 μm and the spheronized rejects were a Spheronized Purified Graphite (SPG) by-product and a purified SPG by-project.

13.2 Testwork results and review

13.2.1 Dewatering test

Concentrate dewatering testwork was carried out in 2012 to support the dewatering plant design. The results of this test work indicate that thickening is not viable and that pressure filtration in horizontal or vertical filters yield filter cake moisture contents of 22% and 20% respectively.

13.2.2 Aminpro Tests

In January to June 2013 a major testwork programme was undertaken by Aminpro to reassess the process. Using the historical process as a basis, GBMMEC and Aminpro designed a test programme to explore the various optimisations considered pertinent to the plant design with the objectives of increasing the coarse flake recovery, overall plant recovery and product grade. The key tests were:

- bond Work Index on feed and assessment of regrinding parameters;
- rougher flotation to assess variables such as reagents, reagent dosage, % solids, pH and grind;
- rougher kinetic tests with screen analyses using optimum conditions;
- cleaner flotation kinetic tests including grind size, pH, % solids, redox, reagent addition points, column parameters;
- rougher flotation and mineralogy variability;
- locked cycle flotation tests;
- microscopy in the form of mineral liberation analysis;
- settling tests on tailings; and
- magnetic separation test work on graphite concentrate.

The following summarises the testwork results:

- Bond Work Index (BWI) of the prime sample was 18.5 kilo watt hour per tonne (kWh/t) and for cleaner feed material it was greater than 40 kWh/t, dependent on the graphite grade. This confirms historical reports.
- Rougher flotation tested at various grind sizes, indicated that:
    - graphite and gangue (silicates) are the coarsest components of the sample with iron sulphides (mainly pyrrhotite) being present as fines.
    - graphite floated very fast with good recoveries (>95%) in all tests. Gangue and iron sulphides had lower recoveries and flotation speeds.
    - flash flotation in roughers should be considered to capture as coarse a flake product as possible and that the finer graphite would be recovered in the scavengers.
- In testing other variables in the rougher flotation, the following was observed:
    - no effect was observed on graphite recovery and grade when varying the feed % solids;
    - no significant effect was seen in varying the pH;

    - no positive effect occurred in the testing of other reagents (MIBC was compared to: diesel; Lila Flot GS13; and Aerofroth 88).
- Contact cell tests on rougher feed gave good results with regard to recovery but failed to produce high (above 90% C) concentrate grades.
- Microscopy (Mineral Liberation Analyser) and liberation studies found that in the majority of the sample's muscovite minerals were locked with the graphite flakes, possibly preventing upgrading concentrates in the cleaner stage without further regrind. It showed that between 14% and 20% of the graphite was locked. This finding was confirmed on a concentrate sample.
- The following conclusions were reached for the cleaner flotation:
    - regrinding was necessary to achieve higher grades;
    - flotation at low densities was found very important;
    - use of dispersants was seen to help depression of gangue and
    - concentrate grades with above 90% C were achieved in a number of tests aimed at developing the cleaner circuit configuration, at the optimised conditions.
- A locked cycle test was performed at optimised flotation conditions on the second sample with a head grade of 12.2% C. The following conclusions were made:
    - the overall graphite recovery obtained was above 96%, and e produced concentrate grades above 93% C;
    - approximately 12% of the feed mass was recovered as concentrate. Of the 12%, 5.3% reported into the rougher cleaners and the rest into the rougher-scavenger cleaners.
    - the products of the locked cycle test showed that over 18% of the concentrate reported to the +250 µm size assaying 95% graphite and over 39% of the concentrate reported in the +180 µm size assaying over 94% graphite.

The assay results of the tests with particle size are shown in Table 13-2.

Table 13-2: Results of the Locked Cycle Flotation Tests

Size fraction	Rougher-cleaner concentrate		Rougher-scavenger-cleaner concentrate		Combined concentrate	
µm	% Retained	% C	% Retained	% C	% Retained	% C
+250	13.9	95	22.0	95	18.4	95
+180 -250	18.8	97	23.4	92	21.4	94
+100 -180	26.5	94	29.6	91	28.3	92
-100	40.8	89	25.0	87	31.9	88.0

Source: Aminpro, 2013

Settling and rheology tests were carried out on a tailings sample to assess the potential for thickening the final tailings before pumping to the TSF. The following conclusions were reached from these tests:

- 16 g/t of SNF 2070 flocculant is required to achieve a settling rate of 0.15 m^2.d/t;
- thickened tailings will be between 65% and 68% solids to allow pumping by horizontal type pumps.

13.2.2.1 Conclusions

This was a comprehensive test programme primarily using a sample, the second 12.2% C sample, selected to represent the mineralised material at Woxna. This sample's grade is higher than the expected feed to the concentrator, which is approximately 10% C. This sample could be Type A which has higher grades and possibly larger graphite flakes than type B.

The test programme achieved an increase in the coarse flake recovery, the overall plant recovery and final concentrate grade compared to historical records, that is pre-2013. The overall concentrate carbon recovery is 96% and grade is 91.7% C.

13.2.3 BGRIMM test programme

BGRIMM carried out a series of tests to find the parameters for producing a high-grade flotation concentrate. One sample was used for all tests. The flotation tests included:

- analysis of the feed;
- grinding fineness test for rough flotation;
- water-glass dosage test for rough flotation;
- lime dosage test for rough flotation;
- diesel (collector) dosage test;
- frothier type and dosage tests;
- flotation time test;
- test on first-stage regrinding fineness;
- test on lime dosage for cleaning flotation after first-stage regrinding;
- open-circuit flotation; and
- locked-cycle flotation.

The following summarises the testwork results:

- The head grade was 10.89% Total C. 98.44% of the C was in the form of graphite
- Total Fe was 9.26%, SiO_2 47.68%, Al_2O_3 12.58%.
- The carbon recovery increases initially but then decreases in the rougher. The grade of the rough concentrate increases with the increasing grinding fineness. The optimum grinding is 55% passing 150 µm.
- Water glass is not required.
- Lime addition for rougher flotation was beneficial.
- There was an increase in carbon recovery with increasing diesel (collector) dosage, but for more than 120 g/t, the increase is marginal.
- In the first three minutes of the rougher flotation the carbon recovery was high and in 6 minutes the carbon recovery reached 94%. Increased flotation time produced only marginal increases in recovery.
- Grade of concentrate increases with regrind fineness. Finer than 77% passing 150 µm the increase become negligible. Note that the locked cycle final concentrate had 90.8% passing 150 µm
- The locked cycle flotation test involved a rougher stage with two scavenger stages, and 10 regrind/cleaner stages with cleaner tailings being recycled, and a final two cleaner stages. Table 13-3 shows the grades and recoveries. The overall grade was 93.19% and the recovery 93.74%.

Table 13-3: Grades and Recoveries of the BGRIMM Locked Cycle Test

Size fraction µm	% Retained	% C	Recovery %
Concentrate	10.75	93.19	93.74
Tailings	89.25	0.75	6.27
Feed	**100**	**10.69**	**100.00**

Source: BGRIMM, 2016

Table 13-4: Size Fractions for the BGRIMM Locked Cycle Test

Size fraction µm	% Retained	% C	Recovery %
-300 + 150	9.28	95.36	9.50
-150 +74	32,99	94.15	33.35
-74	57.73	92.18	57.15
Total	**100**	**93.12**	**100.00**

Source: BGRIMM, 2016

- The test to assess improving the grade of the concentrate by magnetic separation showed that little improvement was possible.
- The impurities in the concentrate are mainly made up of mica (muscovite, silicate, biotite), clay minerals, pyrite, pyrrhotite, limonite, quartz, chlorite, etc. The grain sizes of these impurities are mostly under 50 µm. The chemical analysis of the concentrate showed that the main contaminants were 2.95% S, 1.47% total Fe, 0.51% SiO_2 and 0.23% Al_2O_3.

13.2.3.1 Conclusions

This was a comprehensive test programme using an undefined sample which had a similar head grade as the expected feed grade to the flotation plant.

The locked cycle flotation test used an extended series of regrind/cleaner flotation stages to achieve a high-grade concentrate. The proportion of the +150 µm fraction in the concentrate was relatively low, presumable due to BGRIMM trying to maximum the concentrate grade. However, the fines, -74 µm, had a relatively high grade. This compares to Aminpro and historical production records which indicate that less than 90% C in this fraction is more normal.

13.2.4 Spheronizing testwork

The first set of tests were regarding flake analysis testing. The conclusions were: sample showed low reversible capacity, which is likely due to impurities, the purity of the sample was low, good crystallinity, and the main conclusion being that the sample would most likely perform better after some purification.

The second set of tests assessed the physical, composition and electrochemical properties. Spheronization with analysis of the products was then carried out. A spheronized sample was sent to another laboratory, for purification and analysis of the purified sample was then performed by the laboratory. The conclusions were:

- Average of approximately 70% product yield for basic flake spheronization is a high percentage return.
- The spheronized material is within the targeted range, particle size of 10-30 µm (target: 10-30 µm).
- Key properties of purified sample:
  - surface area of 4.75 m^2/g;
  - tap density of 0.95 g/cc; and
  - reversible capacity of 364 mAh/g.

A third test programme was conducted. The results after purification were:

- tap density 0.95 g/cc;
- BET average 4.50 m^2/g;
- main impurities were Cl (12 parts per million (ppm)), with total impurities of 12 ppm;
- electrode properties loading 20.82 mg/cm^2;
- calendered density 1.441 g/concentrate; and
- electric vehicle and high-power specification cells performed well for natural graphite.

13.2.5 Spheronizing testwork on a flotation concentrate

Spheronizing tests on a sample of flotation concentrate were carried out. The first step was grinding to an optimal size distribution before shaping was performed. Five grinds were carried out. For each grind two spheronizing and fines separation were performed. Characterisation was performed.

The milled product was classified to a d_{50} of 17.5 µm before being spheronized. Spheronizing produced achieved a yield of 57.7% and d_{50} of 20.6 µm and a BET of 5.77 m^2/g. The fines produced by spheronizing had a d_{50} of 9.9 µm.

A trial using previous settings was run. This included grinding to optimal size for shaping, shaping and fines separation and characterisation.

The milled product was classified to a d_{50} of 17.1 µm before being spheronized. Spheronizing produced achieved a yield of 62% and d_{50} of 22.2 µm and a BET of 4.98 m^2/g. The fines produced by spheronizing had a d_{50} of 9.75 µm.

The conclusions were that the size distribution of the spheronized material matched that of previous tests, SEM shows that the material appears to be smooth and spheronized and that spheronized size control of 17.3 µm to 25.5 µm can be achieved.

13.2.6 Production parameter testwork

A graphite flotation concentrate sample was assessed to determine the production parameters for spherical graphite products in the range d_{50} of 15 µm to 25 µm using a rotary mill.

The sample was analysed at a carbon content of 92.9% on average. SEM micrographs of the flake material show predominantly thin graphite flakes. Mica was found to be present on the graphite flake surface and intercalated in between the graphite layers. XRD analysis evidences the presence of minor amounts of muscovite, chlorite and quartz.

Processing parameters for the production of spherical graphite in the range of 16 µm to 23 µm were determined to support the future scale up and cost estimation for an appropriately sized pilot plant. Based on these data, scale up and further product optimisation is possible. These parameters also serve as processing parameters for a sample production or a basis for further research and development tests with the pilot plant. The results of these test were also used to provide sizes and cost estimates for full size units for spheronizing the whole graphite flotation concentrate. The test results are given in Table 13-5.

Table 13-5: Results of spheronization

SPG Product	Tap density g/cm^3	d_{50} µm	Ratio d_{90}/d_{10}	BET	Yield
15 µm	0.98	16.4	3.4	8.7	42.8
20 µm	0.96	17.8	3.6	7.7	49.9
25 µm	0.99	22.6	4.5	6.7	56.7

Source: Laboratory, 2018

Comments were:

- All three spherical graphite products (15 µm, 20 µm, 25 µm) generated a tap density above 0.95, which is well within the range of typical spherical graphite products.
- BET values for all spherical graphite products compared well to reference materials.
- For the 15 µm and 20 µm products a ratio of about 3.5 was reached, which compares well to typical values of reference products.
- The ratio d_{90}/d_{10} is 4.5 for the 25 µm, slightly higher than reference products (< 4). Improvement of the ratio of d_{90}/d_{10} is likely to be achieved by further adjusting the air classifying regime.

The main quality parameters of spherical graphite products are tap density, shape of particles, specific surface area (BET), particle size distribution (PSD; characterized by ratio of d_{90}/d_{10}) and the purity of graphite.

The ratio of d_{90}/d_{10} gives an indication of the steepness of the slope of the PSD. For spherical graphite, a ratio <4 for finer spherical graphite products and <3 for coarser spherical graphite products is typical, ensuring a narrow PSD.

13.2.7 Testwork to assess milling and shaping

Tests were carried out to mill the flotation concentrate sample to increase the bulk density and to produce a round shape of the particles with d_{50} of 15 µm. A classifier mill was used with a cyclone before the filter. Different classifier heads were used for milling and shaping.

The conclusions were that the results were commiserate with its experience. Milling produced d_{50}'s between13.8% and 15.6% and spheronized products with d_{50}'s oversize between 13.5% and 14.6%.

13.2.8 Heat treatment testwork

Graphite samples were held at different temperatures for specified times to assess the purification of the graphite. A total of 17 heat treatments were carried out and no problems were reported. The carbon assay for each test is shown in Table 13-6.

Table 13-6: Results of heat treatment

Time at temperature	Temperature °C					
Hours	1800	2000	2200	2400	2600	2700
2					99.950	99.994
4					99.995	99.998
6		98.270	98.610	99.973	99.996	99.998
8	96.830	98.350	98.990	99.986	99.997	
12		98.330	99.470			

Source: Laboratory, 2019

It was reported that yield after heating was of the order of 80%. These tests indicated that to achieve grades of better than 99.99% C either holding times of 2 hours at a temperature of 2,700 °C was required or 4 hours at 2,600 °C.

On the larger sample tests, the fines graphite samples were run to a set temperature and time with little or no unexpected issues.

The large sample test with spheronized graphite was successful, with a yield comparable to the small sample tests. LEM reported that for the spherical sample a grade of 99.95% C for a period of 2 hours and 99.995% C for a period of 4 hours in the furnace were produced.

Peter Young, of LEM, wrote a report on tests carried out on a flotation concentrate sample which was purified at 2,600°C for 4 hours. The result was a grade of approximately 98.7% C.

13.2.9 Magnetic Separation Tests

Magnetic separation tests were carried out on two concentrate samples, one with a particle size greater than 75 µm and another less than 75 µm to find out if it is possible to use the magnetic properties of the impurities to increase the grade of the graphite concentrate. A wet Low intensity magnetic separator (WLIMS) was able to remove highly magnetic minerals from both samples. A high gradient magnetic separator (HGMS) was then used to assess whether the graphite tailings from the wet low intensity magnetic separator (WLIMS) tests could be further upgraded. It showed that iron and sulfur impurities reported to the magnetic fraction, hence producing an improvement in grade. The conclusion was that the mass pull to the magnetic fractions was not sufficient to increase the graphite concentrate grade significantly. But it also stated that that it might be possible to improve the grade by using a stronger magnetic field and/or lower volumetric flow.

13.2.10 Tests regarding environmental impacts

Potential environment impacts were assessed. The only one directly relevant to the design of the process flow was a tailings sample. Its characteristics were evaluated especially regarding the potential for producing acid drainage. The sample varied between sand-fine sand and silt-coarse silt. The tailings consisted of the quartz, feldspar, mica, calcite and graphite. The predominant sulfide mineral was magnetic pyrrhotite. There was also pyrite and sphalerite. The sulfur concentration in the uncontaminated tailings sample was 3.88%.

13.2.11 Micronizing tests

Tests to evaluate whether it is possible to mill graphite concentrate to a specific particle size distribution, d_{90} passing 8 µm and d_{50} passing 4 µm were carried out. The tests showed on the that it is possible to micronize all three feed materials, flotation fines, spheronized by-product and purified spheronized by-product, into this size range. In practice it would be easier to put all feed material through the mill/classifying unit, to get one product only and to avoid complicated material streams. No test or operating parameters were given.

13.2.12 Coating

Coating testwork has been done to demonstrate the Woxna products can be treated to produce marketable products.

13.3 Recovery estimates

13.3.1 Flotation recoveries

Two locked cycle flotation test programmes were undertaken with different aims;

- The 2013 Aminpro programme was to produce concentrates with as large a flake size as possible and hence employed few regrind stages; whilst
- The 2016 BGRIMM programmes aim was to produce as high a grade concentrate as possible for downstream processing and hence used many regrind and cleaner flotation steps.

Aminpro produced a concentrate grading 91.7% C and BGRIMM's concentrate was 93.19% C with a 93.74% recovery.

The Aminpro's concentrate was coarser than BGRIMM's.

The particle size distribution of the flotation concentrate is not critical as all the flotation concentrate will be sent to the VAP to be spheronized and micronized down to below 20 µm. In addition, since the upgrading will include thermal treatment to remove all impurities, the concentrate grade is not specifically crucial either. The concentrate grade cannot be improved sufficiently by flotation or gravity methods to influence the furnace type or associated upgrading specifications.

Therefore, the BGRIMM results have been used for the estimate of the graphite recovery and grade produced by flotation.

13.3.2 Spheronizing recovery

Tests have been used to the estimate the yield for producing a d_{50} of 15 µm. 46% of the feed will report as spheronized product.

13.3.3 Thermal purification recovery

The thermal tests have been used as the basis for the thermal purification of the spheronized graphite, that is 2600 °C is required for approximately 2 hours to produce 99.95% C.

14 MINERAL RESOURCE ESTIMATES

Historical Mineral Resource estimates have been developed by various parties and at different points in the development of the Woxna graphite deposits. The historical estimates have been provided in Section 6.1.1.4 and for the purposes of this 2021 PEA have been superseded by the current estimates.

The historical and previous Flinders 2015 Mineral Resource estimates were developed without the constraint of an applied mine plan and open-pit shell. In the light of more rigorous compliance requirements, the Mineral Resources were reported by ReedLeyton within the constraints of the 2021 PEA mine plan as a means of demonstrating “reasonable prospects for economic extraction” as required by numerous international reporting codes. No new exploration data was included in the re-estimation process.

14.1 Kringel – Mineral Resource estimate

14.1.1 Drilling data

The disclosure pertaining to the historical and 2012 drilling campaigns on Kringel has been provided in Section 6.1.1and Section 10.1

Ninety-two DDs totalling 6,581-metres, were drilled into the Kringel deposit in 1988, 1989 and 2012 (see Table 14-1). The geological information provided by these 92 drillholes was included in the 2021 Mineral Resource estimate albeit that 44% of the DDs are considered historical with no information regarding the drilling protocols applied at the time. Nonetheless, the Qualified Person reviewed the drillhole collar positions in the field, interrogated the historical database, examined the drillhole data in the light of historical production information and undertook a sampling verification exercise and is satisfied that this geological information can be relied upon in a mineral resource estimate.

Table 14-1: Combined Kringel drilling database summary

Hole Type	Drill Series	Drill Number	Drill Meters	Resource Intersection Meters
DD	88	28	1,595	512.2
DD	89	23	1,313	284.9
DD	12	41	3,673	960.5
Total		**92**	**6,581**	**1,757.6**

Source: ReedLeyton 2012, 2013, 2015

14.1.2 Database integrity

The verification process applied to the Kringel drillhole database has been provided in Section 12.1.

Digital data has been both randomly and systematically checked by ReedLeyton and shown to be correct using a number of checks as discussed in Section 12.1 The digital data was imported directly into Microsoft Excel, validated in Microsoft Access and exported into a csv format. The database was then imported into Maptek Vulcan software in the csv format and visual check applied in the form of cross-section generation and drillhole cross referencing.

14.1.3 Drillhole orientation

The Kringel drilling campaign was conducted on a 50 × 50 m grid and drillholes drilled mostly at two orientations 12^0-degrees and 348^0. For wire framing purposes the strike direction of the mineralisation varied between 80^0 to 100^0 and a 90^0 strike was considered the optimal orientation.

14.1.4 Chemical analysis

Core produced from the 1988 and 1989 campaign was sampled and analysed by Leco Analyser and ICP method at the laboratory of SGAB ANALYS, Luleá, Sweden. The method applied by was the standard for the industry of the day, and although no quality assurance data is available, it is considered to be of a very high quality.

Core produced from the 2012 campaign sampled and analysed by Leco Analyser and ICP method at the laboratory of ALS Chemex, Pitea. Sweden. The method applied is to current industry standards, and although no standard data is available, it is considered to be of a very high quality.

A total of 374 samples from the 51 1988 and 1989 drill holes and total of 1,433 samples from the 41 2021 drill holes were analysed in total for the Kringel for diamond drillholes within the current resource estimation.

14.1.5 Sample length

All DDs drilled at Kringel were sampled with an average of 1 m intervals. Composites of the drillhole samples and assays were generated using Maptek Vulcan software with run lengths of 1 m.

These composites honour the geological wireframes and verification was undertaken by generating an Isis (Vulcan Database) file and visually inspecting the result of the composite.

Specific components of the compositing include:

- Run Lengths of 1 m;
- Data Field C_pct was composited; and
- the composite file was then applied a tag for each composite with the character (a2,a3,a6,a9,a13,a14, c01-c03, c05-c06, c08-c19) in the 'bound' column. This new composite isis file was called viersa.cmp.isis and used in the estimation process.

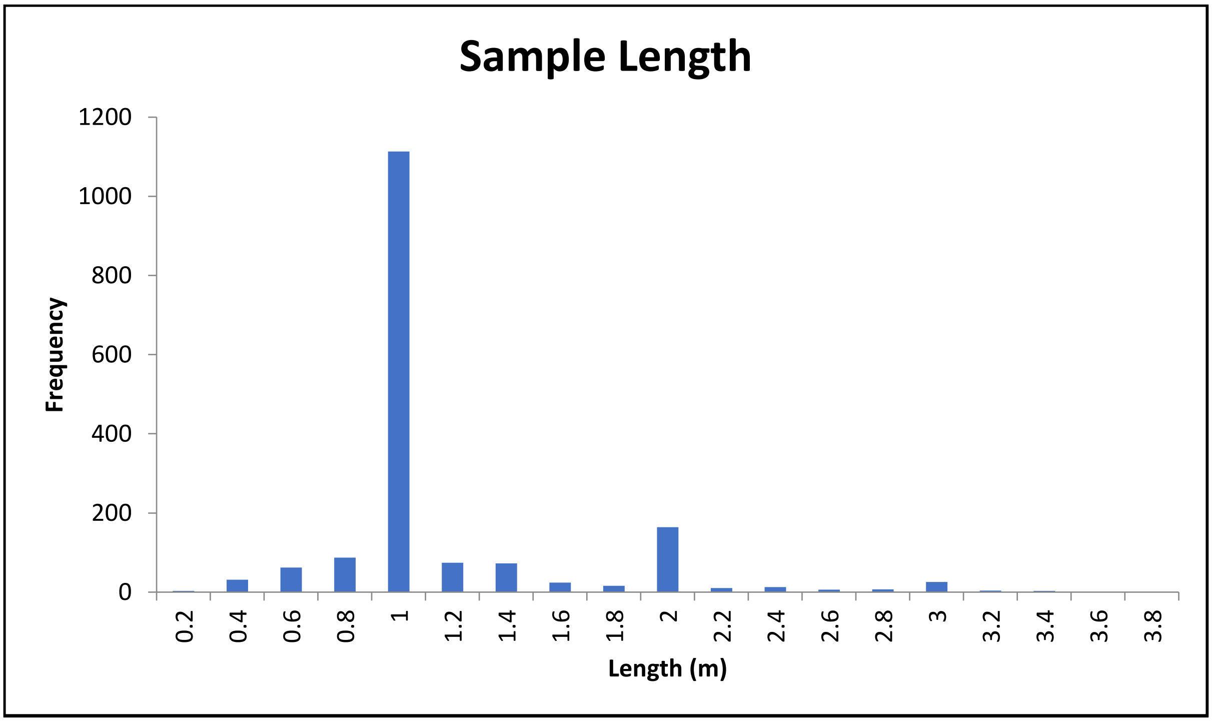


Figure 14-1: Histogram of raw sample lengths for Kringel

14.1.6 Density

Using the *bulk density (BD*) density default function of Vulcan, the variable *BD* was populated. Note that this is not true bulk density but a software nomenclature for normal density as per the description provided in Section12.3.

The value *BD* 2.7 was run according to density testwork by Flinders previously attributed to various assays within the geology database. ReedLeyton has created a file with an average *BD* taken between various Cg % grades within the resource and waste blocks outside the resource.

14.1.7 Geological model and block model

The geological model constructed for the Mineral Resource estimation by ReedLeyton was created using the following parameters and criteria:

- the mineralisation is generally tabular in nature conforming with the host sequences in the region ;
- a strike bearing of 90^0 for the mineralisation;
- a single block model was constructed with a parent block size of 5 × 25 × 5 m with sub blocks at 1.25 × 5 × 1.25 m. An offset of 1500 × 600 × 400 m was applied;
- an estimation area of an area approximately 1,200 m by 100 m to 200 m (see Figure 14-2);
- intersection of mineralisation in all drillholes suggestive of continuous mineralisation throughout the area;
- depth determination of 150 m below surface;
- mineralisation dip of 60^0 to 90^0 S;
- mineralisation thickness varying between 5 m and 15 m averaging 15 m thickness;
- the mineralisation grade distribution is not uniform and higher grade mineralised envelopes or bodies were identified according to cut-off grades;
- six main higher grade Type A zones were identified with a cut-off grade of 7% Cg;
- an outer, lower grade Type B domains were identified within which 11 small mineralised envelopes/bodies exist;
- faulting is present although no geological loss was determined;
- the mineralisation remains open laterally and at depth.

The variables include the type and their default values before estimation.

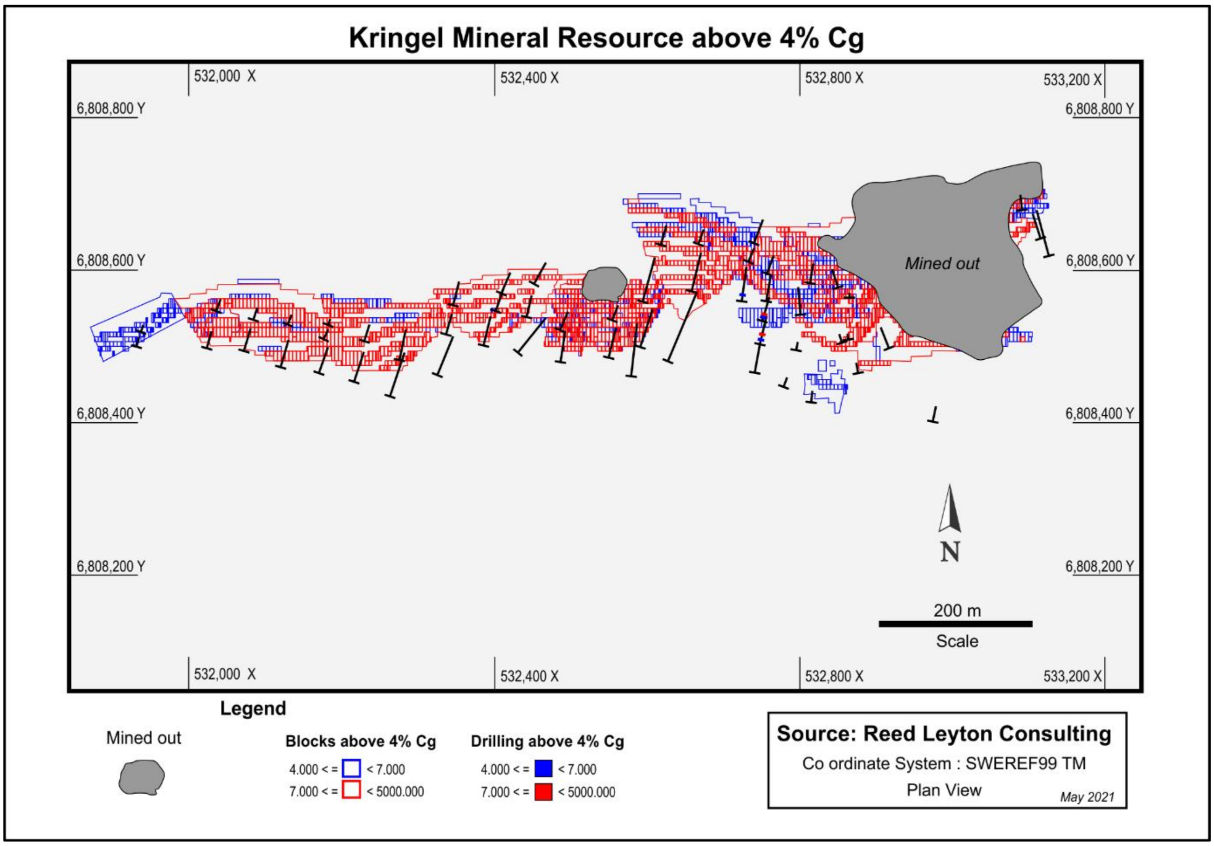


Figure 14-2: Mineral Resource – Kringel

14.1.8 Wire framing

Wire framing of the geological boundaries was performed by joining digitised section outlines at a 50 m spacing. The digitised sections were snapped to drillholes within ±25 m influence using above 7% Cg for the six Type A grade wireframe domains at Kringel.

Vertical plane sections were digitised at 12^0 and 348^0 degree orientation at a 50 m spacing. There is sufficient evidence for continuity of the mineralised envelope between sections.

All modelled wireframes were checked in plan, cross section, long section, and 3D rotated views

All geological wireframes were checked for crossing, inconsistencies, and closure.

All wireframes were updated to match the new drillhole collar coordinates and the adjusted mineralisation data points.

Table 14-2: Kringel domain volume validation

Domain	Wireframes Volume	Model Volume	All Domains
Total	**4,802,886**	**4,745,191**	**99%**

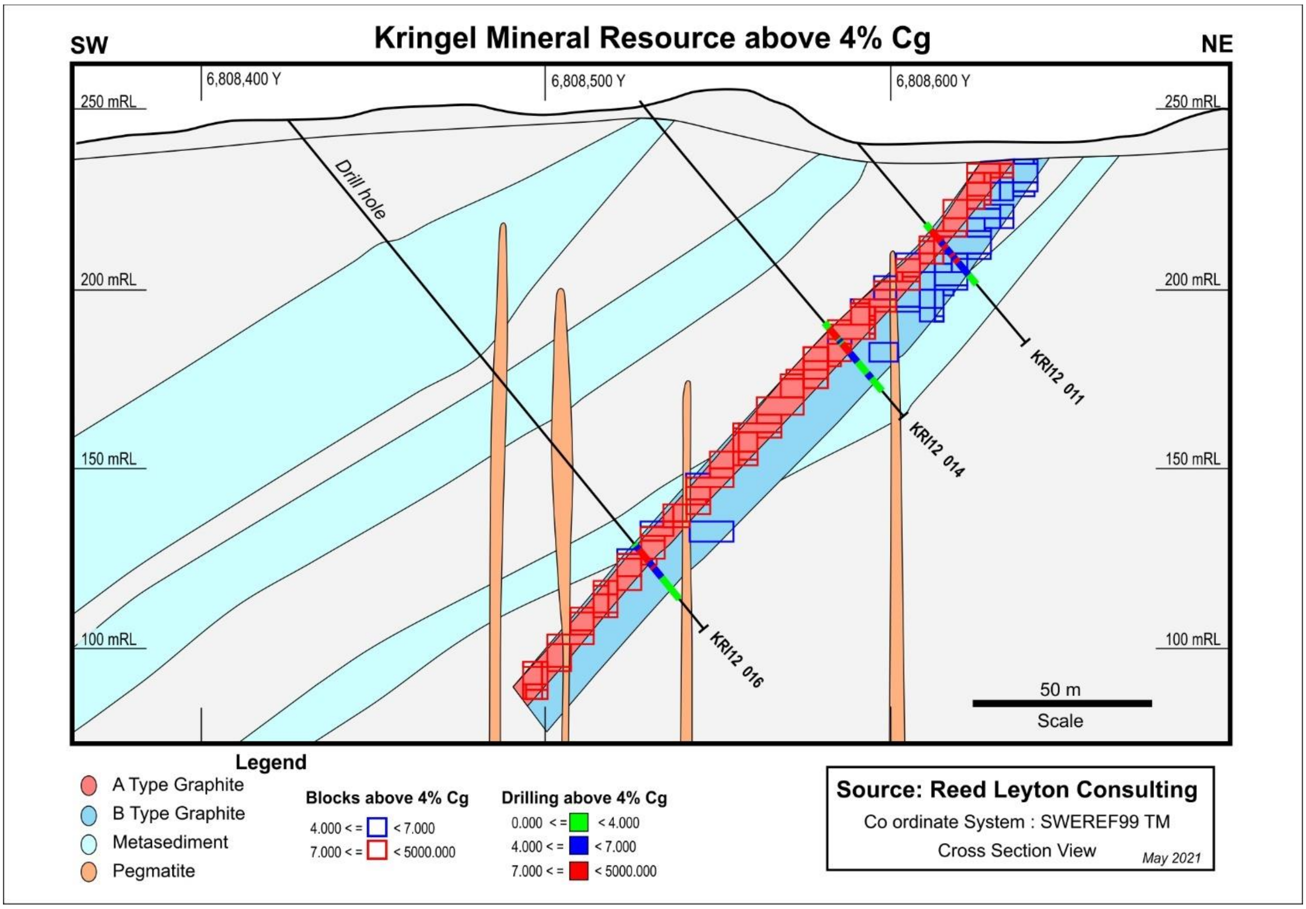


Figure 14-3: Mineral Resource cross section - Kringel (looking NW).

14.1.9 Grade interpolation

Grade interpolation was undertaken using inverse distance defined by the domain wireframes. The allocations of composites were calculated using a hard boundary at the domain wireframes.

Using Maptek Vulcan's Estimation Editor the grade estimation was run for Kringel. Variables were populated using single search ellipses with no cut-off to the mineralisation domains.

Constant parameters used in this block estimation file include:

- The grade variable populated was C_uncut. The default given was 0.
- The number of samples used was populated in the variable numsam. The default given to this variable was 0.
- The number of drill holes used was stored in nodrill. The default given was 0.
- The sample distance used was stored in the variable samdis.
- The weighted average anisotropic distance to the samples used was populated in the variable samdis.
- The inverse distance method was applied.

Table 14-3: Block Model Parameters for Kringel

Variables	Description
c_uncut	Carbon grade - reportable
s_uncut	Sulfur grade – not reportable
nnp_uncut	Nnp grade – not reportable
bd	Bulk Density

Variables	Description
category	Resource category by script
mintype	Mineralisation Domain
nodrill	Number of Drill holes
samdis	Average sample distance
numsam	Number of samples
pass	Estimation flag
type	Air or fresh rock
mined	Mined or in situ
lithtype	Graphite Pegmatite Metasediment Overburden
rsc_cat	Final Resource Category meas = 1, ind = 2, inf = 3, additional min = 4.

Table 14-4: Search Parameters for the Kringel deposit

Pass	Min Sample	Max Sample	Distance
1	2	12	70
2	1	20	140
3	1	30	400

Table 14-5: Estimation Parameters for Kringel deposit

Domain	Strike	Plunge	Dip	Major	Semimajor	Minor	Discretisation
Type A	82	0	0	4	2	1	2x:4y:2z
Type B	82	0	0	4	2	1	2x:4y:2z

14.1.10 Minimum width

No minimum width has been applied in the estimation of the Kringel Mineral Resource estimate.

14.1.11 Cut-off Grade

A grade cut-off of 7% Cg has been applied to the Mineral Resource estimation modelling to define Type A Graphite. A grade cut off of 4% Cg has been applied to the Mineral Resource estimation for reporting purposes of Type A and Type B Graphite.

14.1.12 Additional Variables

Once the estimations had run, a number of additional variables were added or calculated. These variables included:

The category variable, category. A script, resourcecatflagged.bcf was run on the block model. This script looked at the nearest neighbour distance variable ("samdis"). If samdis was >0, then the category variable was set to inf (inferred). This variable was used to classify the resource.

The category variable, called rsc_cat. A calculation, rev_rsccat1_2_3_4.bcf was run on the block model. This calculation looked at the previous script run. This variable was used to classify the resource based on drilling density, continuity and general confidence in each modelled wireframe.

Using the BD density calculation function of Vulcan the variable bd was populated. The script was run according to density test work by the Issuer previously attributed to various assays within the geology database. ReedLeyton has created a script file with an average BD taken between various C grades.9

14.1.13 Mining and Metallurgical Assumptions

MPlan 2021 prepared constraining pit shells for ReedLeyton to be used for Mineral Resource Estimation reporting of the Kringel deposit using optimised pit shells generated using Datamine's NPVS software.

The key assumptions used in the generation of the resource constraining pit shells for the Kringel deposit were:

- overall slope angle for resource pit shell: 55 degrees;
- mill cut-off grade = 4.00%;
- break even cut-off grade = 4.21%;
- process cost: 84.18 USD/t mill feed
- dilution 2.5%
- mining recovery 97.5%
- process recovery 93.7%

The cut-off grades assumed:

- graphite price of: 2,320 USD/t;
- recovered value of 2,103 USD/t after applying costs, taxes, mining, and process recovery factors; and
- mining cost: 4.51 USD/t rock mined

14.1.14 Kringel Mineral Resource Estimate

14.1.14.1 Previous Kringel Mineral Resource estimate

This Mineral Resource estimate (Table 14-6) has been prepared by ReedLeyton in accordance with the CIM Definition Standards of June 2011. The classification of the resource at the prescribed levels of confidence is considered appropriate by the Qualified Person on the basis of verified drilling results and spacing, sample interval, geological interpretation, verified analytical integrity and past production.

The Kringel Mineral Resource, quoted to the appropriate level of confidence, is provided in Table 14-6:

Table 14-6: Kringel Mineral Resource estimate (2013)

Classification	Tonnes (Mt)	Grade Cg %
Measured	0.99	10.68
Indicated	1.86	10.63
Total	2.85	10.65

Source: ReedLeyton 2013

Reported according to CIM Definition Standards 2011
No mining parameters applied
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.7 t/m^3 applied to Graphite Domains.
7% Cg cut-off grade applied for reporting purposes;

The above numbers are literal, whereas the accuracy of the techniques requires that the estimates' parameters should actually result in figures rounded down to better reflect the order of accuracy. Hence ReedLeyton has rounded down the mineralisation tonnage to the nearest ten thousand tonnes. The resource estimates then become as shown on Table 14-7.

Table 14-7: Kringel Mineral Resource estimate, (2013 - rounded)

Classification	Tonnes (Mt)	Grade Cg %
Measured	1.0	10.68
Indicated	1.8	10.63
Total	2.8	10.65

Source: ReedLeyton 2013

Reported according to CIM Definition Standards 2011
No mining parameters applied
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.7 t/m^3 applied to Graphite Domains.
7% Cg cut-off grade applied for reporting purposes;

14.1.14.2 Current 2021 Mineral Resource estimate

ReedLeyton undertook an update of the previous 2015 Mineral Resource estimate in order to constrain the global resource within the open-pit shell designed by MPlan (2021). The application of this pit-shell provides reassurance with regard the current prospects for economic extraction.

Table 14-8: Kringel Measured and Indicated Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Measured	0.96	9.21
Indicated	1.65	9.09
Total	2.61	9.13

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.7 t/m^3 applied to Graphite Domains.
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

The above numbers are literal, whereas the accuracy of the techniques requires that the estimates' parameters should actually result in figures rounded to better reflect the order of accuracy. Hence ReedLeyton has rounded the mineralisation tonnage to the nearest ten thousand tonnes. The resource estimates then become as shown on Table 14-7.

Table 14-9: Kringel Measured and Indicated Mineral Resource estimate, (2021 - rounded)

Classification	Tonnes (Mt)	Grade Cg %
Measured	1.0	9.2
Indicated	1.6	9.1
Total	2.6	9.1

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.7 t/m^3 applied to Graphite Domains.
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

Table 14-10: Kringel Inferred Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	0.39	8.72
Total	0.39	8.72

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.7 t/m^3 applied to Graphite Domains.
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

The above numbers are literal, whereas the accuracy of the techniques requires that the estimates' parameters should actually result in figures rounded to better reflect the order of accuracy. Hence ReedLeyton has rounded the mineralisation tonnage to the nearest ten thousand tonnes. The resource estimates then become as shown on Table 14-7.

Table 14-11: Kringel Inferred Mineral Resource estimate, (2021 - rounded)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	0.4	8.7
Total	0.4	8.7

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m3 applied to in situ, then Density of 2.7 t/m3 applied to Graphite Domains.
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

14.1.14.3 Discussion

The previous Kringel Mineral Resource estimate (Flinders 2015) only reported Type A graphite above a 7% Grade cut off and the Kringel Mineral Resource estimate was also modelled using a 7% Grade cut off without any applied mining parameters. The estimation resulted in Measured and Indicated Mineral Resources being reported.

The Current Kringel Mineral Resource estimate now reports Type A and Type B graphite to a 4% Grade cut off. The Kringel Mineral Resource estimate is now constrained by the Mplan 2021 pitshell for reporting purposes. The estimation resulted in Measured, Indicated and Inferred Resources being reported.

14.2 Mattsmyra – Mineral Resource Estimate

14.2.1 Drillhole data

The historical drilling and analytical information is provided in Section 6.1.2 and Table 6-5 and the process of verification discussed The data included in the Mineral Resource is summarised below:

- thirty three (33) DD drillholes totalling 2,690 m, drilled into the Mattsmyra concession area in 1983,1989, 1990 and 1992. Twenty nine (29) diamond drillholes were included in the current mineral resource estimation;

- data relating to the collar locations, drill collar orientations were sighted by the ReedLeyton in sections and plans of the day;
- the profile spacing is approximately 50 m and distance between holes on section is generally 25 m. 70% of the holes are dipping 50 degrees, the remaining are dipping 60 degrees. Drillhole lengths average around 80 m, resulting in a vertical depth test of around 60 m. The maximum drillhole length was the first hole drilled in 1983 at 158 m hole depth. Shorter holes were drilled where the graphite was intersected close to surface.
- no drillholes have been deviation surveyed to date. The start azimuth was measured with a hand held compass. Any uncertainty in drillhole trend cause by the lack of surveys is considered minor at the spacing of the drillholes and relatively short hole length in relation to a scale of the resource.
- the position for Mattsmyra drillholes has been calculated by converting the historic local grid into coordinates of RT90 national grid and are assumed accurate.
- all drill core for 33 DD drillholes has been located by Woxna Graphite staff in Woxna, Sweden. Core from 8 holes has been inspected by the Qualified Person.
- for wire framing purposes 150 degrees strike was considered the optimal orientation. Strike of mineralisation varied from 135 degrees to 155 degrees. Polygons were created every 50 m through the 29 DD drillholes.
- Drillholes have been drilled mostly on two orientations of 40 degrees and 45 degrees, average of 47 degrees with a maximum orientation of 75 degrees and a minimum orientation of 30 degrees.
- due to the amount of drilling and orientation, the true thickness is generally considered to be 60%-70% of drilled thickness.
- the likelihood that mineralisation is developed in an orientation other than that interpreted is considered to be low.

14.2.2 Database integrity

- Capture of digital data was completed by the Issuer's staff. Hard copy data has been verified and all data is stored in a database and managed by Woxna Graphite;
- drilling data from drilling programmes were transferred in digital format by the Woxna Graphite staff. Digital data has been both randomly and systematically checked by the Qualified Person and shown to be correct Assay data in original laboratory sheets has not been sighted from the 1983, 1989, 1991, 1992 drilling programme;
- the digital data was compiled directly into Microsoft Excel by the Issuer, validated in Microsoft Access and exported into a csv format. The database was then imported into Maptek Vulcan software in the csv format;

14.2.3 Chemical Analysis

- A total of 390 samples were analysed in total and included in the current Mineral Resource estimation. Drilled diamond core was sampled and analysed by Leco analyser and ICP method at the laboratory of SGAB ANALYS, Luleá, Sweden. The methodology applied by was the standard for the industry of the day, and although no quality assurance data is available, it is considered to be of a very high quality.

14.2.4 Sample length

- all drillholes drilled at Mattsmyra were sampled with an average of 2.4 m intervals. Check sampling used identical sample intervals.
- composites of the drillhole assays are generated using Maptek lengths of 3 m (Figure 14-4);
- these composites honour the geological wireframes. Checking was undertaken by generating an Isis file and visually inspecting the result of the composite.

- specific components of the compositing include: run Lengths of 3 metre; data field C_pct and S_pct was composited; and the composite file was then applied a tag for each composite with the character (hg1,lg1) in the 'bound' column. This new composite isis file was called vierscmat3.cmp.isis and used in the estimation process.

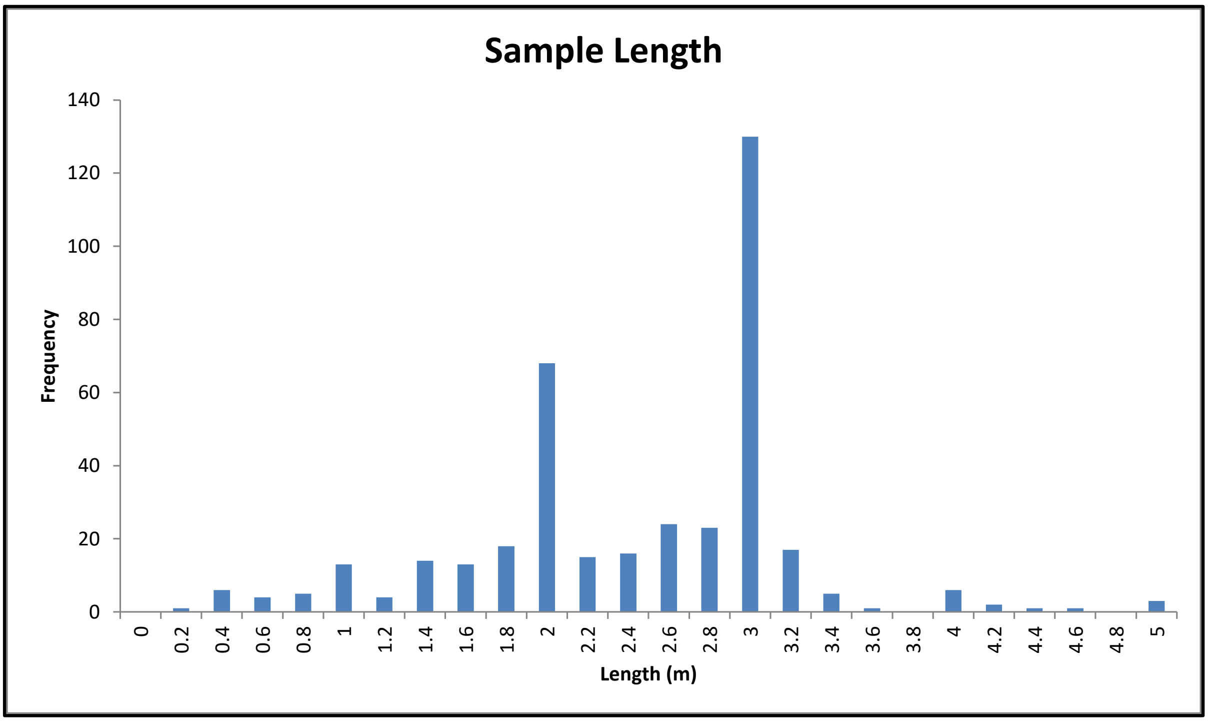


Figure 14-4: Histogram of raw sample lengths - Mattsmyra

14.2.5 Density

Using the *bulk density (BD)* density default function of Vulcan, the variable *BD* was populated. Note that this is not true bulk density but a software nomenclature for normal density as per the description provided in Section12.3.

The in situ value *BD* 2.7 was run according to density testwork by Flinders previously attributed to various assays within the geology database. ReedLeyton has created a file with an average *BD* taken between various Cg % grades within the resource and waste blocks outside the resource.

The density value 2.82 for Type A Graphite and the value 2.86 for Type B Graphite was run according to density testwork by Flinders previously attributed to various assays within the geology database. ReedLeyton has created a file with an average BD taken between various Cg % grades within the resource and in situ blocks outside the resource.

14.2.6 Geological Model

- Mineral Resource have been estimated by ReedLeyton on a bearing of 150-degree strike.
- The deposit was drilled within an area approximately 2,000 × 100 metres.
- The mineralisation was intersected on all the drilling sections and is so far known to at least a depth of 180 m below the surface.
- Mineralisation strikes NW-SE, and dips varies between 70 and 80 degrees to the SW.
- Mineralisation is present as four main mineralised bodies with two grade domains defined using. A single grade domain was defined using a cut-off grade of 7% Cg for Type A Graphite. Type B Graphite was

defined using the geological limit of the Graphite. The thickness in the section of the plane was usually more than 23 m, but varied between 8 m and a little more than 155 m.

- A single block model was constructed. The parameters used are located in the setup file for Mattsmyra.
- The block model was created using the one bdf file. This original block model contained only default values except for the variable domain, which was populated in relation to the wireframes in which the blocks resided in.
- A Block rotation of 150 Bearing, 0 Plunge and 0 Dip was applied.
- Parent block size was 5 × 25 × 5 metres, with sub blocks at 1 × 5 × 1 metres.
- An offset of 2600 × 1,100 × 400 metres was applied
- The variables include the type and their default values before estimation.

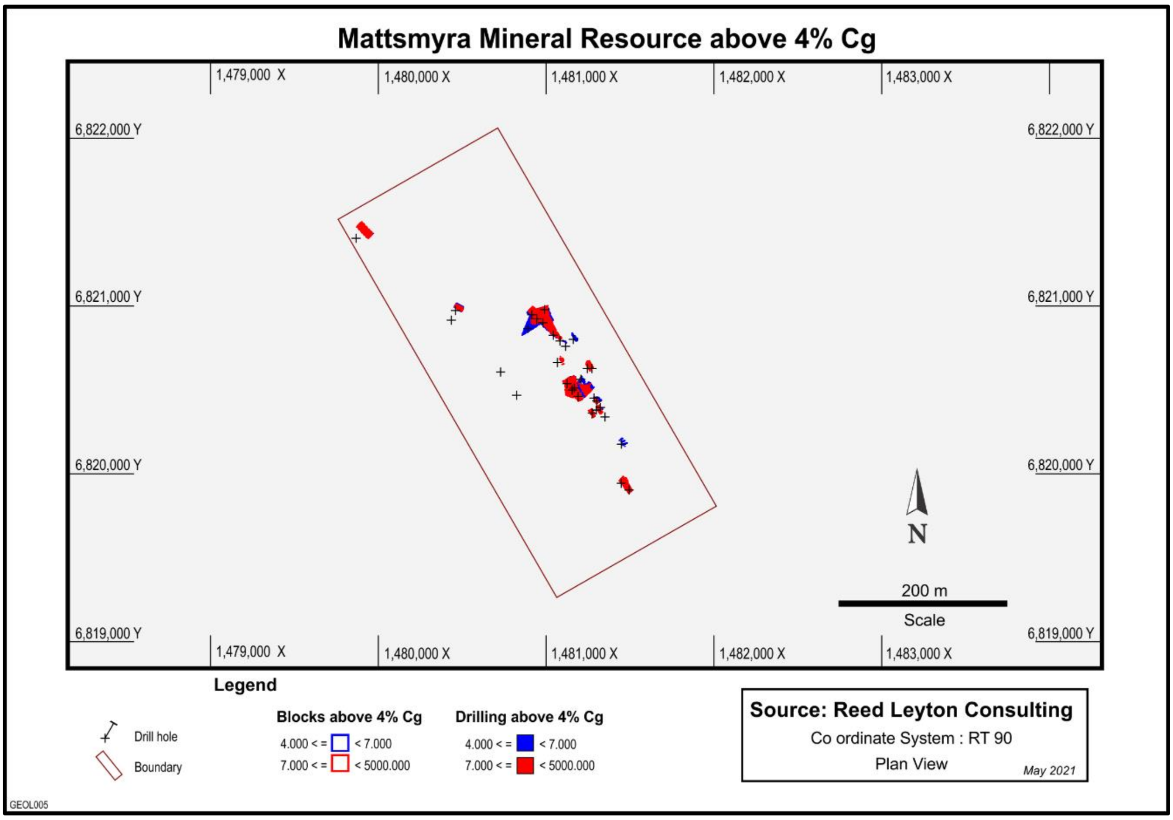


Figure 14-7: Mineral Resource in Plan View, Mattsmyra

14.2.7 Wire framing

- Using the above drillhole data, wire framing of the geological boundaries was performed by joining digitised section outlines at a 50 m spacing.
- The digitised sections were snapped to drillholes within ±25 m influence using above 7% Cg for a single Type A wireframe domain at Mattsmyra. Type B Graphite was defined using the geological limit of the Graphite.
- Vertical plane sections were digitised at 150-degree orientation at 50 m spacings. There is sufficient evidence for continuity of the mineralised envelope between a number of sections however the drillhole coverage has not been sufficient to determine if the mineralisation is continuous over the 2,200 m strike distance.
- All modelled wireframes were checked in plan, cross section, long section, and 3D rotated views.

- All geological wireframes were checked for crossing, inconsistencies, and closure.

Table 14-12: Domain volume validation, Mattsmyra

Domain	Wireframes Volume	Model Volume	Domains
Type A	2,299,331	2,298,020	100.06%

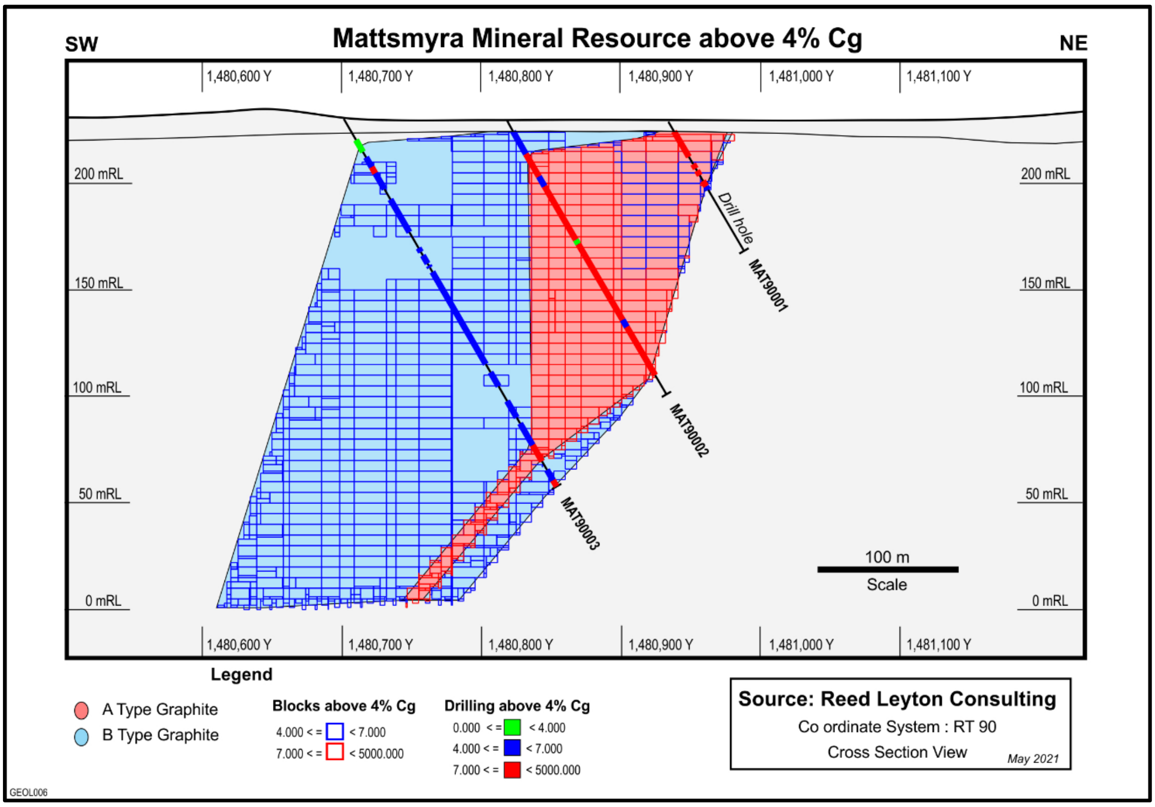


Figure 14-8: Mineral Resource cross section -Mattsmyra (looking NW).

14.2.8 Grade Interpolation

Grade interpolation was undertaken using inverse distance defined by the domain wireframes. The allocations of composites were calculated using a hard boundary at the domain wireframes.

Using Maptek Vulcan's Estimation Editor the grade estimation was run for Mattsmyra. Variables were populated using one single search ellipses with no cut off to the mineralisation domains.

Constant parameters used in this block estimation file, vieuncutivdcpctgro14juln140.bef include:

- The grade variable populated was C_uncut. The default given was 0.
- The number of samples used was populated in the variable numsam. The default given to this variable was 0.
- The number of drill holes used was stored in nodrill. The default given was 0.
- The sample distance used was stored in the variable samdis.
- The inverse distance method was applied.

Table 14-13: Block model parameters - Mattsmyra

Variables	Description
c_uncut	Carbon grade - reportable
s_uncut	Sulfur grade – not reportable
bd	Bulk Density
category	Resource category by script
mintype	Mineralization Domain, In situ, Overburden or Air
nodrill	Number of Drill holes
samdis	Average sample distance
numsam	Number of samples
pass	Estimation flag
mined	Mined or in situ
lithtype	FGRF, GRF, In situ, Overburden
rsc_cat	Final Resource Category meas = 1, ind = 2, inf = 3

Table 14-14: Search parameters - Mattsmyra

Pass	Min Sample	Max Sample	Distance
1	2	12	50
2	1	20	100
3	1	30	800

Table 14-15: Estimation parameters - Mattsmyra

Domain	Strike	Plunge	Dip	Major	Semi Major	Minor	Discretisation
Hg1	150	0	60	4	2	1	4x:8y:4z
Lg1	150	0	60	4	2	1	4x:8y:4z

14.2.9 Minimum width

No minimum width has been applied in the estimation of the Mattsmyra Mineral Resources.

14.2.10 Cut-off Grade

A grade cut-off of 7% Cg has been applied to the Mineral Resource estimation modelling to define Type A Graphite. A grade cut-off of 4% Cg has been applied to the Mineral Resource estimation for reporting purposes of Type A and Type B Graphite.

14.2.11 Additional Variables

Once the estimations had run, a number of additional variables were added or calculated. These variables included:

- Using the block calculation function of Vulcan the variable category was populated for a first pass look at resource categories. This calculation looked at the nearest neighbour distance variable ("samdis"). If samdis was >60, then the category variable was set to inf (inferred). If samdis was <60, then the category variable was set to ind (indicated).

- Using the block calculation function of Vulcan the variable rsc_cat was populated for final resource categories, indicated and inferred. The category variable, called rsc_cat. A calculation was run on the block model using a bounded wireframe. This variable was used to classify the resource based on drilling density, continuity and general confidence in each modeled wireframe. Therefore inside the indicated wireframe, the rsc_cat variable was set to ind (indicated). Anything outside of the indicated wireframe, the rsc_cat variable was set to inf (inferred).
- Using the block calculation function of Vulcan the variable bd was populated for bulk density. The script was run according to density test work by the Issuer previously attributed to various assays within the geology database. ReedLeyton has created a script file with an average BD taken between various C grades within the 'fgrf' and 'grf' rock type.

14.2.12 Mining and Metallurgical Assumptions

MPlan 2021 prepared constraining pit shells for ReedLeyton to be used for Mineral Resource estimation reporting of the Gropabo and Mattsmyra deposits using optimised pit shells generated using Datamine ™ NPVS software.

The input assumptions were based upon input assumptions used for the Kringel deposit. Although there was no engineering or geotechnical work performed explicitly to support these assumptions, they were benchmarked on the Kringel deposit input parameters and were deemed to be a reasonable representative for these neighbouring deposits.

The key assumptions used in the generation of the resource constraining pit shells for both Gropabo and Mattsmyra deposits were:

- overall slope angle for resource pit shell: 55 degrees;
- mill cut-off grade = 4.00%;
- break even cut-off grade = 4.21%;
- process cost: USD 84.18/t mill feed
- dilution 2.5%
- mining recovery 97.5%
- process recovery 93.7%

The cut-off grades assumed:

- graphite price of: 2,320 USD/t;
- recovered value of 2,103 USD/t after applying costs, taxes, mining, and process recovery factors; and
- mining cost: 4.51 USD/t rock mined.

14.2.13 Mattsmyra Mineral Resource Estimate

14.2.13.1 Previous Mattsmyra Mineral Resource estimate

The previous Mattsmyra Mineral Resource published in 2015 is provided below. This 2015 estimate was conducted exactly as described above and the applied levels of confidence were considered appropriate on the basis of drillhole spacing, sample interval, geological interpretation and all currently available assay data.

Table 14-16: Mattsmyra Indicated Mineral Resource estimate (2015 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Indicated	3.4	8.4
Total	3.4	8.4

Source: ReedLeyton 2015

Reported according to CIM Definition Standards 2011

Default Density of 2.7 t/m3 applied to in situ, then Density of 2.82 t/m3 applied to Type A Graphite and Density of 2.86 t/m3 applied to Type B Graphite
7% Cg cut-off grade applied;
No Mining parameters applied;

Table 14-17: Mattsmyra Inferred Mineral Resource estimate (2015 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	1.2	8.4
Total	1.2	8.4

Source: ReedLeyton 2015

Reported according to CIM Definition Standards 2011
Default Density of 2.7 t/m3 applied to in situ, then Density of 2.82 t/m3 applied to Type A Graphite and Density of 2.86 t/m3 applied to Type B Graphite
7% Cg cut-off grade applied;
No Mining parameters applied;

14.2.13.2 Current 2021 Mineral Resource estimate

ReedLeyton undertook an update of the previous 2015 Mineral Resource estimate in order to constrain the global resource within the open-pit shell designed by MPlan (2021). The application of this pit-shell provides reassurance with regard the current prospects for economic extraction.

Table 14-18: Mattsmyra Indicated Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Indicated	5.83	7.14
Total	5.83	7.14

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.82 t/m^3 applied to Type A Graphite and Density of 2.86 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

The above numbers are literal, whereas the accuracy of the techniques requires that the estimates' parameters should actually result in figures rounded down to better reflect the order of accuracy. Hence ReedLeyton has rounded down the mineralisation tonnage to the nearest ten thousand tonnes. The resource estimates then become as shown on Table 14-7.

Table 14-19: Mattsmyra Indicated Mineral Resource estimate, (2021 - rounded)

Classification	Tonnes (Mt)	Grade Cg %
Indicated	5.8	7.1
Total	5.8	7.1

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.82 t/m^3 applied to Type A Graphite and Density of 2.86 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

Table 14-20: Mattsmyra Inferred Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	1.51	8.06
Total	1.51	8.06

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014

Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.82 t/m^3 applied to Type A Graphite and Density of 2.86 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

The above numbers are literal, whereas the accuracy of the techniques requires that the estimates' parameters should actually result in figures rounded to better reflect the order of accuracy. Hence ReedLeyton has rounded the mineralisation tonnage to the nearest ten thousand tonnes. The resource estimates then become as shown on Table 14-7.

Table 14-21: Mattsmyra Inferred Mineral Resource estimate, (2021 - rounded)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	1.5	8.0
Total	1.5	8.0

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.82 t/m^3 applied to Type A Graphite and Density of 2.86 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

14.2.14Discussion

The previous Mattsmyra Mineral Resource estimate (Flinders 2015) only reported Type A graphite above a 7% Grade cut off. The Mattsmyra Mineral Resource estimate was modelled using a 7% grade cut-off with no mining parameters applied. The estimation resulted in Measured and Indicated Mineral Resources being reported.

The Current Mattsmyra Mineral Resource estimate now reports Type A and Type B graphite to a 4% Grade cut off. The Mattsmyra Mineral Resource estimate is now constrained by the Mplan 2021 pitshell for reporting purposes.

Comparison of the current Mineral Resource estimate with that of 2015 (ReedLeyton) shows significantly higher tonnage for Indicated Mineral Resources as expected due to the inclusion of Type A and Type B graphite to a 4% grade cut-off.

14.3 Gropabo – Mineral Resource estimate

14.3.1 Drilling data

- Thirty-eight diamond drill holes totalling 1,789 m, drilled into the Gropabo area in 1991 and 1992. Thirty-five diamond drillholes were included in the current mineral resource estimation.
- Data relating to the collar locations, drill collar orientations were sighted by the Geoff Reed in sections and plans of the day.
- Geoff Reed inspected the area with the Issuer's personnel and was able to locate many 1991 and 1992 drill hole collars.

- The profile spacing is approximately 50 m and distance between holes on section is generally 25 m. 55% of the holes are dipping 50-degrees, the remaining are dipping 60-degrees. Hole lengths are typically around 50 m, resulting in a vertical depth test of around 45 m. Shorter holes were drilled where the graphite was intersected close to surface. Hole numbering starts with the year (91 or 92) and ends with a continuous hole number from 91001 to 91017 and 92001 to 92021.
- No drillholes have been deviation surveyed to date. The start azimuth was measured with a handheld compass. Any uncertainty in drill hole trend cause by the lack of surveys is considered minor at the spacing of the drill holes and relatively short hole length in relation to a scale of the resource.
- The position for Gropabo drillholes has been calculated by converting the historic local grid into coordinates of RT90 national grid and are assumed accurate.
- All drillhole core for thirty-eight diamond drill holes has been located by the Issuer's staff in Woxna, Sweden. Core from 10 holes has been inspected by the Geoff Reed.

Table 14-22: Drilling Database Summary, Gropabo

Hole Type	Drill Series	Drill Number	Drill Meters	Resource Intersection Meters
DD	91	17	858	247
DD	92	21	931	257
Total		38	1,789	504

14.3.2 Database Integrity

- Capture of digital data was completed by Flinders' staff. Hard copy data has been verified and all data is stored in a database and managed by the Issuer.
- Drilling data from drill programs were transferred in digital format by the Issuer's staff.
- Digital data has been both randomly and systematically checked by ReedLeyton and shown to be correct using a number of checks listed below. Assay data in original laboratory sheets has not been sighted from the 1991, 1992 drilling program.
- The digital data was compiled directly into Microsoft Excel by the Issuer, validated in Microsoft Access and exported into a csv format. The database was then imported into Maptek Vulcan software in the csv format.
- The database for Gropabo was attributed to thirty-eight diamond drillholes, which provided the verified information for compositing (specifically the collar, survey, lithology and assay tables). The database included drill holes with recorded collar elevation. This database was named ddhgro.vis.isis.

14.3.3 Drillhole Spacing

- Thirty-eight drillholes for 1,789 m of diamond drilling were drilled at Gropabo. Thirty-six drillholes intersected mineralisation and were subsequently assayed.
- Drillhole spacing was a 50 × 25 m grid pattern.
- For wire framing purposes 141-degrees strike was considered the optimal orientation. Strike of mineralisation varied from 130-degrees to 150-degrees.
- Polygons were created every 50 m through the thirty-five diamond drillholes at the project.

14.3.4 Drilling Orientation

- Drillholes were drilled mostly on a single orientation averaging 55-degrees with a maximum orientation of 67-degrees and a minimum orientation of 27-degrees.
- Due to the amount of drilling and orientation, the true thickness is generally considered to be 70–80% of drilled thickness.

- The likelihood that mineralisation is developed in an orientation other than that interpreted is considered to be low.

14.3.5 Chemical Analysis

- A total of 389 samples from thirty-six drill holes were analysed in total at Gropabo for diamond drill holes with thirty-five diamond drill holes included in the current mineral resource estimation.
- Drilled diamond core was sampled and analysed by ICP method at the laboratory of SGAB ANALYS, Luleá, Sweden.
- The method applied by was the standard for the industry of the day, and although no quality assurance data is available, it is considered to be of a very high quality.
- The method applied is to current industry standards, and although no standard data is available, it is considered to be of a very high quality.

14.3.6 Sample length

- All holes drilled at Gropabo were sampled with an average of one (1.8) metre intervals. Check sampling by the Issuer at the request of ReedLeyton used identical sample intervals.
- Composites of the drill hole assays are generated using Maptek Vulcan software with run lengths of 2 metre.
- These composites honour the geological wireframes. Checking was undertaken by generating an Isis file and visually inspecting the result of the composite.
- Specific components of the compositing include:
- Run Lengths of 2-metres.
- Data Field C_pct and S_pct was composited.
- The composite file was then applied a tag for each composite with the character in the 'bound' column. This new composite isis file was called vierscgron1.cmp.isis and used in the estimation process.

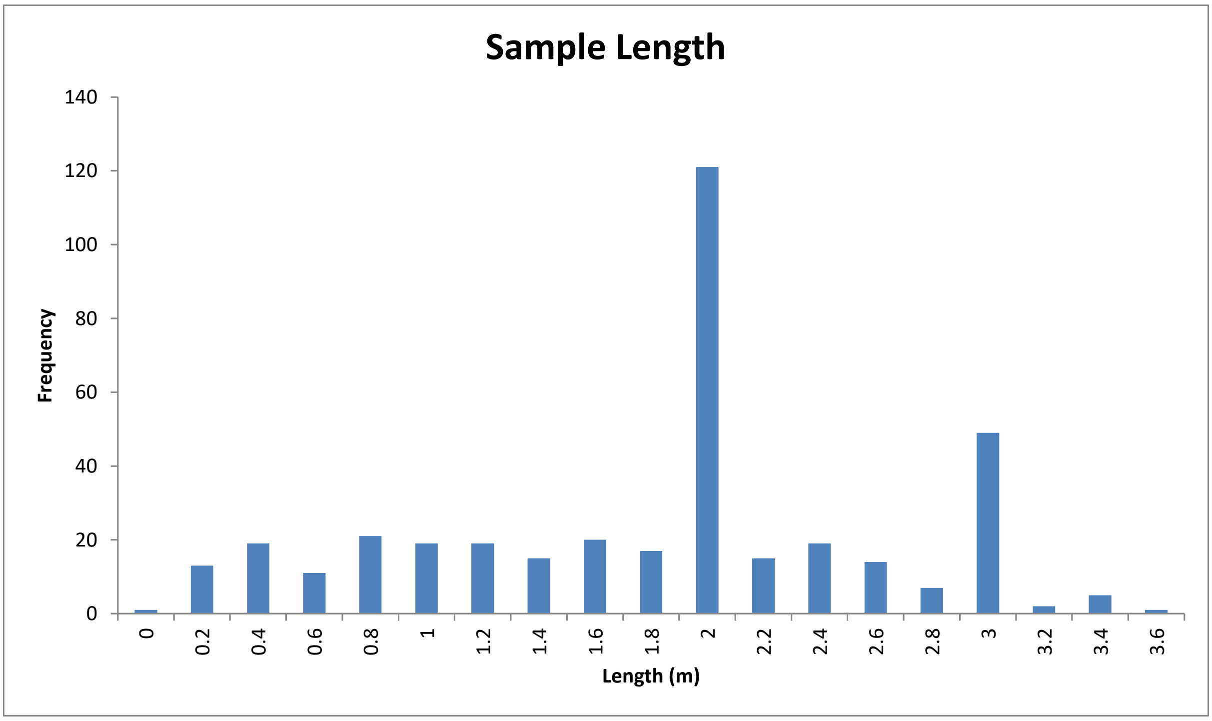


Figure 14-5: Histogram of raw Sample Lengths, Gropabo

14.3.7 Density

Using the *bulk density (BD*) density default function of Vulcan, the variable *BD* was populated. Note that this is not true bulk density but a software nomenclature for normal density as per the description provided in Section12.3.

The in situ value *BD* 2.7 was run according to density testwork by Flinders previously attributed to various assays within the geology database. ReedLeyton has created a file with an average *BD* taken between various Cg % grades within the resource and waste blocks outside the resource.

The density value 2.81 for Type A Graphite and the value 2.83 for Type B Graphite was run according to density testwork by Flinders previously attributed to various assays within the geology database. ReedLeyton has created a file with an average BD taken between various Cg % grades within the resource and in situ blocks outside the resource.

14.3.8 Geological Model

- Mineral Resources has been estimated by ReedLeyton on a bearing of 141-degree strike.
- The deposit was drilled within an area approximately 500 × 100 m.
- The mineralisation was intersected on all the drilling sections and is so far known to at least a depth of 60 m below the surface.
- Mineralisation strikes NW-SE, and dips varies between 65 and 85 degrees to the SW.
- Mineralisation is present as four main mineralised bodies. A single grade domain was defined using a cut-off grade of 7% Cg for Type A graphite. Type B graphite was defined using the geological limit of the graphite. The thickness in the section of the plane was usually more than 6 m, but varied between 1 m and a little more than 25 m.
- A single block model was constructed, named vie_gro_2014nov.bmf. The parameters used in the setup file vieuncutivdcpctgro14nov.bef for Gropabo.
- The block model was created using the one bdf file. This original block model contained only default values except for the variable domain, which was populated in relation to the wireframes in which the blocks resided in.
- A Block rotation of 141 Bearing, 0 Plunge and 0 Dip was applied.
- Parent block size was 5 × 25 × 5 m with sub blocks at 1 × 5 × 1 m.
- An offset of 1,000 × 300 × 300 m was applied
- The variables include the type and their default values before estimation.

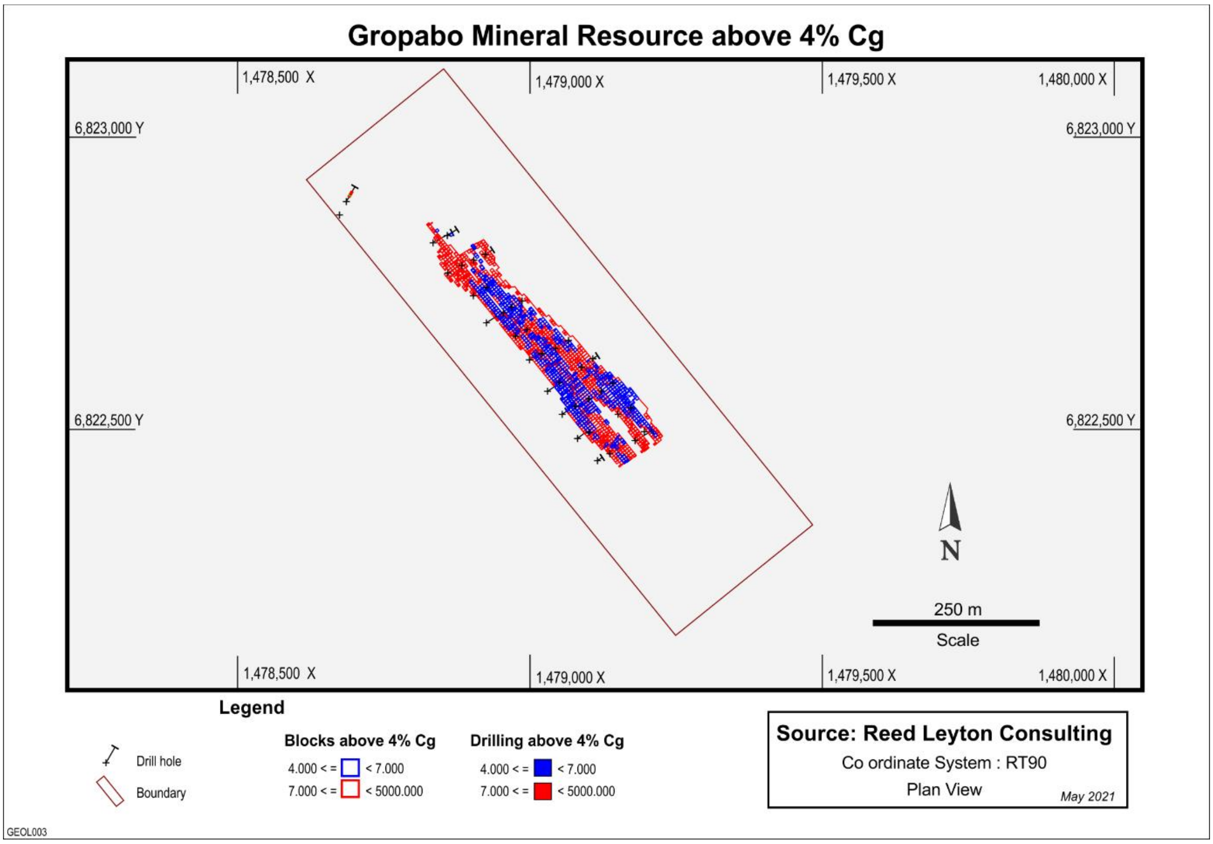


Figure 14-6: Mineral Resource in Plan View, Gropabo

14.3.9 Wire framing

- Using the above drillhole data, wire framing of the geological boundaries were performed by joining digitised section outlines at a 50 m spacing.
- The digitised sections are snapped to drill holes within ±25 m influence using above 7% Cg for a single Type A wireframe domain at Gropabo and Type B Graphite was defined using the geological limit of the Graphite.
- Vertical plane sections were digitised at 141-degree orientation at a 50 m spacing. There is sufficient evidence for continuity of the mineralised envelope between sections.
- All modelled wireframes were checked in plan, cross section, long section, and 3D rotated views.
- All geological wireframes were checked for crossing, inconsistencies, and closure.

Table 14-23: Domain volume validation, Gropabo

Domain	Wireframes Volume	Model Volume	Domains
Total	970799	966640	100%

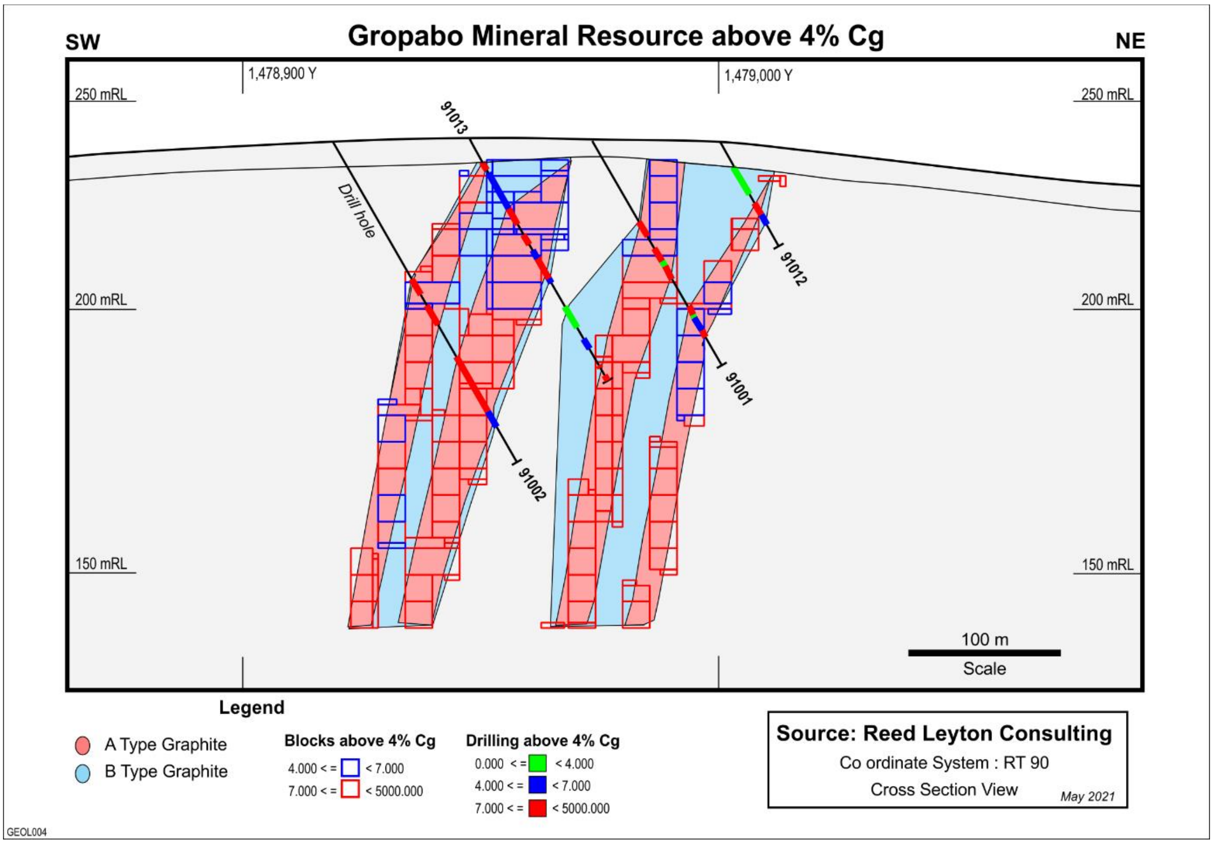


Figure 14-7: Mineral Resource Cross Section, Gropabo, (looking NW)

14.3.10 Grade Interpolation

Grade interpolation was undertaken using inverse distance defined by the domain wireframes. The allocations of composites were calculated using a hard boundary at the domain wireframes.

Using Maptek Vulcan's Estimation Editor the grade estimation was run for Gropabo. Variables were populated using one single search ellipses with no cut off to the mineralization domains.

Constant parameters used in this block estimation file, vieuncutivdcpctgro14nov.bef include:

- The grade variable populated was C_uncut. The default given was 0.
- The number of samples used was populated in the variable numsam. The default given to this variable was 0.
- The number of drill holes used was stored in nodrill. The default given was 0.
- The sample distance used was stored in the variable samdis.
- The inverse distance method was applied.

Table 14-24: Block Model Parameters, Gropabo

Variables	Description
c_uncut	Carbon grade - reportable
s_uncut	Sulfur grade – not reportable
bd	Bulk Density
category	Resource category by script
mintype	Mineralization Domain, In situ, Overburden or Air
nodrill	Number of Drill holes

Variables	Description
samdis	Average sample distance
numsam	Number of samples
pass	Estimation flag
mined	Mined or in situ
lithtype	FGRF, GRF, In situ, Overburden
rsc_cat	Final Resource Category meas = 1, ind = 2, inf = 3

Table 14-25: Search Parameters, Gropabo

Pass	Min Sample	Max Sample	Distance
1	2	12	50
2	1	20	100
3	1	30	800

Table 14-26: Estimation Parameters, Gropabo

Domain	Strike	Plunge	Dip	Major	Semi Major	Minor	Discretisation
Type A	140	0	60	4	2	1	4x:8y:4z
Type B	140	0	60	4	2	1	4x:8y:4z

14.3.11 Minimum width

No minimum width has been applied in the estimation of the Gropabo Mineral Resources.

14.3.12 Cut-off Grade

A grade cut-off of 7% Cg has been applied to the Mineral Resource estimation modelling to define Type A graphite. A grade cut off of 4% Cg has been applied to the Mineral Resource estimation for reporting purposes of Type A and Type B graphite.

14.3.13 Additional Variables

Once the estimations had run, a number of additional variables were added or calculated. These variables included:

- Using the block calculation function of Vulcan the variable category was populated for a first pass look at resource categories. This calculation looked at the nearest neighbour distance variable ("samdis"). If samdis was >60, then the category variable was set to inf (inferred). If samdis was <60, then the category variable was set to ind (indicated).
- Using the block calculation function of Vulcan the variable rsc_cat was populated for final resource categories, indicated, and inferred. The category variable, called rsc_cat. A calculation was run on the block model using a bounded wireframe. This variable was used to classify the resource based on drilling density, continuity, and general confidence in each modelled wireframe. Therefore inside the indicated wireframe, the rsc_cat variable was set to ind (indicated). Anything outside of the indicated wireframe, the rsc_cat variable was set to inf (inferred).
- Using the block calculation function of Vulcan the variable bd was populated for bulk density. The script was run according to density test work by the Issuer previously attributed to various assays within the

geology database. ReedLeyton has created a script file with an average BD taken between various C grades within the 'fgrf' and 'grf' rock type.

14.3.14 Mining and Metallurgical Assumptions

MPlan 2021 prepared constraining pit shells for ReedLeyton to be used for Mineral Resource estimation reporting of the Gropabo and Mattsmyra deposits using optimised pit shells generated using Datamine TM NPVS software.

The input assumptions were based upon input assumptions used for the Kringel deposit. Although there was no engineering or geotechnical work performed explicitly to support these assumptions, they were benchmarked on the Kringel deposit input parameters and were deemed to be a reasonable representative for these neighbouring deposits.

The key assumptions used in the generation of the resource constraining pit shells for both Gropabo and Mattsmyra deposits were:

- overall slope angle for resource pit shell: 55 degrees;
- mill cut-off grade = 4.00%;
- break even cut-off grade = 4.21%;
- process cost: USD 84.18/t mill feed
- dilution 2.5%
- mining recovery 97.5%
- process recovery 93.7%

The cut-off grades assumed:

- graphite price of: USD 2,320;
- recovered value of USD 2,103/t after applying costs, taxes, mining, and process recovery factors; and
- mining cost: USD 4.51/t rock mined

14.3.15Gropabo Mineral Resource Estimate

14.3.15.1 Previous Gropabo Mineral Resource Estimate

The 2015 Mineral Resource estimate provided below was prepared in accordance with the CIM Definition Standards of June 2011. The classification of the resource at the applied levels of confidence is considered appropriate on the basis of drillhole spacing, sample interval, geological interpretation and all currently available assay data.

Table 14-27: Gropabo Indicated Mineral Resource estimate (2015 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Indicated	1.5	8.8
Total	1.5	8.8

Source: ReedLeyton 2015

Reported according to CIM Definition Standards 2011

Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.81 t/m^3 applied to Type A Graphite and Density of 2.83 t/m^3 applied to Type B Graphite

7% Cg cut-off grade applied;

No Mining Parameters applied

Table 14-28: Gropabo Inferred Mineral Resource estimate (2015 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	0.7	8.7
Total	0.7	8.7

Source: ReedLeyton 2015

Reported according to CIM Definition Standards 2011
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.81 t/m^3 applied to Type A Graphite and Density of 2.83 t/m^3 applied to Type B Graphite
7% Cg cut-off grade applied;
No Mining Parameters applied

14.3.15.2 Current 2021 Mineral Resource Estimate – Gropabo

ReedLeyton undertook an update of the previous 2015 Mineral Resource estimate in order to constrain the global resource within the open-pit shell designed by MPlan (2021). The application of this pit-shell provides reassurance with regard the current prospects for economic extraction.

Table 14-29: Gropabo Indicated Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Indicated	2.33	7.72
Total	2.33	7.72

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.81 t/m^3 applied to Type A Graphite and Density of 2.83 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

The above numbers are literal, whereas the accuracy of the techniques requires that the estimates' parameters should actually result in figures rounded down to better reflect the order of accuracy. Hence ReedLeyton has rounded down the mineralisation tonnage to the nearest ten thousand tonnes. The resource estimates then become as shown on Table 14-7.

Table 14-30: Gropabo Indicated Mineral Resource estimate, (2021 - rounded)

Classification	Tonnes (Mt)	Grade Cg %
Indicated	2.3	7.7
Total	2.3	7.7

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m^3 applied to in situ, then Density of 2.81 t/m^3 applied to Type A Graphite and Density of 2.83 t/m^3 applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

Table 14-31: Gropabo Inferred Mineral Resource estimate (2021 ReedLeyton)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	0.61	8.07
Total	0.61	8.07

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
Default Density of 2.7 t/m³ applied to in situ, then Density of 2.81 t/m³ applied to Type A Graphite and Density of 2.83 t/m³ applied to Type B Graphite
4% Cg mill cut-off grade applied for reporting purposes constrained within the MPlan 2021 pitshell
The 2021 PEA mine plan pitshell determines the "reasonable prospects for economic extraction"

The above numbers are literal, whereas the accuracy of the techniques requires that the estimates' parameters should actually result in figures rounded down to better reflect the order of accuracy. Hence ReedLeyton has rounded down the mineralisation tonnage to the nearest ten thousand tonnes. The resource estimates then become as shown on Table 14-7.

Table 14-32: Gropabo Inferred Mineral Resource estimate, (2021 - rounded)

Classification	Tonnes (Mt)	Grade Cg %
Inferred	0.6	8.0
Total	0.6	8.0

Source: ReedLeyton 2021

Reported according to CIM Definition Standards 2014
No geological losses applied;
Default Density of 2.7 t/m³ applied to in situ, then Density of 2.81 t/m³ applied to Type A Graphite and Density of 2.83 t/m³ applied to Type B Graphite
7% Cg cut-off grade applied;
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability

14.3.15.3 Discussion

The previous Gropabo Mineral Resource estimate only reported Type A graphite above a 7% grade cut-off. The Gropabo Mineral Resource estimate was modelled using a 7% grade cut-off with no mining parameters applied. The estimation resulted in Measured and Indicated Mineral Resources being reported.

The Current Gropabo Mineral Resource estimate now reports Type A and Type B graphite to a 4% grade cut-off. The Gropabo Mineral Resource estimate is now constrained by the Mplan 2021 pitshell for reporting purposes.

Comparison of the current Mineral Resource estimate with that of 2015 (ReedLeyton) shows significantly higher tonnage for Indicated Mineral Resources as expected due to the inclusion of Type A and Type B graphite to a 4% grade cut-off.

14.4 Mineral Resource Estimate - Mansberg

The limited exploration data for the Historical Mansberg deposit does not currently support a Mineral Resource estimate to be stated.

15 MINERAL RESERVE ESTIMATES

Mineral Reserve has not been estimated for the Project.

16 MINING METHODS

The mine design, scheduling, and costing for the Woxna Mine was undertaken by independent mining specialist MPlan. The selection of the most appropriate and cost-effective method of mining was based on both historical information and the characteristics of the mineralisation and testwork results. A Basis of Design (BoD) was prepared which provides an effective tool to clearly present the 2021 decisions, assumptions and specifications that were used to develop the mining scenario for Kringel deposit at the Woxna Mine.

16.1 Mining method selection

The methodology used for selection of the most appropriate mining method for the Kringel deposit comprises three interconnected stages:

- Stage I: interrogates many site-specific parameters such as rock properties of waste and mineralised material;
- Stage II: selection of the top-ranked mining methods;
- Stage III includes a more detailed examination of the top-ranked mining methods in terms of mining selectivity, planned production rate and economic considerations.

16.1.1 Stage I – Key parameters

Many variables and factors have an influence on mining method selection. Most important group of parameters are the mineralisation characteristics and style of deposit. Many other parameters, listed in Table 16-1, have an influence on the selection process were considered as key factors in the selection of the mining method.

Table 16-1: Parameters considered grouped by key factors

Key Factors	Parameters
Geometrical	Deposit geometry: shape, thickness, dip and plunge
	Depth
Geotechnical	Overburden materials and graphite horizon rock properties
	Nature and irregularity of hanging wall and footwall
	Geotechnical aspects of the deposit and surrounding rock
Geographical	Topographic and climatic conditions;
	Social conditions
	Surrounding communities and environmental protection
Geological	Grade distribution and cut-off grade
	Economic value of the different grades of mineralisation identified
	Geological losses, dilution and recovery
Economic	Maximise graphite recovery (selectivity)
	Support economic viability
	Operating or anticipated costs of production
	Mining operation sequence
	Mining development costs and preparatory work
Productivity	Productivity and mechanisation
	Support planned mine production
Safety, Environment and Regulations	Provide a safe working environment
	Reduce negative environmental impact of mining

The Kringel deposit geometry combined with a relatively flat topography favours surface mining methods. As the mineralisation has been previously mined with an open-pit mining method it is almost certain that this will be the preferred method.

The overburden is composed of Quarterly age moraine, ranging from 0.5 m to 20 m in thickness with an average of 3.5 m as listed in Table 16-2. Moraine is mainly described as micrite, chert and limestone (i.e., muddy). Waste rock is calcareous quartz-rich meta sediment. In some areas over the concession, the graphite mineralisation is almost cropping out on surface. The main graphite mineralisation bodies are faulted.

Table 16-2: Kringel Deposit Lithology Information

Lithology	In situ wet Density (t/m^3)	Thickness (m)		
		Min	Max	Average
Overburden	2.3	0.5	20	3.5
Banded meta-stuff	2.7	0.5	65	30
Pegmatite	2.7			
Graphite mineralisation (FGRF) (B type mineralisation)	2.7	5	15	10
Higher grade graphite mineralisation (HG) (A type mineralisation)	2.7			

To summarise, tabular shaped deposit dipping south between 60^0 to 80^0 degrees, relatively hard waste rock and graphite bearing material with variable thickness and distribution will be the most important factors guiding the selection of the mining method.

16.1.2 Stage II – Mining method ranking

Different mining methods were considered, common extraction and material handling systems used at different mines throughout the world are listed in Table 16-3 and Table 16-4. Two approaches were used to rank the mining methods and to evaluate their technical feasibility and relevance: the University of British Columbia (UBC) method and the Nicholas method.

Table 16-3: Common Extraction Methods

Surface	Underground
Dragline	Continuous mining machine
Bucket Wheel Excavator (BWE)	Longwall
Shovel	Shaft/decline
Front-end loader	Cut & Fill
Excavator	Room & Pillar
Surface Miner (SM)	

Table 16-4: Common Material Handling Systems

Surface	Underground
Slurry pipeline	Conveyor
Road or haul trucks	Feed bins
Conveyors	Shaft or decline haulage
Stockpile and reclaimer	

16.1.2.1 The Nicholas Method

This method numerically ranks geometry and rock mechanics characteristics of the mineralisation, overburden hanging wall and overburden footwall. It also includes the rock quality designation (RQD) and the rock substance strength (RSS) which consists of the UCS divided by overburden pressure.

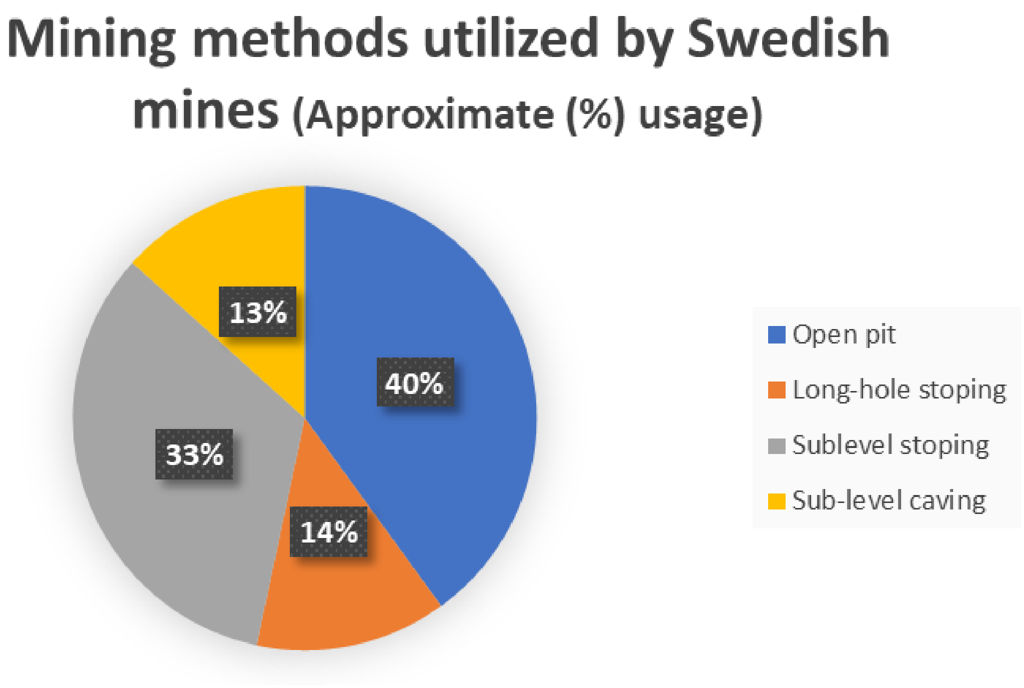
Each mining method is then rated numerically as to its suitability for each category of factors. The rankings are then summed together with the higher rankings being the more promising or expected mining methods. Each ranking consists of a number from 0 to 4 or -49.

- A number -49 completely eliminates a mining method from being feasible.
- A value of 0 strongly suggests that this characteristic makes that particular mining technique less attractive.
- The value of 1 or 2 indicates that a characteristic should not have a negative impact on a method.
- A ranking of 3 or 4 indicates a very favourable characteristic for that particular mining method.

The rock mechanics characteristics are scaled with a weighting factor scale according to the mineralisation zone, hanging wall or footwall. The Nicholas approach provides a quantitative approach for selecting a mining method.

16.1.2.2 UBC Mining Method Selection

The Rock Mechanics Group of the University of British Columbia (UBC) Mining Method Selection is a modified version of the Nicholas approach for selection of mining method based on mineralisation characteristics. The Nicholas method adapted version was developed in an effort to set more prominence on stope mining instead of mass mining techniques such as caving. The purpose for using this method is that most Scandinavian mines utilise open-pit, open stoping and sublevel stoping. The relative percentages of these mining methods as used during 2020 in Sweden are illustrated in Figure 16-1 below.



Source and date: Gosselin Mining 2021

Figure 16-1: Mining methods used by Swedish mines

Selection involves summation and ranking of numerical values associated with mineralisation characteristics that reflect the suitability of a particular method. This numbering system follows a very similar pattern to the Nicholas method. A value, -10, was introduced to strongly discount a method without eliminating it as with the -49 value. Moreover, the rock mechanics ratings were adjusted to reflect improvements with ground support and monitoring techniques. Some of the mineralisation parameters values defined in Stage I are listed in Table 16-5 and mining method ranking results from the UBC Mining Method are given in Table 16-6. The UBC selection process is listed below.

Table 16-5: UBC Mining Method Selection values for Kringel deposit

Deposit Characteristics	Values	Description	
Geometry and Grade Distribution	General shape/width	Platy-tabular	Two dimensions are larger than the thickness
	Ore Thickness	Narrow	3 - 10 m
	Ore Plunge	Intermediate	20º – 55º
	Depth below surface	Shallow	<100 m
	Grade Distribution	Gradational	Grade has a zonal characteristic which gradually changes from zone to zone
Rock Mass Rating	Graphite Zone	Medium	40 - 60
	Hanging Wall	Strong	60 - 80
	Footwall	Strong	60 - 80
Rock Substance Strength	Graphite Zone	Medium	40 - 60
	Hanging Wall	Strong	60 – 80
	Footwall	Strong	60 - 80

Using the input data from Table 16-5, the UBC method selection process is then applied and the resultant top four ranked mining methodologies determined, as listed in Table 16-6, and proved to be, in order top down, open-pit, sublevel stoping, shrinkage stoping and lastly cut and fill. This result is mainly explained by the deposit geometry, and the high strength classification of the excavated material, i.e. ranging from 100-200 mega pascals (MPa), on both hangingwall and footwall.

Table 16-6: UBC Mining Method selection results ranking

Mining Method	Ranking
Open pit	36 Best
Sublevel stoping	36 Best
Shrinkage stoping	33
Cut and fill	30
Room and pillar	29
Longwall mining	21
Top slicing	18
Square set	11
Sublevel caving	-25
Block caving	-33 Worst

16.1.3 Stage III - Top Ranked method evaluation

There are a number of ways to proceed to surface mining depending on the rock breakage material excavating characteristics and loading components, as summarised below:

- Rock breakage:
  - no breakage necessary (material: topsoil);
  - ripping with a dozer (material: stiff soil or weaker rock formations such as weathered overburden); and
  - drilling and blasting (material: medium to hard rock formations such as overburden horizon).
- Materials handling:
  - dragline (direct casting);
  - shovel or front-end loader and trucks;
  - bucket wheel excavator (BWE) and belt conveyor,
  - surface miner (SM) and trucks.

The excavating/hauling methods in surface operations can be divided into cyclic and continuous as listed in Table 16-7 below.

Table 16-7: Classification of loading-excavating methods and equipment for surface operation

Surface operation	Category or Method	Machine (Application)
Cyclic	Shovel	Power shovel, front-end loader, hydraulic excavator, backhoe (mining ore or stripping overburden)
	Dragline	Crawler, walking (stripping overburden)
	Dozer	Rubber-tired, crawler (blade)
	Scraper	Rubber-tired, crawler
Continuous	Mechanical excavator	BWE (overburden), cutting-head (soil, coal), surface miner (mining ore or overburden removal)
	Highwall mining	Auger, highwall miner (coal)
	Hydraulicking	Monitor or giant (placer)
	Dredging	Bucket ladder, hydraulic (placer)

The relative cost of the four top ranked mining methods for Woxna Mine are listed in Table 16-8:

Table 16-8: The relative cost of typical mining methods and excavation tonnages yielded

Mining Method	Relative Operating Cost per tonne	Average t/day	Average t/shift
Sublevel Stoping	7 to 25	1,500 – 20,000	20 - 115
Shrinkage stoping	20 to 50	200 – 800	20 - 28
Cut and fill	20 to 70	500 – 1 500	12 - 48
Open pit	2 to 22	10,000 – 50,000	5,000 -25,000

16.2 Basis of design for selected open pit

The major thought processes and assumptions behind design decisions made to meet Woxna Graphite's production requirements are presented in this section. The BoD is used as the foundation for design calculations and other design decisions.

16.2.1 Planned production rates

The planned mining production rate objective is set at 160 kt RoM (graphite + barren material) or RoM (as defined in Section 2.4) per annum delivered to the primary crusher or RoM pad stockpiles over the life of the project. The reference point at which RoM is defined is at the point where the RoM is delivered to the Woxna Concentrator, i.e. primary crusher or RoM pad stockpiles. The expected LoM is 15 years with a Life of Project (LoP) of 19 years for which the Woxna Concentrator will be fed from stockpiles.

The process plant overall recovery is estimated at 93.8%. The graphite will be extracted from the Kringel exploitation concession.

Table 16-9: Kringel planned LoP production rates

Parameter	Value
VAP planned production	6,604 tpa CSPG at 99.95% C 7,479 tpa micronized graphite at 92.3% C
Woxna Concentrator recovery and graphite concentrate grade	93.8% recovery at 92.3% C
Woxna Concentrator graphite planned production	14.7 ktpa at 92.3% C
Woxna Concentrator planned RoM throughput	160 ktpa

16.2.2 Beneficiation requirements

The planned mining production target is approximately 160 kt RoM from Woxna Mine targeting first the higher-grade graphite mineralisation of A-type (see Section 14).

RoM feed will be beneficiated in the Woxna Concentrator with an overall process graphite concentrate recovery of 93.8%.

The mine plan aims to provide solely A-type mill feed, to be provided for the first 15-years, after which B-type graphite beneficiation will be required for the RoM feed. The Woxna Concentrator process recoveries are not sensitive to RoM grades and therefore there are no variation limits defined beyond Type A and Type B and waste. Variations will occur on a daily basis depending on the mine plan and grade control, and throughout the project life because mining will start in areas with the highest C grades which will decrease in subsequent years. The assumed carbon and sulfur grades for RoM are presented in Table 16-10 and were used as guidelines to generate a LoP plan.

Table 16-10: Guideline variations in RoM feed

	Carbon	Sulfur
Expected variation during plant life	12% or more at beginning of plant life 8% at end of plant life	
Allowed variation range over LoP	10% +/- 2%	0% to 4.0
Allowed 5 days rolling average	12% +/- 1% for year 1 to 8% +/- 1 for year 20	

16.2.3 Time allocation

Availability and utilisation factors for equipment and staff were applied to calculate scheduled hours as well as operating hours for the mining operation.

Mechanical availability is a measure of time that a piece of equipment is mechanically capable of operating. Mechanical availability includes scheduled and unscheduled maintenance and is a function of the equipment usage intensity and its application in the mine and is summarised as follows:

- 80% for mining trucks, shovels and ancillary plant;
- 75% for drill rigs due to difficult drilling conditions at the mine site.

MPlan estimated that the mine will have in average of 350 working days excluding 14 stoppage days annually due to bad weather conditions and industrial stops. The mining workforce will work on 8 hours shift with two shifts per day and seven days per week. Gosselin Mining assumed that three rotating crews working would accomplish continuous coverage of the mine, i.e., two crews working while one crew on a rest cycle. Therefore, three crews will be counted for each position when estimating manpower requirements.

In addition of estimated 14 stoppage days per year, operating delays (waiting time, queuing for trucks) and idle time (meal, break, shift changes and others) contribute to decreasing the operating time. This additional loss is accounted for with the operational usage factor. Effective working time estimation is listed in Table 16-11.

Table 16-11: Annual effective operating time

	12 h	24 h
Calendar time hours (364 days)	4,368	8,736
Mechanical availability (80%)	3,494	6,988
	8 h	**16 h**
Calendar hours (364 days)	2,912	5,832
Effective calendar hours (350 days)	2,800	5,600
Effective day shift (85% operational usage)	2,380	
Effective night shift (80% operational usage)	2,240	
Effective annual operating hours		**4 ,620**

The effective hours for mining personnel are estimated at 2,240-night shift hours and 2,380-day shift hours for a total of 4,620 hours per year.

16.2.4 Unit rate

Unit rates of the following commodities were provided by Woxna Graphite and last updated in January 2021. Costs were used in the Mining Study in pit optimisation and operating cost (OPEX) estimates where SEK refers to Swedish Krona:

Table 16-12: Unit rates used in the Mining Study

Product	Rate	Unit	Source
electricity	380	SEK/MWh	Statistiska centralbyrån (SCB)
diesel (March 01, 2021)	15.82	SEK/litre	GlobalPetrolPrices
graphite as spheronized – coarse <25 µm at 99.99% Carbon	3,000	USD/t	LEM
graphite as micronized <10 µm at 99.95% Carbon graphite	2,000	USD/t	LEM

The pricing for spheronized and micronized graphite presented in Table 16-12 were specified prior to the marketing study and the inclusion of coating into project. After coating, the spheronized product stream increased, and the micronized graphite decreased slightly, with a net increase in per unit product. Since the mine pit was limited by other criteria (pit depth and permit perimeter) it was not required to modify the life-of mine plan based on open pit mine design.

16.2.5 General arrangement

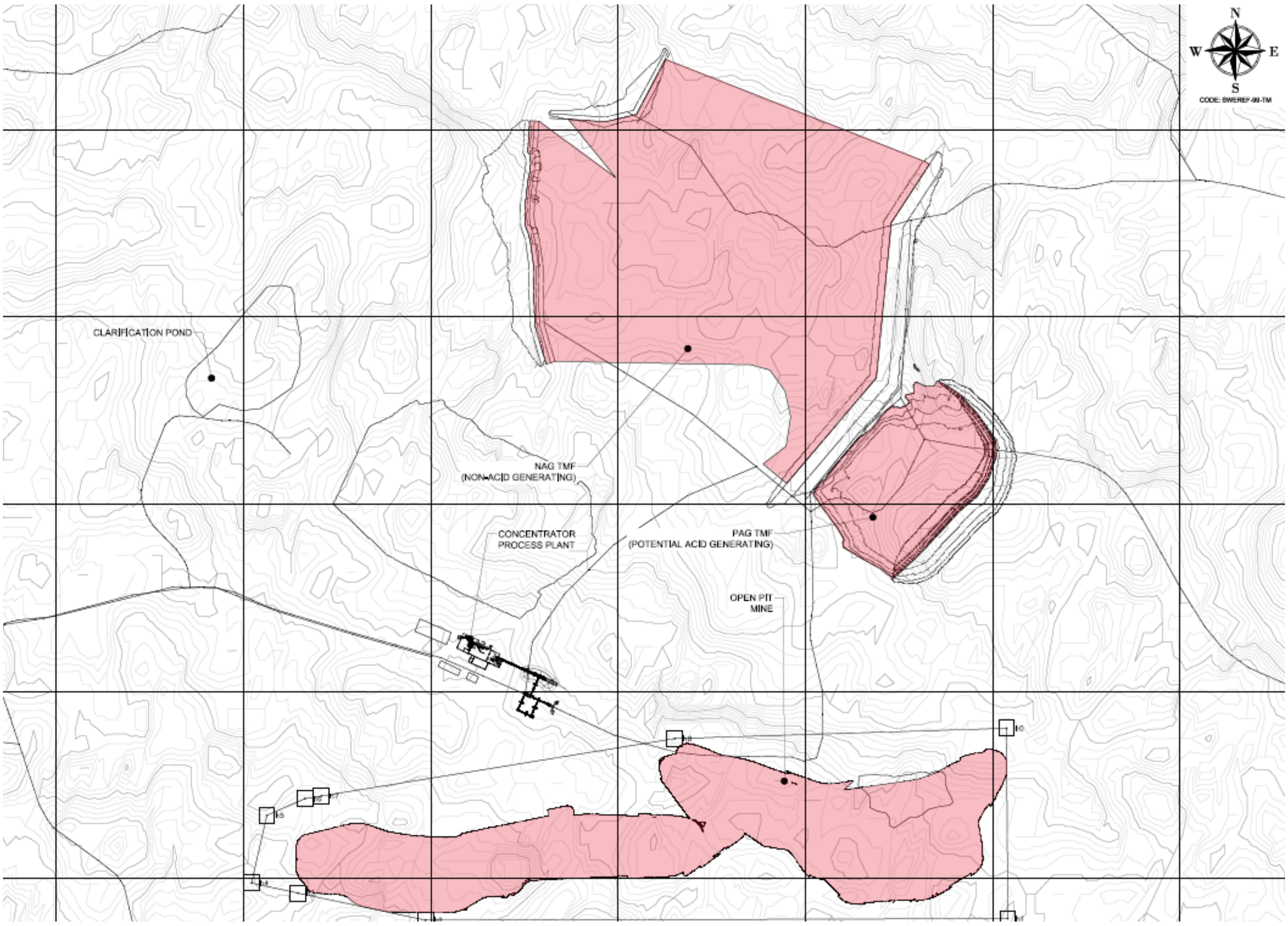


Figure 16-2: Site plan for the Kringel Mining and Beneficiation facilities

The RoM will be extracted from the open-pit within the Kringel exploitation concession. All beneficiation and administration infrastructure are located northwest of the concession area. The administration and beneficiation facilities are located out-with the concession area for which surface rights permission has been secured. The mine maintenance area is located near the concentrator offices on the access road to the mine site. The explosive storage is located north of this access road within prescribed safety distances.

16.2.6 Mine battery limits

Mining activities include mine planning, mineralisation grade control, drill and blast, excavating/loading and hauling of overburden and graphite, management of stockpiles and waste dump for the Kringel deposit only.

The Mine Battery Limits are set where beneficiation starts i.e. at the entrance of the primary crusher located nearby the Woxna Concentrator. The hauling of waste reject from the plant to the waste dump has also been included in the Mine Battery Limit.

For the purposes of the mine plan RoM is limited to the reference point where mined material is delivered to the Woxna Concentrator primary crusher or stockpiles.

16.2.7 Open pit design parameters

The Kringel tabular mineralisation has been previously mined in an open-pit which is considered to be the optimum mining method going forward.

The purpose of slope design safety analysis is to determine an optimal slope that will achieve the best safety factor as well as ensuring the best mineralisation mining recovery, and financial return in the context of maximising graphite production.

Pit slopes designs are considered excessively conservative if throughout the operation LoM no instability occurs. Pit wall stability continuous monitoring as the pit development progresses and preventive measures implementation to detect and manage slope instability through design are important industry best practice guidelines and standards.

In order to steepen the inter-ramp slopes, and minimise waste rock excavation, the pit slopes at Kringel will incorporate single benches with face height to a maximum 10 m in the waste for a safe mining operation. The graphite mineralisation benches height will depend on the morphology of the mineralisation domains and will have a maximum height of 5 m, in accordance with the size of the excavator selected for mining.

Overall slope angles achieved for the Woxna Mine open-pits will be flatter than the maximum inter-ramp angle listed below in Table 16-13, due to the inclusion of access ramps and safety berms as illustrated in Figure 16-3 below.

Table 16-13: Open-Pit Design Parameters

Parameter	Unit Symbol	Value
Maximum bench height in overburden	m	10
Maximum bench height in graphite	m	5
Face angle	°	80
Berm width	m	5
Ramp width	m	15
Ramp gradient	%	10
Final slope angle	°	62
Minimum mining width	m	40

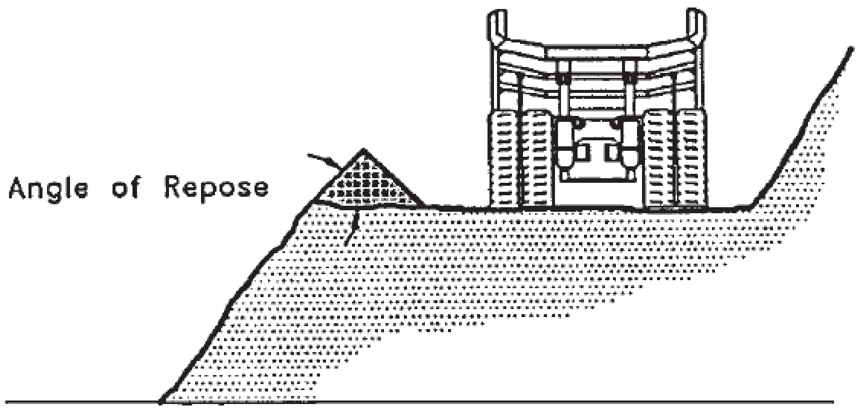


Figure 16-3: Haulage ramps and safety berms at bench edge [4]

Edge protection should be a minimum of 1.5 m or the radius of the mining and haulage equipment wheel, i.e. half the diameter of the wheel or the axle height, whichever is the larger. Safety berm heights would be a minimum half the tire height plus 300 mm. All safety berms will be examined quarterly for erosion, instability, structural weakness, or other hazardous conditions and maintenance performed as necessary.

Haulage road widths are designed to be 3 to 4 times the width of the largest haulage equipment for two-way traffic with extra width employed on the curves. Gosselin Mining recommends that haul road gradients should not exceed 12% so that mining vehicles can be loaded to their maximum payload without slowing down too much driving up in-pit ramps. Drainage cross falls should be approximately 2% to a culvert. Horizontal curves are designed to ensure the driver of the haul truck can negotiate the curve safely at a given speed. On some haul road curves design, the combination of vehicle speed and radius would result in a centrifugal force that is greater than the resisting force, i.e. mine vehicles would skid sideways. Haul road curves banking, also called superelevation, is required to reduce the centrifugal force and assist mine vehicles negotiating these curves. Superelevations and their transitions are designed to minimise the centrifugal force. At this stage, average hauling distances are estimated at 1,500 m one way.

The stability parameters for the Woxna Mine are derived from the previous mining open- pit slope stability operational experience at the mine site. No mine slope stability analysis has been undertaken using any slope stability analysis software program. This is considered to provide reliable guidance at the PEA level of definition for the Woxna Mine design.

16.2.8 Mine Waste Storage Engineering Standards

Swell or bulking factor is defined as the percentage increase in volume once rock material is removed from its original state. When excavated, rock breaks up into different size particles that do not fit together, resulting in an increase in voids (air pockets) between the rock particles as well as a reduction in weight per volume.

The only way to measure a reasonably accurate swell factor estimate of excavated rock volume is to survey the rock volume before and after excavation. Waste rock is generally coarse material but includes a wide range of sizes from very large boulders that weigh several metric tonnes to dust sized particles. In this study, a swell factor of 30% is selected for mine waste storage, which is a widely accepted average estimate for waste rock derived from blasted open-pit mine converge with industry best practice guidelines and standards for exploration and mineral resources.

Several criteria must be considered when estimating the amount of mine waste generated and waste rock storage facility. The volumes requirements for waste rock storage have been estimated for LoM from in situ volume of both waste and graphite loss produced using a swell factor of 30%.

Swell factor is also used as an adjustment factor used to rate shovel bucket capacity in loose cubic meter (LCM) to an equivalent capacity in bank cubic meter (BCM) for a given percent rock material swell. A swell factor of 40% is assigned to both waste material and RoM when loaded in trucks as listed in Table 16-4. A 40% swell is similar to 28.6% voids and a 0.714 load factor.

Table 16-14: Swell-voids-load factors

Swell (%)	Voids (%)	Load Factor
5	4.8	0.952
10	9.1	0.909
15	13.0	0.870
20	16.7	0.833
25	20.0	0.800
30	23.1	0.769
35	25.9	0.741
40	28.6	0.714

Swell (%)	Voids (%)	Load Factor
45	31.0	0.690
50	33.3	0.667
55	35.5	0.645
60	37.5	0.625
65	39.4	0.606
70	41.2	0.588
75	42.9	0.571
80	44.4	0.556
85	45.9	0.541
90	47.4	0.526
95	48.7	0.513
100	50.0	0.500

Mechanical compaction of loose rock material will usually occur, some degree after placement on the waste rock storage dump. Mechanical compaction over time is frequently caused by heavy mining machinery, soil decomposition and natural compaction. Some factors influencing the level of compaction are as follow:

- rock disposal method,
- elapsed time,
- height of the rock storage dump,
- moisture content,
- size distribution, and
- the type of material.

Based on industry best practice guidelines, total compaction estimates in an open-pit mine varies from 5 to 15%. For this study, 10% compaction factor was assumed resulting in a total swell factor estimate of 30% for waste rock in waste storage facility such as ex-pit storage and in-pit storage, i.e. in-pit backfilling.

16.2.9 RoM Pad

The RoM will be hauled to the RoM pad area adjacent to the primary crusher of the Woxna Concentrator. No blending will take place at the unloading RoM pad area. The crushing plant feed hopper and surge bins are configured so that trucks can directly unload RoM to the concentrator. RoM stockpiles next to the concentrator will be temporary storage in case of temporary interruption to processing or congestion. Any RoM feed re-handling will be done by a front-end loader.

After the crushing plant, there are two areas that will be used as RoM stockpiles.

16.2.10 Truck Cycle Time

The average vehicle speed assumptions used to develop haul times simulations are listed in Table 16-15 below.

Table 16-15: Average Cruising Speed for Vehicles Uphill or Downhill

Vehicle	Uphill (Loaded) max. 14% grade Km/h	Downhill (Loaded or Unloaded) Km/h
Scania 8x4 at 40 t tipper	15	30
Scania 10x4 at 50 t tipper	14	30

16.2.10.1 Time studies

An important input to the mining production rate is the number of complete trips a truck makes per hour. The truck's cycle time is estimated by desktop time studies and provides opportunity to evaluate the balance of the spread and job efficiency.

To further define performance of each truck (also called 'unit'), segments of the unit cycle such as haul time, dump time, etc. are estimated and included in the calculation below:

$$Travel\ Time = \frac{Distance\ (m)}{Speed\ (\frac{m}{min})}$$

Average estimated transport cycle times for both RoM and waste production levels for the LoM Schedule are presented in Table 16-16.

In order to determine cycles-per-hour at 100% efficiency, the following is applied less all wait and delay time.

$$\frac{Cycles}{hr} = \frac{60\ min/hr}{Cycle\ time} = \frac{60\ min/hr}{29.5\ min/cycle} = 2\ cycles/hr$$

Table 16-16: Average Cycle Time Estimations by Year

Equipment Estimate	Units	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Avg.
Tonnes Moved	Mt	0.45	1.39	1.07	0.86	1.00	1.02	1.00	1.01	1.01	0.98	1.06	0.94	0.87	0.66	0.51	0.43	0.07	0.00	0.00	0.00	0.73
Distance Factor	factor	0.7	0.7	0.7	0.8	0.8	0.9	0.9	1.0	1.0	1.1	1.1	1.2	1.2	1.3	1.3	1.4	1.4	1.5	1.5	1.6	1.1
Average Distance	km	0.98	0.98	1.05	1.13	1.20	1.28	1.35	1.43	1.50	1.58	1.50	1.73	1.80	1.88	1.95	2.03	2.10	0.50	0.50	0.50	1.37
Load Time	minutes	3.5	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Haul Time	minutes	3.9	3.9	4.2	4.5	4.8	5.1	5.4	5.7	6.0	6.3	6.0	6.9	7.2	7.5	7.8	8.1	8.4	2.0	2.0	2.0	5.5
Dump Time	minutes	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Return Time	minutes	2.0	2.0	2.1	2.3	2.4	2.6	2.7	2.9	3.0	3.2	3.0	3.5	3.6	3.8	3.9	4.1	4.2	1.0	1.0	1.0	2.7
Queue	minutes	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Total Cycle	minutes	11.9	11.3	11.8	12.2	12.7	13.1	13.6	14.0	14.5	14.9	14.5	15.8	16.3	16.7	17.2	17.6	18.1	8.5	8.5	8.5	13.6
Cycles / Hour	#	5.1	5.3	5.1	4.9	4.7	4.6	4.4	4.3	4.2	4.0	4.2	3.8	3.7	3.6	3.5	3.4	3.3	7.1	7.1	7.1	4.40

16.2.11 Mineral resource depletion

In Sweden, test mining is defined as advanced exploration work and means that the soil layer is removed, and the bedrock is exposed in an area within a near-surface deposit. A certain amount of the rock is taken for concentration tests to establish the quality of the deposit.

The exploration also provides further input for assessment of the environmental factors to limit the environmental impact and costs of remediation and disposal of waste products. Test mining is a one-time activity that involves sampling a large volume of mineralised graphite rock in order to be able to assess the technical concentration properties of the graphite mineralisation.

Woxna Graphite undertook a test mining exercise in 2014 as illustrated in Figure 16-4, Figure 16-5, and Figure 16-6.

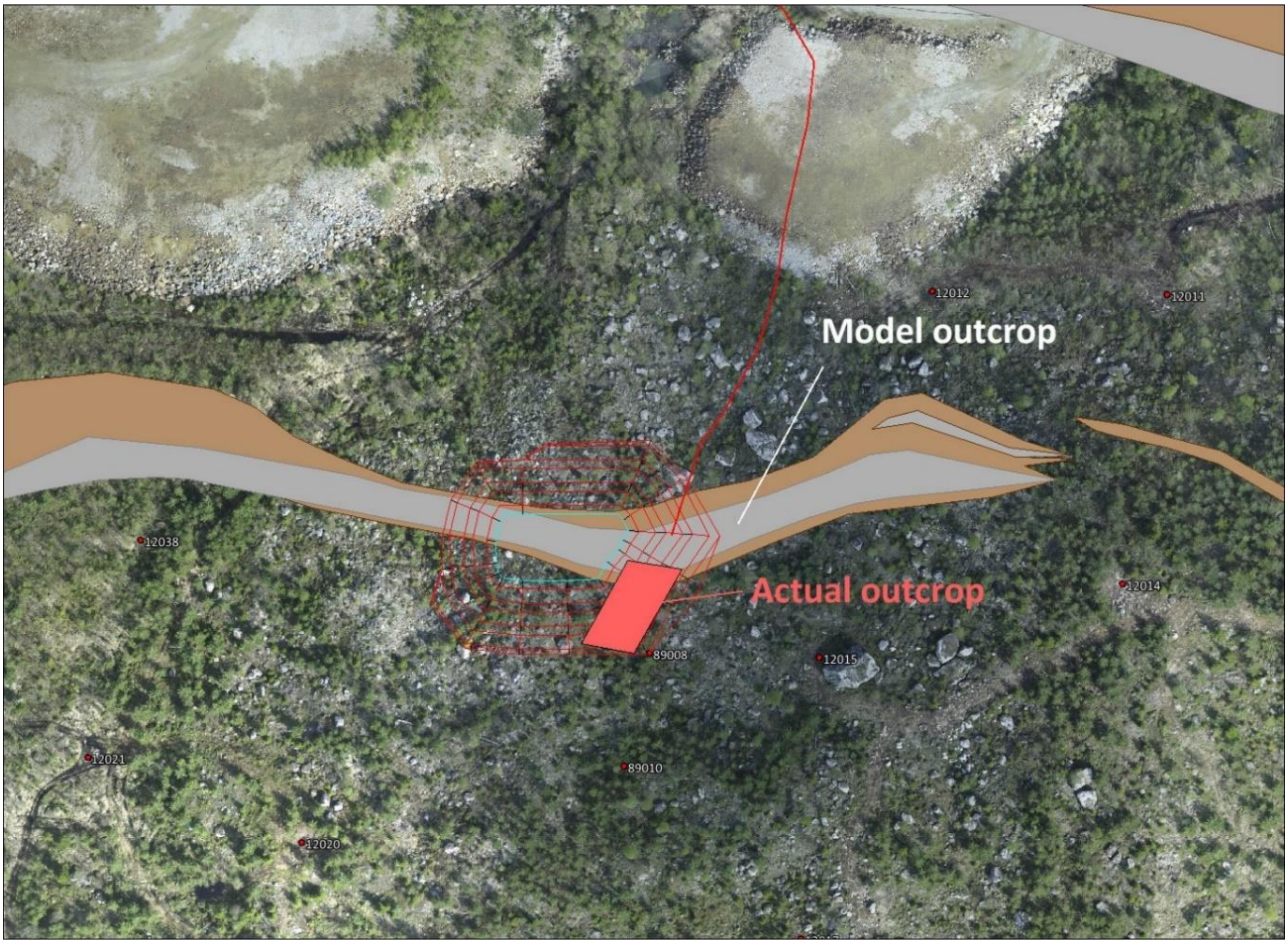


Figure 16-4: Test mining campaign pit excavated in 2014.

All of the excavated mineralised material was moved to the concentrator and there was no distinction made between A and B types of RoM

The overburden average thickness over the test mining area was not measured or recorded. Overburden was not difficult to remove using mainly with a backhoe excavation machine. Not all the overburden had to be drilled and blasted but only some large boulders in the moraine that were too large to move with the equipment.

In order the excavate the mineralised material that would constitute the RoM some drill and blast was required.

There was also no water present in the test mining pit.

Figure 16-5: Equipment used during test mining (left) and mineralisation hangingwall contact on the (right)

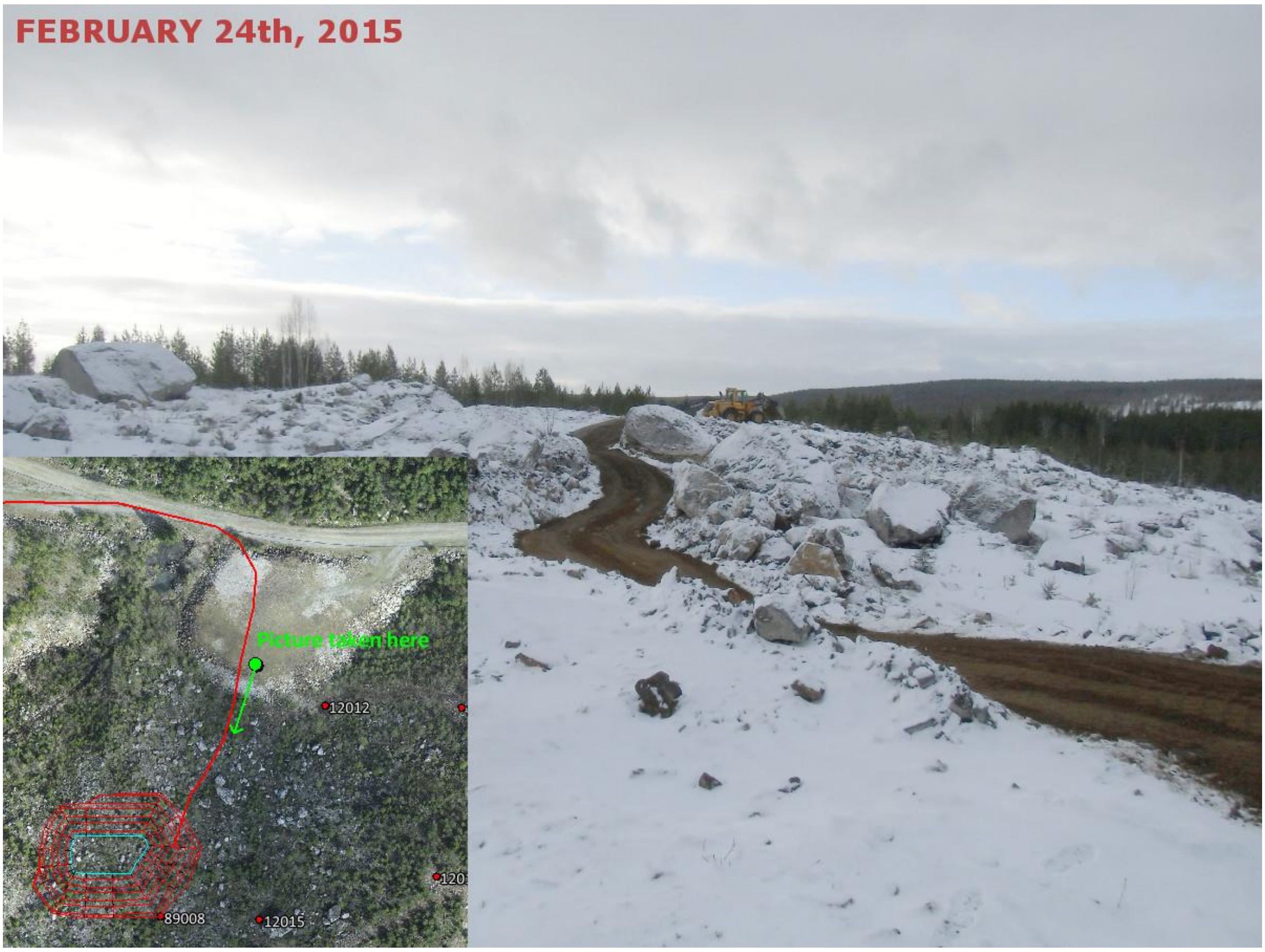


Figure 16-6: Test mining campaign photo looking south to the pit area overburden

The mineralisation in the hangingwall and footwall contact are observed in Figure 16-7 and Figure 16-8 below.

Figure 16-7: Mineralisation outcrop of hangingwall and footwall (looking south)

Figure 16-8: Mineralisation outcrop of hangingwall and footwall (looking north)

The block model provided by ReedLeyton was depleted with the material removed in the test mining campaign undertaken in 2015 which amounted to 4,035 t at a grade of 7.9% C, to produce 296 t product.

16.2.12 Rock breakage

Based on high UCS values for waste and graphite+gangue and the presence of hard rock, Drill and Blast operations will be required before graphite+gangue excavation (except for the surface miner method but with consequences on equipment degradation). Drilling is planned to be performed using conventional down-the-hole drill rigs. Blasting will use commercial bulk explosive with down-hole delay initiation.

Some limited areas having weathered overburden with low thickness scheduled for early mining, could be extracted using bulldozer blade and ripping device at a lower cost. Material will then be hauled and stored in a topsoil storage area outside the pit. With the exception of the surface moraine overburden material, ripping is not achievable.

16.2.13 Shovels or front-end loader and trucks

This conventional open pit method uses shovels or front-end loaders, supported by bulldozers, to load mining vehicles with either waste or graphite+gangue refer Figure 16-9.

Figure 16-9: Hydraulic shovel loading mining truck

Drill and blast will be necessary prior to excavating/loading and hauling due to the waste high strength rock classification and the graphite+gangue hardness's.

Waste removal and graphite+gangue selective mining will be achievable. This mining method has high operating costs (due to fleet size, maintenance, spare parts, tyres) which are balanced by the fact that smaller mining equipment is able to achieve the plant planned production rates.

A shovel advantage is its strength and ability to excavate poorly blasted, irregular and tougher material better than draglines, BWE or loaders. Harder strength rock material is commonly encountered at the Kringel deposit such as calcareous quartz-rich meta-tuff, with interbedded metasedimentary units and cross-cutting pegmatite and would therefore require shovel excavator that would be able to handle these higher rock strength mining areas.

Shovels are flexible enough to be used for graphite+gangue mining due to their excellent ability to selectively excavate different grade domains and accommodate, fault planes [5] or different graphite mineralisation types.

Dozers can extend the range of shovels when operated in tandem. Loaders are more mobile and are applied to rehandling at the stockpile. Graphite+gangue selective mining is done using backhoes with smaller bucket sizes allowing mining height to be adjusted according to grade variations, geological losses, or dilution, i.e. mine geology grade control.

The flexibility and selectivity of the shovels are an advantage in cases of changes in the mining plan, especially in the first years of mining which may be confronted with unforeseen geological conditions in the graphite mineralisation. Trucks and shovel surface mining method main advantages and disadvantages are listed in Table 16-17.

Table 16-17: Trucks and shovels advantages and disadvantages comparison

	Advantages	Disadvantages
Production	Adapted to handle hard material	Higher traffic
	Flexibility with mine plan, hauling	
Economic	High flexibility	High operating costs,
	Competitive subcontractor or supplier market	Auxiliary mobile equipment necessary for roads maintenance, truck maintenance, etc.
	Lower CAPEX compared to dragline/BWE	
	Low investment risk, well known mining method,	
Selectivity	High selectivity, different sizes available	

Mining trucks are the preferred equipment to be coupled with shovels for hauling waste and graphite+gangue material and generating ex-pit Overburden Storage Facilities (OSF), in-pit Overburden Backfill, RoM pad stockpiles or direct dumping at the primary crusher. This will allow more flexibility in following mining fronts and adjusting to volumes hauled. During the mine operating shifts, the high equipment traffic requirements on operating and travel surfaces due to reliance on truck haulage, requires more support equipment (surface water diversion, dust suppression, haul road maintenance and safety berms maintenance).

16.3 Mining Methods comparison

The main advantages and disadvantages of the different alternatives are listed and compared in Table 16-18.

Table 16-18: Comparison of Mining Methods

Mining Method	Advantages	Disadvantages
Trucks and shovels	Flexibility	High operating cost (fleet size, parts, fluids…),
	Low investment cost, low risk,	High traffic and support needed
	Contractor/Supplier easy to find,	
Surface Miner	High selectivity,	High maintenance costs due to peak wear on hard rock,
	Road trucks suffice,	Surface bench preparation required
	No need for drill and blast,	Dust and finer-grained graphite issues
	No need for primary crusher.	
Dragline	Production capacities	Not adapted to graphite thickness variations
	Low maintenance	Low flexibility and mobility,
	No hauling costs	High risk due to high CAPEX investment
		More difficult to find a contractor
Bucket Wheel Excavator	Continuous operations	Not adapted to graphite thickness variations
	Overburden and graphite	Low flexibility and mobility
		High CAPEX investment, i.e. high risk
		More difficult to find a contractor
Sublevel Stoping	Least Environment Impact	High level of development
	Low Blast Costs	High Early Capex
	Good Recovery	Low Early Revenue
	Low Dilution	Not High Selectivity

In conclusion, Dragline and BWE represent high investments and high risks and are not adapted for hard rock deposits with only a few meters graphite mineralisation thickness with faults resulting in increased graphite losses. Both methods, as well as the surface miner method, are not the most suitable methods for hard rock strength and lack the flexibility required for a mining operation with small planned production rates and minimum RoM feed grades.

Conventional drill and blast combined with trucks and shovel mining method have been selected as the optimum method for Woxna Mine based on the following advantages:

- Well adapted to rock material characteristics (variable waste thickness, high rock strength material, faults between the mineralisation domains;
- High flexibility and reactivity to change in planned production rates, mine plan, hauling destinations, economic parameters;
- High selectivity, with a large range of bucket sizes;
- Low initial capital expenditure (CAPEX) investment and ease to find a local subcontractor or supplier;
- Low risk for a well-known mining method that has been previously used at Woxna Mine.

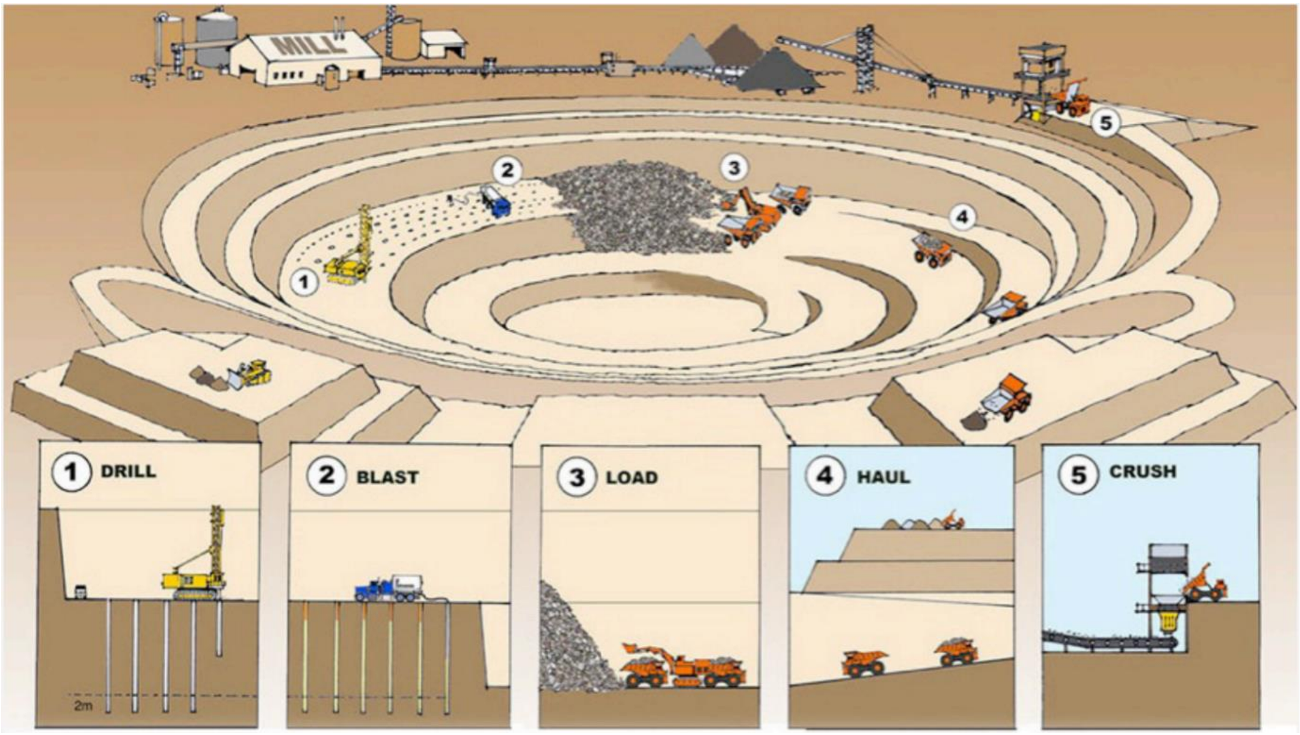


Figure 16-10: Open pit with multiple benches selected mining method: conventional drill and blast with trucks and shovels.

16.4 Selected mining method

16.4.1 Drill and blast

Explosives will be used to fragment both waste and graphite+gangue. Drilling depth will vary depending on the actual overburden and the graphite mineralisation depth for different mining areas over the mining property. Multiple benches are used to handle overburden.

The commercial explosive selected for blasting is ammonium nitrate-fuel oil mix (bulk ANFO) due to its relatively low cost, ease of loading properties. Standard ANFO is a mixture of pelletised industrial grade ammonium nitrate and 5.7% diesel fuel oil resulting in a product density of 0.85 g/cm^3.

Loading of explosives will be undertaken using a custom-built bulk pumping truck vehicle. All production holes are expected to be wet due to the high groundwater table in the mine site area. Blasting agents will not function properly if placed in wet drillholes for extended periods of time. Blasts will be fired using a non-electric down-hole delay initiation system. Production blasts patterns will be blasted at fixed times during the day. Notification of blasts will be performed as per Swedish regulations and an alarm will always sound prior to production blasts several minutes in advance.

The typical geometry and terminology of a blast pattern are schematically represented in Figure 16-11 and Figure 16-12.

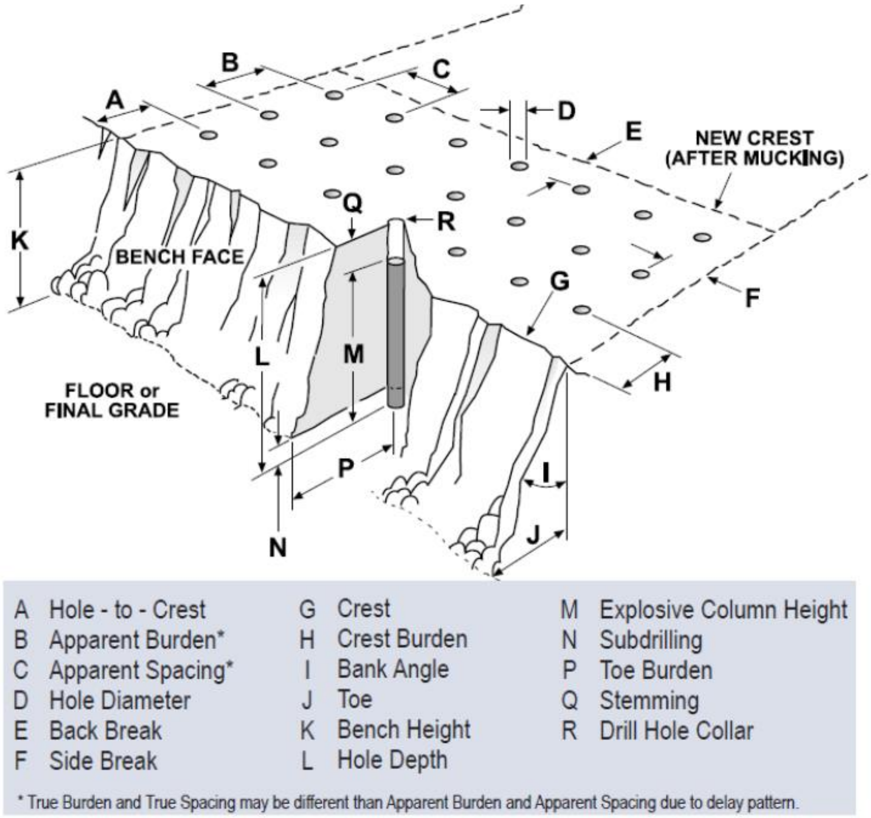


Figure 16-11: Blast Design Terminology [6]

Source: Dyno Nobel, 2020.

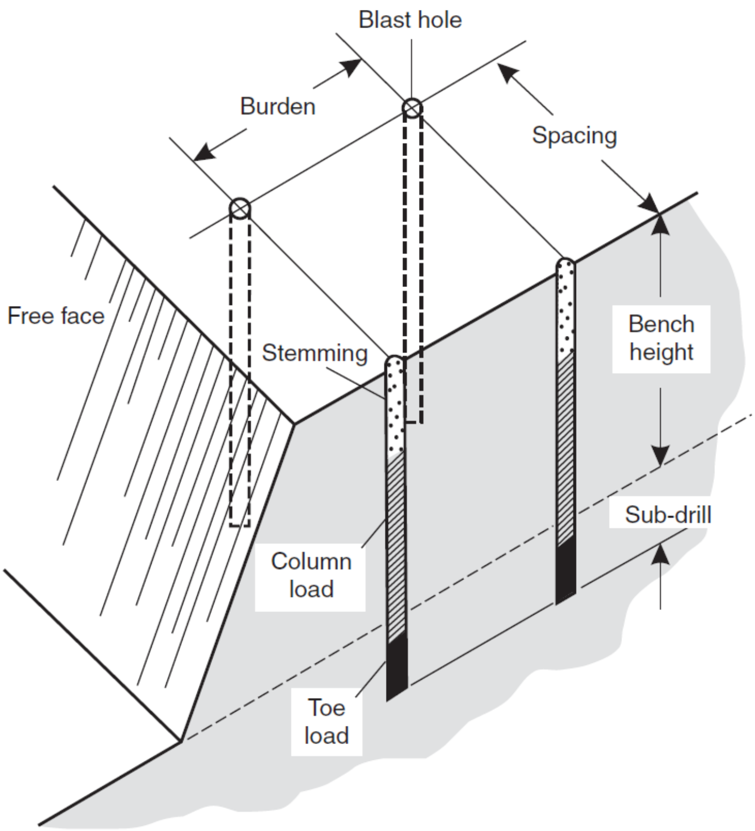


Figure 16-12: Definition of bench blasting terms [7]

Source: Rock Slope Engineering, Wyllie and Mah, 2004.

Surface drilling rigs and dry blasting agents will be used. Surface drilling rig equipment and dry blasting agents' specifications are as follow:

- blasthole diameters: 115 mm for graphite and 165 mm for waste;
- prilled Ammonium Nitrate Fuel Oil (bulk ANFO), e.g. Prillex;
- cartridges, e.g. Kemulek;
- detonating cords;
- boosters;
- stemming material, i.e. crushed gravel size between 9 mm to 20 mm diameter;
- bulk mixing and pumping truck; and
- dry blasting agent storage.

The drilling speed is function of the diameter of the drillholes, which is controlled by the height of the benches. Smaller benches require smaller drillholes and fewer explosives. The higher thickness hangingwall waste allows 165 mm drillholes to be drilled, except for the first years of mining where overburden thickness is reduced and smaller diameter drillholes such as 115 mm shall be sufficient. Production holes of 165 mm in diameter have estimated drill rates of circa 25 m/h whereas graphite drilling rates with 115 mm holes have estimated rates of circa 30 m/h. The higher drilling rates are somewhat compensated by the higher number of drillholes needed due to smaller diameters.

Rock drillability is determined by several factors led by mineral composition, grain size and brittleness. In basic terms, rock compressive strength or hardness can be related to drillability for rough calculations, but it is usually more complicated.

The Norwegian Technical University has determined more sophisticated methods: The Drilling Rate Index (DRI) and the Bit Wear Index (BWI). The DRI describes how fast a particular drill bit can penetrate. It also includes measurements of brittleness and drilling with a small, standard rotating bit into a sample of the rock. The higher the DRI, the higher the penetration rate, and this can vary greatly from one rock type to another, as shown in Figure 16-13. The lower the DRI value, the more difficult it is to bore the rock.

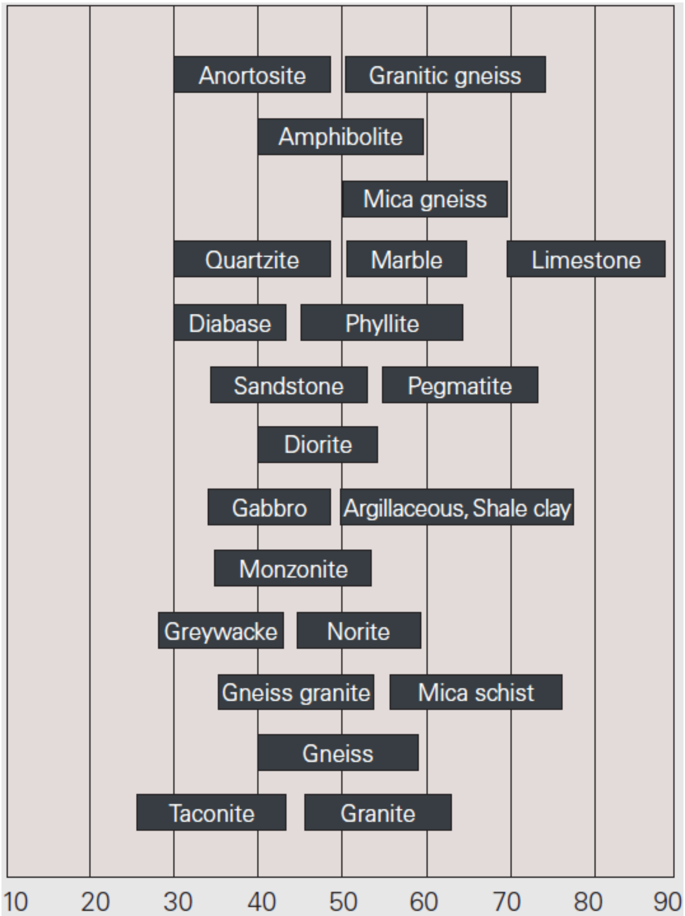


Figure 16-13: Relationship between Drilling Rate Index (DRI) and various rock type

Source: Rock Excavation, Dr. Sean Dessureault, 2004.

The DRI is assessed on the basis of two laboratory tests, the Brittleness Value S_{20} and Sievers' J-value SJ. The DRI is derived from a chart between both laboratory tests. The following intervals, presented in Table 16-19, are recommended for the classification categories of DRI.

Table 16-19: Drilling Rate Index (DRI) classification categories

Category	DRI
Extremely low	≤ 25
Very low	26–32
Low	33–42
Medium	43–57
High	58–69
Very high	70–82
Extremely high	≥ 93

According to Figure 16-13 and Table 16-19, the prevalent lithologies encountered correspond to DRI classification category from low to medium.

16.4.2 Waste removal

Waste material comprises meta sediment with calcareous quartz-rich hangingwall and footwall of the graphite mineralisation as listed in Table 16-2. The minimum, maximum and average thickness as well as in situ wet/natural density of the total waste horizon is approximately 0.5 m, 65 m, 30 m and 2.7 t/m^3. The upper most overburden layer composed of Quarterly age moraine have a thickness ranging from 0.5 m to 20 m and an average thickness of 3.5 m as listed in Table 16-2.

Waste excavation will progress ahead of the graphite+gangue excavation in maximum 10 m face height mining production benches for safe operational purposes. Because the waste thickness is generally less than or equal to 60 m within the open pit, multiple waste mining benches will be developed and maintained ahead of the graphite+gangue excavation. Bench height and mining depths will vary depending on the actual waste rock thickness in different mining areas over the deposit. Waste mining fronts will be mined approximately 500 m in advance to graphite+gangue mining fronts where possible.

Front-end shovels are best adapted to handle larger boulders from blasted rock heaps at mining front in moraine and bedrock geological layers. Shovels and trucks will be at the same level corresponding to the base of the blasted heaps.

Waste excavated while developing the starter pit will be stored outside the pit, i.e. ex-pit, until the starter pit area is large enough to allow back-hauling waste in the mined-out starter pit volume without impeding mining activities. Waste rock stored outside the pit will be hauled to an ex-pit Waste Storage Dump (WSD) In-pit backfilling areas will become available in the east pit after year 11 of planned mining production. The west pit may be backfilled with waste at the end of the LoM as desired and/or required.

16.4.3 Graphite mineralisation mining

Grade control in an open-pit mine involves sampling of blasthole cuttings produced by from the production drillhole cutting. The detailed implementation of grade control typically consists of sampling and assaying to determine the quantity and location of the mineralisation and then defining economic mining zones or mineralisation type, i.e. A-type or B-type.

The target material at the Kringel deposit has a rock strength too high for free-digging and requires breakage prior to mining excavation. Graphite mineralisation bench height will average 5 m but will vary, as the mining deepens, depending on the actual graphite domain thickness and overburden thickness, for different mining areas over the deposit.

Backhoe shovels are preferred for handling the RoM and mineralised faulted material. The size of the equipment will be smaller compared to that used for the overburden, allowing more selectivity for the different graphite A-type and B-type. The shovels will be located on top of the graphite mineralisation and excavated material loaded onto the trucks at the base level of the bench allowing additional loading height. The shovels can be supported by bulldozers to enlarge their range of action.

RoM hauled using mining trucks directly to the primary crusher located at the Woxna Concentrator or stored at the adjacent RoM pad. Average hauling distances are estimated to 1,388 m one way.

The RoM will be directly tipped into the primary crusher from dump trucks or else stored on temporary RoM pads. Re-handling of the stockpile is estimated to not exceed 20% and will be undertaken with a front-end loader.

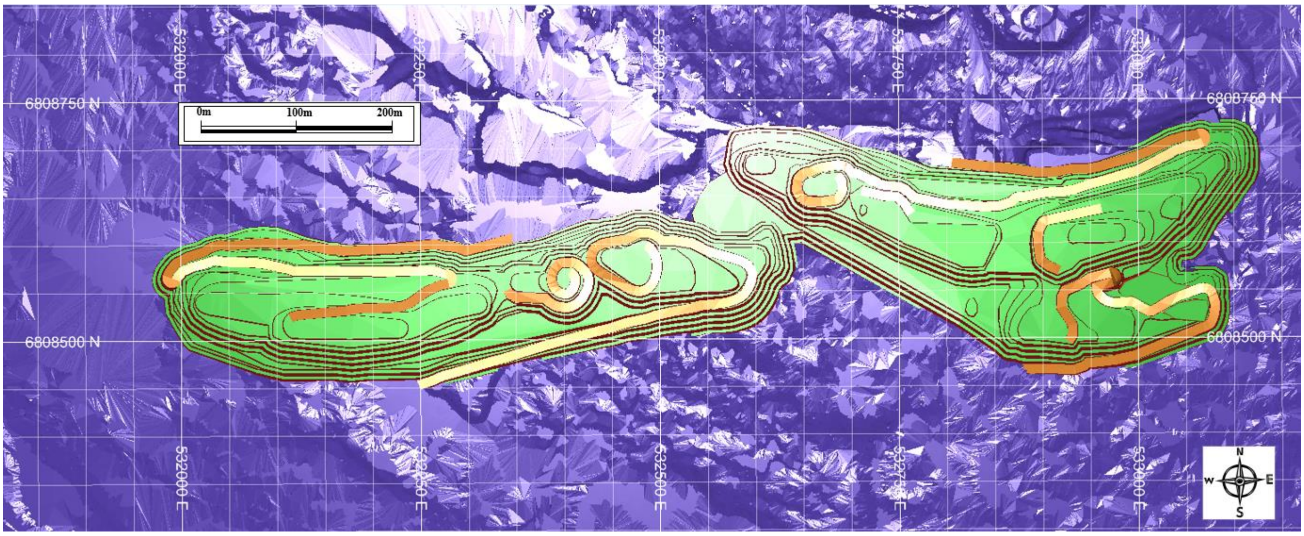


Figure 16-14: Pit Designs used as basis for mining schedule

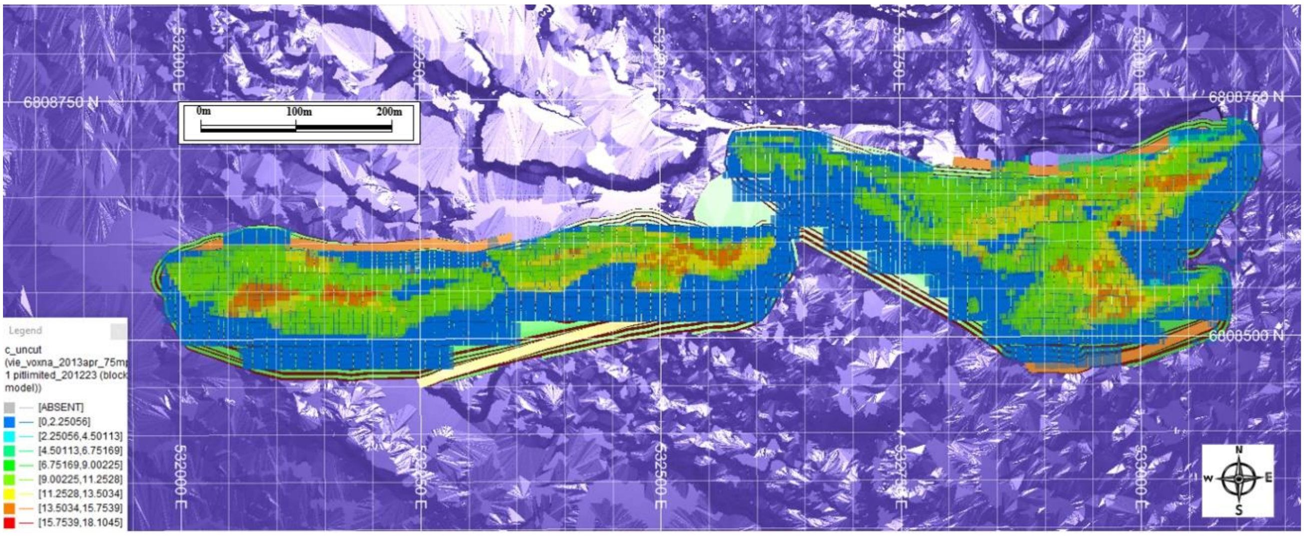


Figure 16-15 Pit design limited block model showing %carbon grades

16.5 Open pit optimisation

16.5.1 Methodology

The Woxna Mine LoM production plan was scheduled from the most economical part of the deposit to the lesser economical areas. The mine plan was developed;

- utilising NPV Scheduler™ (NPVS) pit optimisation software on the geological 3D block model;
- the NPV Scheduler preliminary results were matched to the mine plan starting point and mining sequence using different Revenue Factors (RF); and
- and the mine plan was then scheduled in Datamine™.

The initial step in the preparation of the mine plan was the completion of a series of pit optimisation runs using NPV Scheduler™ optimisation capabilities.

The second step involved the generation of a series of nested pits, in which each pit is optimal for a different set of economic criteria. The nested pits were produced by parametrising the graphite product selling price by varying the RF in NPV Scheduler™. The innermost pit in the nested series has the highest value per tonne and the second innermost pit has the next highest value per tonne and so on. The purpose is to present a mining production plan in which nested pits show the gradual open pit mine development of the Kringel mineralisation in accordance with the economic growth and price increase. In practice each nested pit shell could be a singular open pit mine representative of a certain economic condition.

The NPVS optimisation for Kringel mineralisation was based on Measured and Indicated Mineral Resources with a 4.2% C RoM cut-off applied. In previous disclosures the previously applied mineral resource cut-off grade was minimum 7% C grade. No criteria such as a 1 m minimum graphite mineralisation thickness was applied to the optimisation. The exploitation was applied at a later stage when estimating the mine production plan. A mining depth boundary of 70 m from current topography was applied in the optimisation.

At the current PEA level of accuracy of the mine and processing design, no Mineral Reserves have been defined as the basis for the open pit optimisation The NPVS optimisation for Kringel mineralisation was therefore based on Measured and Indicated Mineral Resources to which a 4.2% cut-off grade was applied that represents the break-even cut-off grade at which RoM can be mined and processed at an operating profit, considering all applicable costs, i.e. produce a positive gross margin, with revenues higher than costs. The proposed mine plan at this PEA level estimates a lower average RoM grade than cut-off grade used in mineral resource estimation by taking into account mining dilution factors, presence of geological faults and RoM mining selectivity that could be achieved with planned mining machinery.

A NPVS economic model was developed to inform the NPVS optimisation. The economic model defines the various NPVS inputs developed and determines how the optimisation uses them in the optimisation process.

The economic model includes information regarding the number of optimisation iterations and how they are calculated. The number of optimisations is controlled by Revenue Factors (RF), which was calculated over the range of 0.32 to 1.0 (in steps of 0.02). It multiplies all the prices by a series of 35 RFs ranging, from 0.32 to 1.0, and produces a pit outline for each. A set of nested pit outlines is obtained when the graphite product selling prices are step through a series of values, doing an optimisation for each.

The RF is used to determine open pit limit sensitivity to graphite price. The RF range approach is a simple way to scale values to produce a range of optimal open pit shells for different graphite product prices. This is a powerful tool for mine planning as commodity prices generally are subject to a high degree of uncertainty over time. The RF applies to graphite price only and no costs are factored in the RF.

The goal with producing outlines of the smaller RF values is to produce inner pit shells to point out the best areas to start mining as well as to schedule a profitable mine sequence over time. It is simple to assess what mine sequences are suitable when mining out a specific nested pit shell since all the nested pit shells outlines follow the overall slope angle requirements. The inner pits shells are helpful in sequencing and targeting the best starter pit and following mining sequence to generate highest value early on in the LoM.

The RF range is relative to the defined average graphite product price, i.e. RF 1.0 = USD 2,207 /t. For example, RF 0.5 uses a graphite price of USD 1,103.5 /t and RF 1.2 is using a graphite price of USD 2,648.4 /t.

The last step undertaken in Datamine™ was the annual scheduling of the LoP production.

16.5.1.1 Geological Block Model

MPlan has undertaken open pit optimisation using the mineral resource block model prepared by ReedLeyton Consulting.

NPVS was used to develop optimal pit geometries using the Lerchs-Grossmann (LG) algorithm with the mineral resource block model using preliminary parameters and assumptions. The optimised open pit(s) geometries delimit a mineral resource and are used to generate a LoP production plan.

16.5.1.2 Validation

Validated geological models were provided by Woxna Graphite, and the information has been assumed to be accurate by MPlan. MPlan did not update or validate the previous mineral resource block model. MPlan conducted a simple review of the geological block model "vie_voxna_2013apr_75.bmf" solely to understand the mineralisation domains and the variables used to determine the mineral resource estimate. MPlan did not audit the Kringel mineral resource estimate provided.

MPlan was supplied with a three-dimensional (3D) block model in Maptek Vulcan™ format in SWEREF99 datum and coordinate reference system used in Sweden. The geological model provided did not contain air blocks which had been previously removed.

Manipulation of the geological model was required in preparation for input into Datamine™ NPVS software to undertake pit optimisation. Modelling steps included:

- Importation of the supplied geology block model in comma-separated values (CSV) file format into Datamine™ NPVS and comparison of tonnes and grade to verify conversion; and
- Creation and population of additional model variables used in the optimisation process;

The following densities for waste, graphite and overburden material presented in Table 16-20 are from the technical report published in 2015.

Table 16-20: In situ wet density by material type in geological block model "vie_voxna_2013apr_75.csv"

Model Lithological type Attribute abbreviation	In situ wet Density t/m3	Material
Bmtu	2.7	Banded meta-stuff
Grf (B type graphite)	2.7	Graphite mineralisation
Hg (A type graphite)	2.7	Higher grade graphite mineralisation
Msed	2.7	Meta-sediment
Ovb	1.8	Overburden
Peg	2.7	Pegmatite

The in situ wet density of the overburden at 1.8 t/m^3 was changed to 2.3 t/m^3 because of evidence of high strength and boulder-rich moraine with more than 40% fragmented block content while digging three trenches within the planned open pit perimeter in 2011 as presented in Figure 16-16 through to Figure 16-20 below.

The measured physical properties of different soil samples taken from four excavated pits, near the Woxna Mine TSF listed in Table 16-21, indicate that the glacial till moraine has an average in situ wet/natural density of 2.3 t/m^3.

Table 16-21: Physical properties of different moraine samples sampled near Kringel TSF

Pit	Depth m	Saturated weight g	Unsaturated weight g	Water content %	Saturated density t/m^3	Volume cm^3
2	0.7	88.2	72.3	13.7	2.24	36.67
2	1.4	88.8	80	11.0	2.31	38.43
3	1.4	84.6	72.9	16.0	2.19	38.70
4	3.2	84.4	79.3	6.4	2.45	34.47
4	4.5	88.1	82.4	6.9	2.43	36.26
5	2	83.6	76.7	9.0	2.37	35.31
5	4	92.1	83.9	9.8	2.35	39.27

Source: D. Barkels and J. Åberg, 2011 [8]

The overburden glacial till is a boulder-rich moraine in fragmented blocks and boulders on and below the ground which is also partly also covered by soil and proves very. The moraine depth around the open pit varies between 1–4 m, approximately 2.5 m in average thickness.

In low relief Precambrian gneiss terrain in eastern Sweden, abraded bedrock surfaces were ripped apart by the Fennoscandian Ice Sheet. and the resultant boulder-spreads are covers of large, angular boulders, many with glacial transport distances of 1–100 m as illustrated in

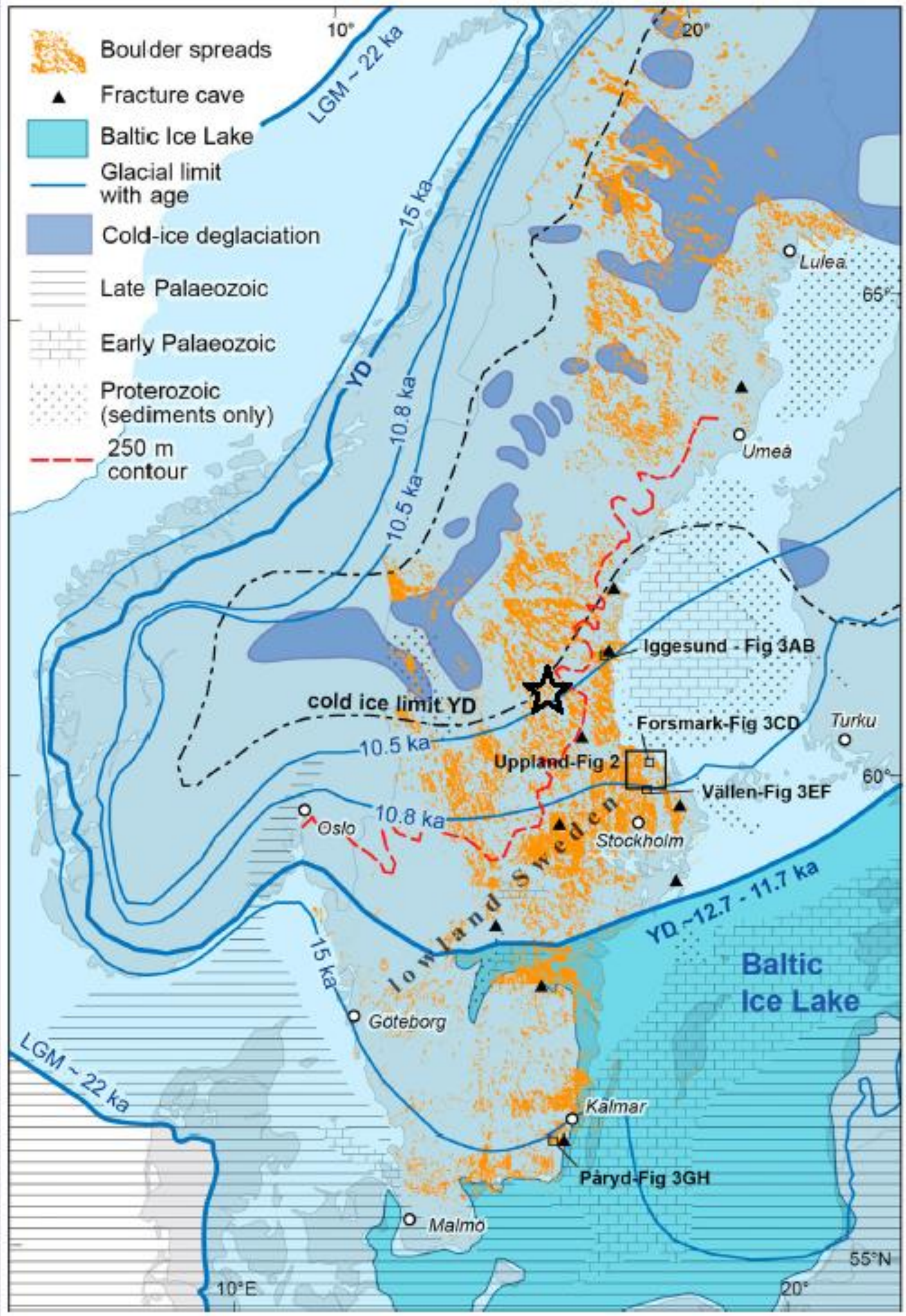


Figure 16-21Figure 16-21 below.

Figure 16-16: Pit 101. Block over

Figure 16-17: Pit 105. Graphite block

Figure 16-18: Pit 104. Note the amount of blocks

Figure 16-19: Blockmark on the south side of the mine

Figure 16-20: The open pit. View towards east.

Figures 16-16 through 16-20 Source: C. Mattsson, 2011

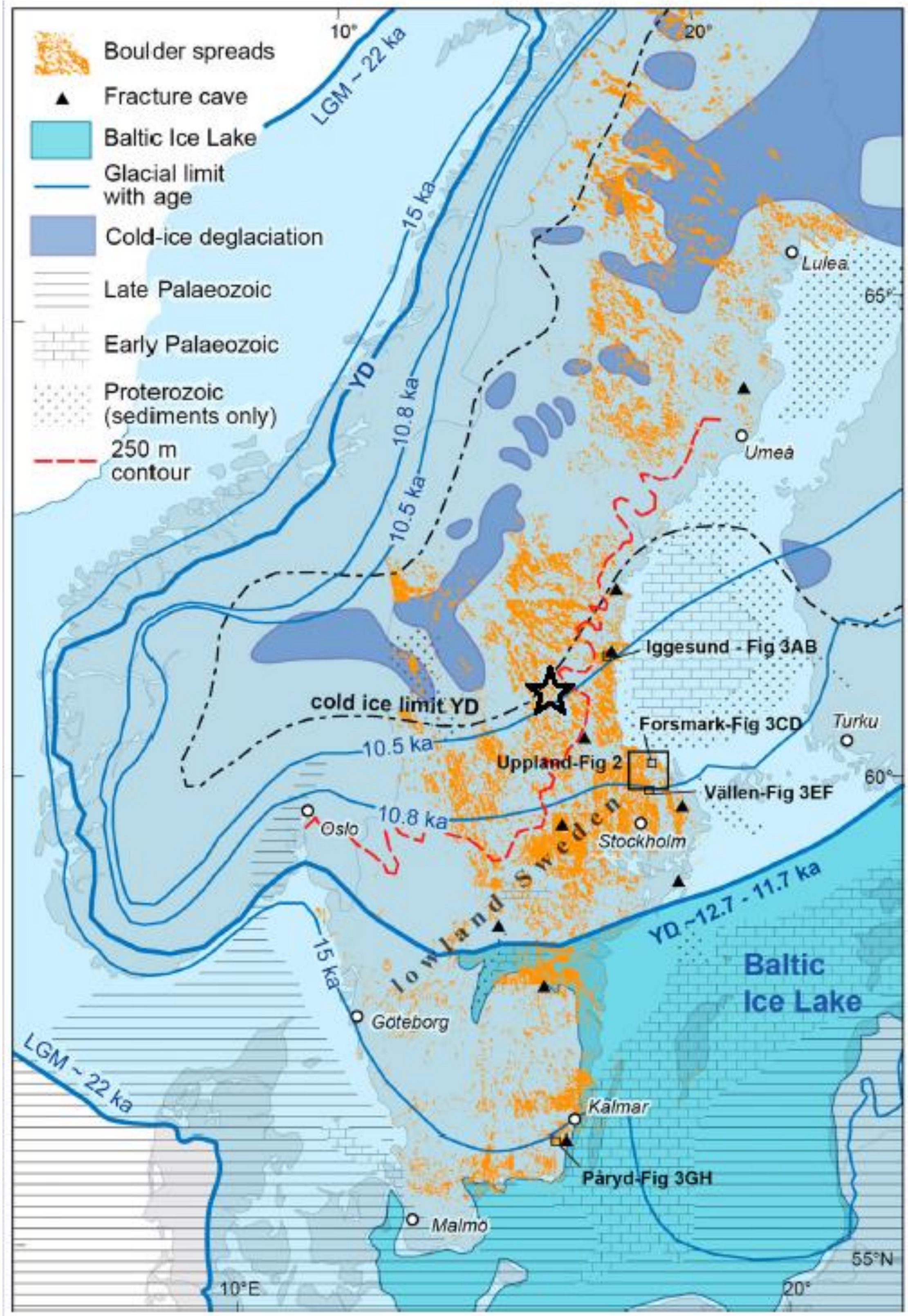


Figure 16-21: Overview map with boulder-spread in orange and Kringel deposit approximate location represented by a black star (Gosselin Mining, 2021)

16.5.1.3 Block Model

Woxna Graphite supplied the following geological block models in two different file formats as follow:

- vie_voxna_2013apr_75.bmf (Vulcan file format)
- vie_voxna_2013apr_75.csv (CSV file format)

Table 16-22: Kringel Geological Block Model parameters.

	X Coordinate (Easting)	Y Coordinate (Northing)	Z Coordinate (RL)
Model origin	531800	6808250	0
Block size (m)	5	25	5
Sub Block size (m) (minimum)	1.25	6.25	1.25
Rotation	90 (Bearing)	0 (Plunge)	0

Table 16-23: Geological Block Model Key Variables.

Variable Name	Description
Lithtype	Banded meta-stuff, graphite mineralisation, fine grained graphite mineralisation, meta-sediment, overburden or pegmatite
C_uncut	Carbon
Bd	In situ wet/natural bulk density
Category	Resource category (inf, ind or meas)
Rsc_cat	0 non classified, 1 measured, 2 indicated, 3 inferred and 4 additional mineralisation
Mined	In-situ or mined
Volume	Block volume
Graphite tonnes	Tonnes

The open pit optimisation is based exclusively on the in situ Measured and Indicated Mineral Resources in this geological model.

16.5.1.3.1 Block model verification

A manual check was made comparing block values to the corresponding CSV block model. Slices were made through the block model and tonnes and grade were interrogated for several blocks.

16.5.1.4 Optimisation block model

The final optimisation block model, vie_voxna_2013apr_75.dm, tonnes reported by material type is included inTable 16-24. The optimisation block model has been confirmed as correct by Reedleyton and MPlan as part of the data handover.

Table 16-24: Optimisation block model global tonnes

Resource Category	Tonnes
Overburden	16,671,069
Waste	557,832,956
Measured	2,043,616
Indicated	4,116,050
Inferred	959,291

Resource Category	Tonnes
Additional mineralisation	38,285

The total measured and indicated mineral resources tonnage in the model is 6.159 Mt.

The Datamine™ optimisation for Kringel deposit was based on Measured and Indicated mineral resources only with no cut-off applied.

The Datamine™ model vie_voxna_2013apr_75.dm was used for open pit optimisation. Re-blocking in Whittle is used to bias the pit optimisation to a realistic mineable block size.

Variables defining the waste, mineral resources tonnes for each block were added to the block model.

In order to run scenarios in Datamine™ it was necessary to add a new calculated variable into the model for the total tonnage of the block.

- TOTN: Total tonnage of the block = density x volume of the block

16.5.1.5 Datamine™ Surfaces and Solids

The surface and geological boundaries were used as is, as per the supplied Vulcan model.

All blocks in the supplied Vulcan block model had a 2.7 t/m^3 value as in situ wet/natural density. The density value of the blocks coded as "ovb" was therefore changed to 1.8 t/m^3 to more accurately represent the overburden tonnage.

16.5.2 Optimisation Parameters

All the parameters used for the open pit optimisation in NPVS are provided below and were adjusted and updated as the PEA progressed.

In NPVS optimisation any expenditure that would stop if mining stopped must be included in the cost of mining, beneficiating, or selling graphite. The costs in this study were developed considering; direct mining costs, beneficiating costs, selling costs, and overhead or time dependent costs.

Table 16-25: Optimisation Parameters

1	Open Pit Optimisation Parameters	Unit	Value
1.1	Direct graphite mining costs	USD/t of graphite	4.51
	Direct waste mining costs	USD/t of waste	4.51
1.2	Processing plant costs	USD/t RoM	84.18
1.3	Processing recovery	%	93.7
1.4	Processing plant throughout	Mt/a	160,000
1.5	USD:SEK exchange rate	USD:SEK	1:8.75
1.6	Average selling price	USD/t of product	2,207.00
1.6a	Average selling price	USD/t of product	1,800.00
1.6b	Average selling price	USD/t of product	4,300.00
17.0	Total Cost + Royalty	USD/t	11.71
1.7a	Average selling costs	USD/t	7.30
1.7b	Average selling costs	USD/t	7.30
1.8	NPV discount rate	%	10
1.9	Mineral compensation	%	0.20
1.10	Mining dilution	%	2.50
1.11	Mining losses	%	2.50

1	Open Pit Optimisation Parameters	Unit	Value
1.12	Inter-ramp slope angle	Degree	55.00
1.13	Swell factor	Percent	30.00
1.14	Graphite density	t/m^3	2.70
1.15	Waste density	t/m^3	2.70
1.16	Overburden density	t/m^3	2.30

16.5.2.1 Geotechnical overall slope angle

An overall slope angle of 55^0 was applied based on the existing open-pit for a high wall design incorporating 5 m high double bench, 2.5 m wide berms and a 78° face angle for all material types. This angle was applied on both the hanging wall and footwall for the optimisation. Ramps were not separately accounted for in the NPVS open-pit economics determination and optimisation.

16.5.2.2 Dilution and Mining Recovery

Based on MPlan's experience mining modifying factors were applied in NPVS for graphite dilution and losses. An overall dilution factor of 2.5% in volume was assumed. The graphite grade of the diluting material surrounding the mineralised domains was assumed to be zero.

The overall mining recovery was estimated at approximately 97.5% based on a 2.5% mining loss assumption. MPlan is assuming the high mining recovery will only be achievable if selective mining practices, along with sampling and grade control are followed.

16.5.2.3 Concentrator overall throughput schedule

An annual concentrator planned production capacity of 160,000 tpa was supplied by Zenito and applied in NPVS.

16.5.2.4 Anticipated Costs

Anticipated cost of production assumptions was supplied by Woxna Graphite, Zenito and MPlan.

The anticipated costs of production for this study were developed considering; direct mining costs, processing costs, selling costs, and overhead or time dependent costs.

16.5.2.4.1 Direct mining costs

Anticipated direct mining costs relate to the cost per unit of planned production such as load and haul, and blasting. A combined anticipated mining cost per tonne was provided by MPlan and listed in Table 16-26 for each material type mined.

Table 16-26: Direct mining costs assumptions.

Material type	SEK per tonne	USD per tonne
Waste	40	4.51
Type B RoM	40	4.51
Type A RoM	40	4.51
Vertical Mining Cost Component	N/A	N/A
Overburden	40	4.51

16.5.2.4.2 Processing costs and recovery

The following assumed values for graphite recovery and operating cost at the processing plant were provided by Zenito:

- Woxna Concentrator and VAP graphite recovery 93.8%
- Plant and infrastructure operating expenses (inclusive of general and administration, and selling costs) USD 84.18 /RoM tonne.

16.5.2.4.3 Indirect or time related costs

Any extra costs which result from extending the LoM must be accounted for. Time costs include site overhead costs such as inductions, training, supervision, workshop, offices, technical services, light vehicles, and the like.

Woxna Graphite did not supply annual indirect cost estimates therefore no cost was included in NPVS

16.5.2.4.4 Selling costs

Selling costs are those costs that are incurred when the product is sold.

16.5.2.4.5 Mineral compensation

Mineral compensation costs are those costs that are incurred when the mineral is mined – in this case it is assumed applicable payment of mineral compensation as summarised below:

For each calendar year exploitation is undertaken, the concessionaire shall pay mineral compensation. This compensation shall be equal to two-thousandths (0.2%) of the calculated value of the minerals covered by the concession and are extracted and brought to the surface within the concession area during the year. The calculation shall be based on the quantity of ore brought to the surface, its concession mineral content and the average price of the mineral during the year or a corresponding value. [9]

The yearly mineral compensation shall be determined by the Chief Mining Inspector.

An annual 0.2% mineral compensation for use in the optimisation, this was converted to a selling cost per unit in NPVS.

16.5.2.5 Revenue

An average graphite sale price of 2,207 USD/t was provided by Zenito based on graphite recoveries and current prices for a blend of products to be generated from both graphite products. Refer to Section 19.5.

Note that the Kringel mine depth is limited to 70-metres (applied due permitting), and by the mining concession limits at its geographic boundary. The proposed mine plan fully exploits the available resource within these limits and is therefore relatively insensitive to other factors that might otherwise expand the pit, such as an increase in product pricing, meaning the pit shells generated using this lower product pricing remain applicable.

16.5.2.6 Discounting

The discount rate applied was 10% annually in this optimisation.

Note that the discount rate for the full PEA economic analysis conducted later was set to 8%.

16.5.3 Optimisation results

NPVS optimisation software generates a series of nested open pit shells by varying the RF parameter based on a set of financial and other parameters such as costs and graphite products price.

The pertinent NPVS open pit shells generated by pit number are given in Table 16-27. The optimisation results are presented graphically in Figure 16-22 with pit waste and graphite+gangue tonnage as well as discounted net value versus pit number.

Table 16-27: Kringel NPVS pit by pit summary of results

Phase/Code	Price Factor	NPV $	Graphite tonnes	Waste tonnes	Strip Ratio
Pit 1 (16)	32%	182,172	896	792	0.88
Pit 2 (17)	34%	423,261	2,162	2,057	0.95
Pit 3 (18)	36%	2,488,852	14,607	8,983	0.61
Pit 4 (19)	38%	19,303,398	128,778	171,121	1.33
Pit 5 (20)	40%	43,260,041	313,638	603,513	1.92
Pit 6 (21)	42%	51,151,922	384,249	770,569	2.01
Pit 7 (22)	44%	66,094,575	532,274	1,247,442	2.34
Pit 8 (23)	46%	76,300,906	653,274	1,563,892	2.39
Pit 9 (24)	48%	82,847,064	740,417	1,867,501	2.52
Pit 10 (25)	50%	94,295,757	922,642	2,421,198	2.62
Pit 11 (26)	52%	106,833,410	1,170,202	3,333,312	2.85
Pit 12 (27)	54%	115,523,558	1,370,488	4,288,615	3.13
Pit 13 (28)	56%	125,950,887	1,700,894	5,802,726	3.41
Pit 14 (29)	58%	128,143,194	1,783,951	6,128,340	3.44
Pit 15 (30)	60%	132,546,303	1,980,913	7,085,482	3.58
Pit 16 (31)	62%	134,005,266	2,060,886	7,348,026	3.57
Pit 17 (32)	64%	136,982,634	2,232,324	8,249,502	3.70
Pit 18 (33)	66%	138,849,826	2,363,290	8,973,542	3.80
Pit 19 (34)	68%	140,650,384	2,511,659	9,764,512	3.89
Pit 20 (35)	70%	142,353,686	2,683,072	10,629,355	3.96
Pit 21 (36)	72%	143,146,906	2,778,153	11,108,004	4.00
Pit 22 (37)	74%	143,876,965	2,889,026	11,492,260	3.98
Pit 23 (38)	76%	144,415,008	2,984,765	11,878,528	3.98
Pit 24 (39)	78%	144,709,493	3,048,100	12,103,841	3.97
Pit 25 (40)	80%	145,062,118	3,130,022	12,516,220	4.00
Pit 26 (41)	82%	145,265,386	3,183,601	12,829,618	4.03
Pit 27 (42)	84%	145,333,677	3,206,382	12,914,535	4.03
Pit 28 (43)	86%	145,472,818	3,257,508	13,215,565	4.06
Pit 29 (44)	88%	145,524,660	3,285,801	13,303,510	4.05
Pit 30 (45)	90%	145,567,244	3,311,297	13,421,875	4.05
Pit 31 (46)	92%	145,646,490	3,375,448	13,724,771	4.07
Pit 32 (47)	94%	145,677,819	3,413,840	13,784,483	4.04
Pit 33 (48)	96%	145,690,146	3,430,372	13,889,166	4.05
Pit 34 (49)	98%	145,702,665	3,471,214	13,941,527	4.02
Pit 35 (50)	100%	145,706,557	3,496,079	14,066,536	4.02

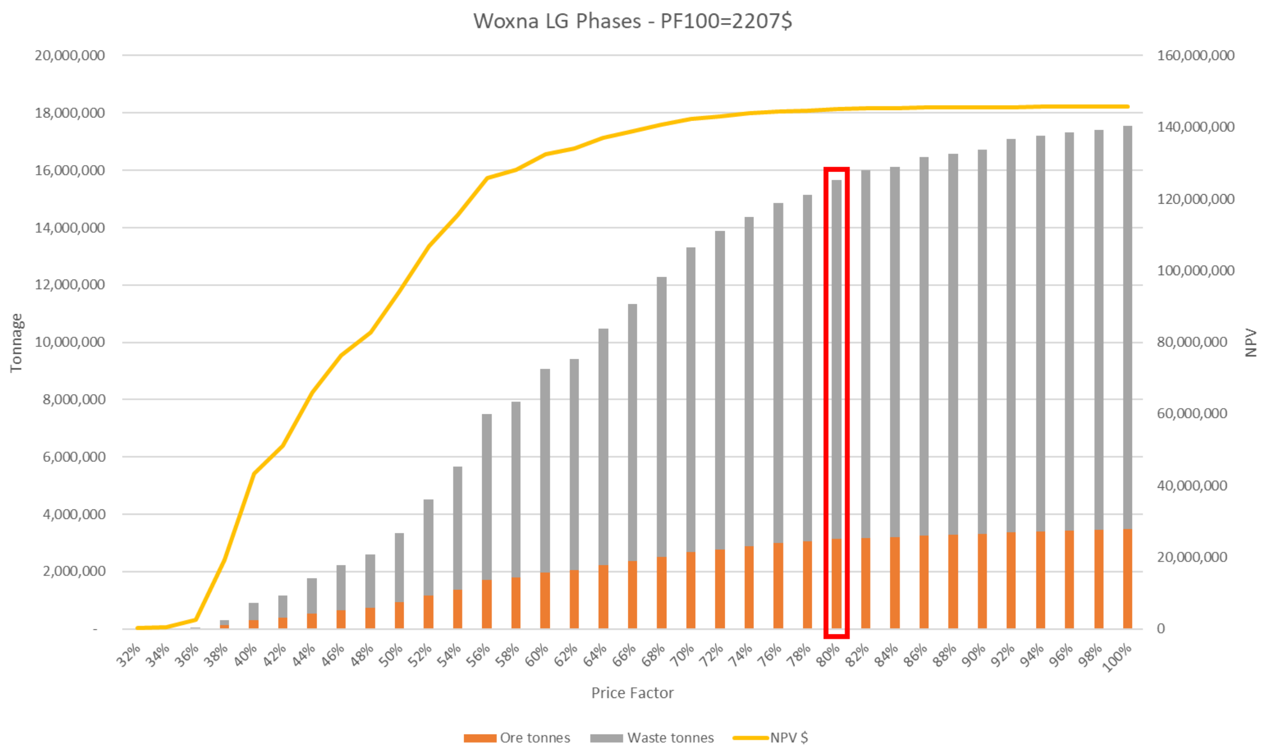


Figure 16-22: NPVS pit by pit summary of results

16.5.3.1 Pit Selection

The selected pit shell was the LG pit shell with a price factor (PF) of 80% (Figure 16-23).

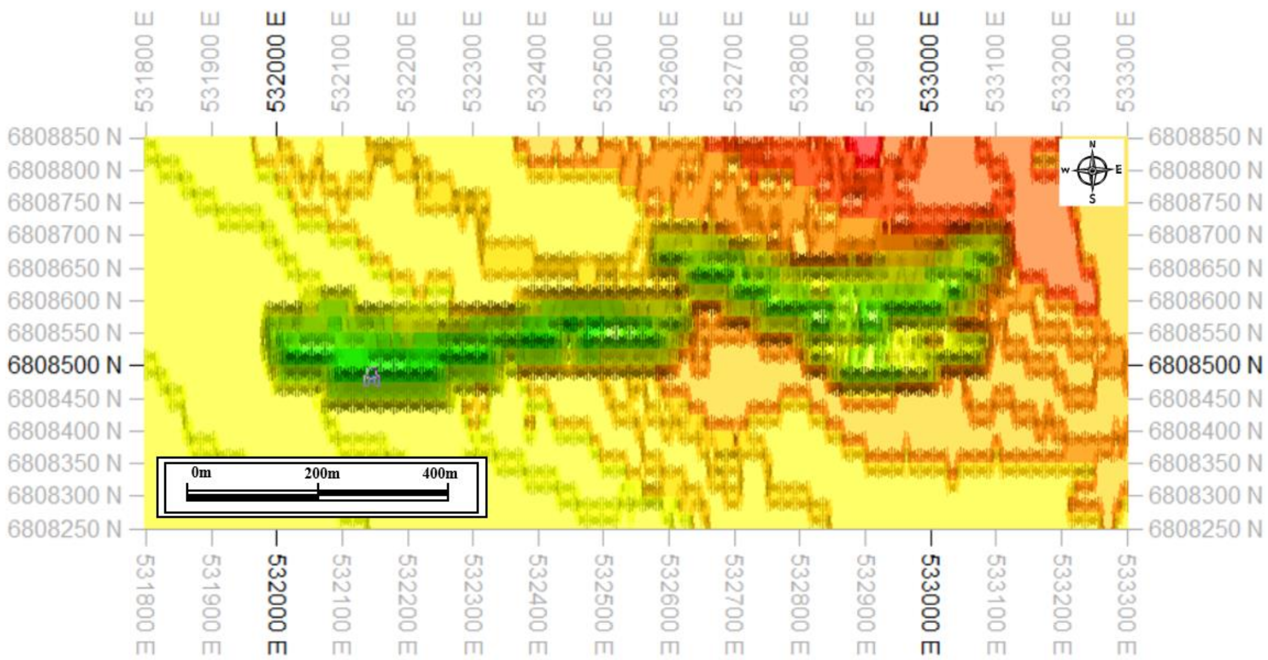


Figure 16-23: Selected pit shell (PF 80%) for use as template for pit design

16.5.3.2 Pushbacks

The tonnes and grades from the four pushbacks are presented in Table 16-28.

Table 16-28: Woxna Mine Pushback Phases

Push-back	Waste, Rom	Revenue	Processing Cost	Mining Cost	Mill Feed	Mill Feed	Waste	Strip Ratio	High Grade Type A	High Grade Type A	Low Grade Type B	Low Grade Type B
Phase	Mt	M USD	M USD	M USD	Mt	% C	Mt	W:O	Mt	% C	Mt	% C
1	7.34	317.82	153.32	33.11	1.82	8.70%	5.52	3.03	1.36	9.9%	0.460	5.19%
2	1.19	41.20	14.71	5.38	0.17	11.75%	1.02	5.82	0.17	11.8%	0.000	0.00%
3	5.85	199.60	78.47	26.38	0.93	10.67%	4.92	5.27	0.87	11.1%	0.064	4.88%
4	0.06	8.10	3.27	0.29	0.04	10.39%	0.03	0.65	0.04	10.5%	0.001	5.07%

The pushback phases (or pushbacks) are illustrated in Figure 16-24 to Figure 16-27. Note that pushback Phase 1 is the east pit while pushbacks Phase 2 and 3 are exclusively in the west pit. Pushback Phase 4 is the pit bottoms of the east and west pit remaining after pushback three.

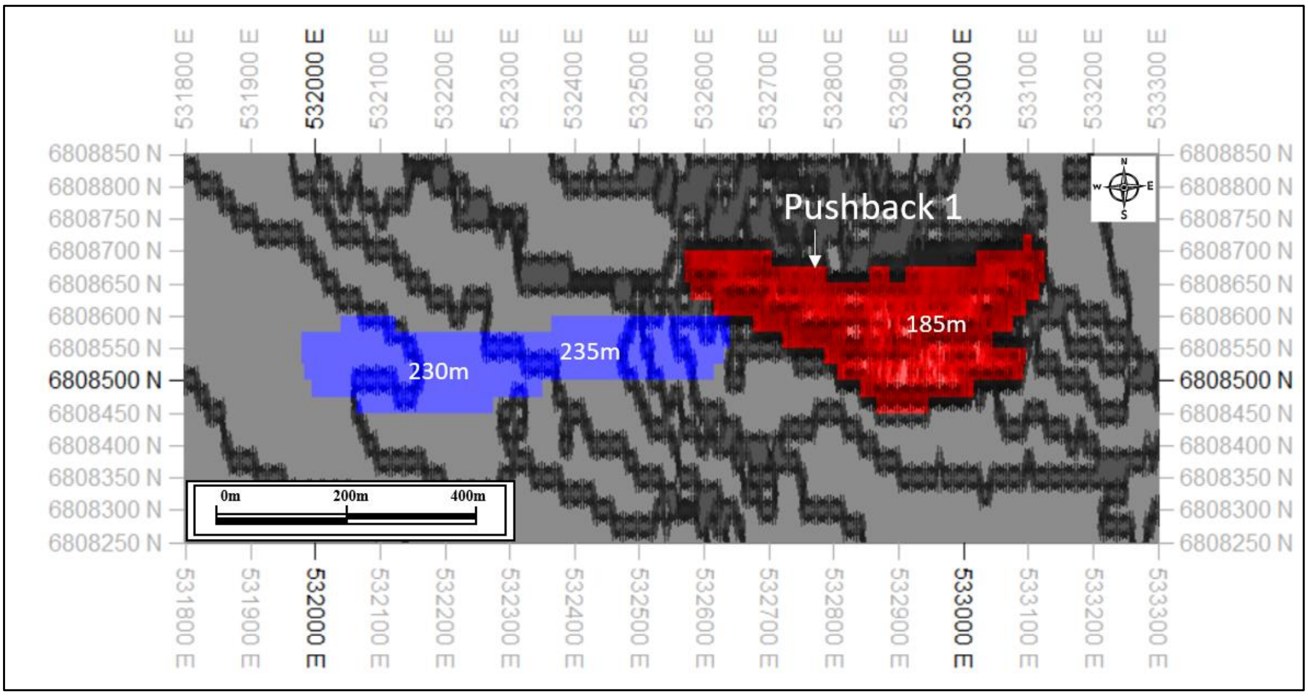


Figure 16-24: Pushback 1

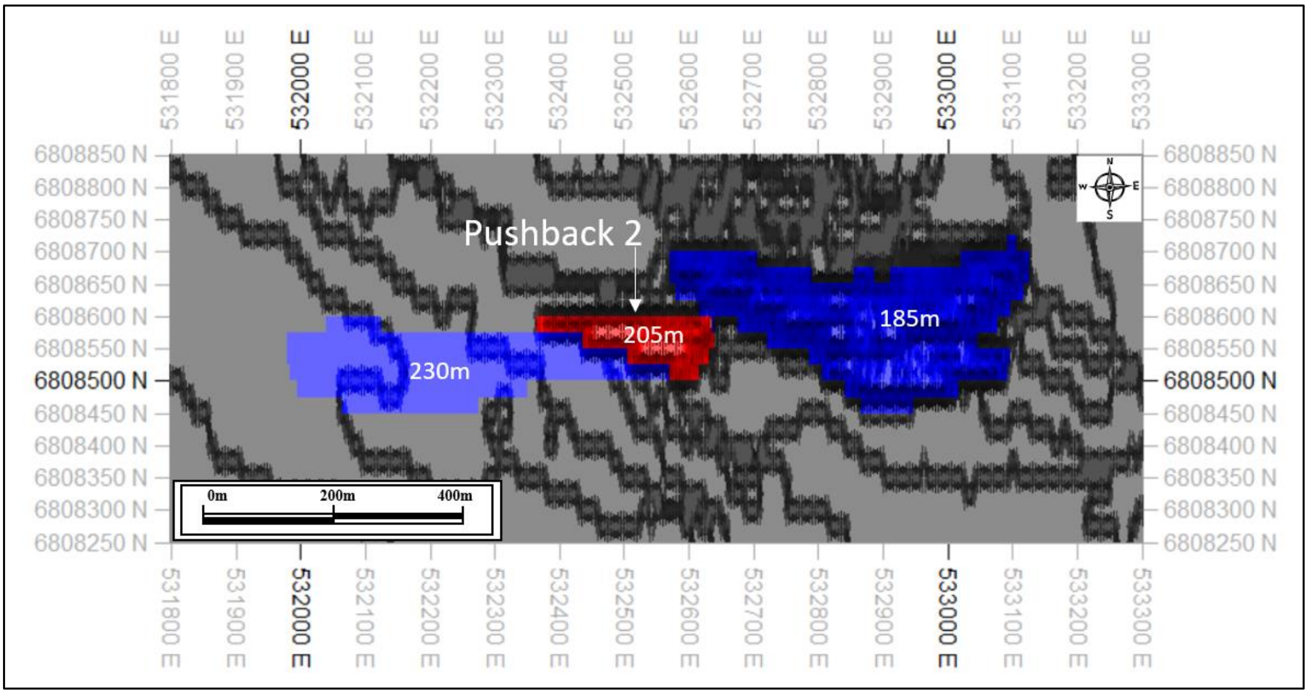


Figure 16-25: Pushback 2

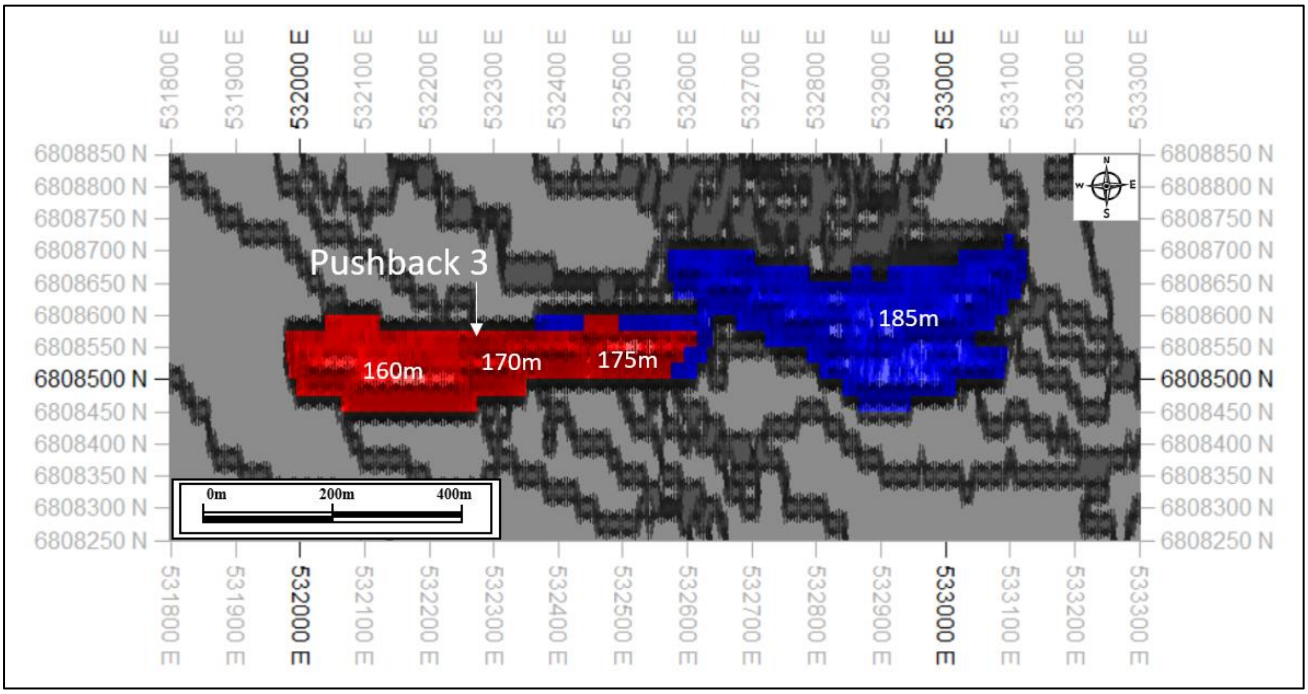


Figure 16-26: Pushback 3

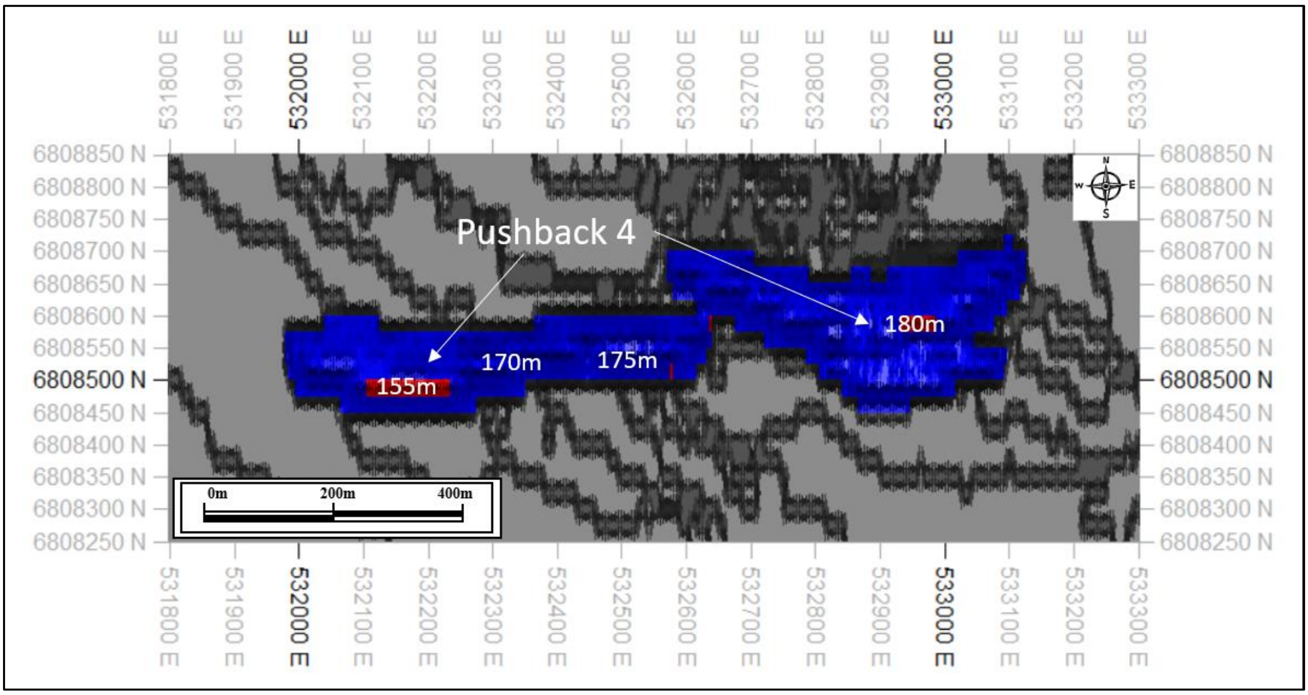


Figure 16-27: Pushback 4

16.6 Mining plan

16.6.1 Mine production scheduling

The last step undertaken in order to generate LoP and LoM production schedules, is through the mine scheduling of production plans per year in Datamine™.

The objective is to present a mining production plan in which capital and operating costs are deferred to reduce investment risk as much as possible. This objective is achieved by sequencing the lowest stripping ratio and highest-grade areas first, and higher stripping and low-grade areas last. Therefore the mine production schedule was matched to the NPVS pit optimisation results.

The mine schedule targeted approximately 160,000 tpa of Type A RoM at an average in situ grade of 10% C RoM over the first 15-years of production. During that time, Type B RoM will be stockpiled for processing during the final years of the mine schedule. In total the LoM production schedule will generate approximately 258,000 tC in concentrate

A pre-production year, i.e. Year 0, has been included to allow sufficient time to excavate overburden and allowing bedrock exposure and preparation for drill and blast ahead of first full mine production Year 1 with a 160 kt of RoM delivered to Plant. The last three years, i.e. Year 17-19 comprise stockpiles of low-grade material being fed to the plant and no mining will plan to take place in the open pit. Stockpile recovery optimisation has not been evaluated as part of this study. The estimated stockpile grade at the end of Year 16 was used as the average percentage carbon grade being fed to the plant for the next three years of reclamation.

The plan mining schedule presented in Section 16.6 provide the estimated annual production requirement in BCM.

Production requirements in BCM per annum are summarised in Table 16-29 and were derived from the mine production schedule. The average years 0 to 9 for overburden and years 1 to 16 for waste and RoM provide an indication of the of the BCM pa necessary for the LoP.

Table 16-29: Production rates in BCM per annum

	Min BCM/a	Max BCM/a	Annual avg. BCM/a
Overburden	8,000	195,000	57,500 (Year 0-9)
Waste	10,000	420,000	233,000 (Year 1- 16)
RoM	16,500	89,000	69,000 (Year 1-16)

The mine production schedule includes circa 10% inferred mineral resources as part of the RoM over the LoP.

Table 16-30: Mine production schedule for Woxna Mine before stockpiling & process schedule

Woxna Mine Production	Units	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Total
Total Rock Mined	Mt	0.45	1.39	1.068	0.862	1.004	1.017	1.005	1.014	1.010	0.984	1.056	0.943	0.874	0.658	0.513	0.430	0.071	0.000	0.000	0.000	14.449
Run of Mine -HG Mined	Mt	n/a	0.161	0.160	0.159	0.161	0.160	0.159	0.160	0.160	0.160	0.160	0.160	0.160	0.160	0.161	0.160	0.043	0.000	0.000	0.000	2.444
Run of Mine -HG Mined	grade	n/a	10.2%	10.2%	9.9%	9.7%	9.6%	9.6%	9.9%	10.4%	10.6%	9.2%	10.1%	11.0%	10.7%	11.1%	10.8%	9.9%	0.0%	0.0%	0.0%	10.2%
Run of Mine - LG Mined	Mt	n/a	0.059	0.031	0.036	0.079	0.073	0.047	0.028	0.037	0.027	0.031	0.036	0.015	0.003	0.012	0.011	0.001	0.000	0.000	0.000	0.525
Run of Mine - LG Mined	grade	n/a	6.0%	5.3%	4.8%	4.6%	4.6%	5.0%	5.2%	5.1%	5.1%	5.1%	5.2%	4.9%	4.2%	4.6%	4.8%	4.9%	0.0%	0.0%	0.0%	5.0%
Overall Mined C_pct	%	n/a	9.1%	9.4%	9.0%	8.0%	8.1%	8.5%	9.2%	9.4%	9.8%	8.6%	9.2%	10.5%	10.6%	10.7%	10.4%	9.8%	0.0%	0.0%	0.0%	9.3%
Total Waste	Mt	0.45	1.720	0.876	0.668	0.764	0.784	0.799	0.825	0.813	0.797	0.866	0.747	0.699	0.495	0.341	0.259	0.027	0.000	0.000	0.000	11.480
Strip Ratio	t:t	n/a	5.4	4.6	3.4	3.2	3.4	3.9	4.4	4.1	4.3	4.5	3.8	4.0	3.0	2.0	1.5	0.6	0.0	0.0	0.0	3.7
	Bcm:t	n/a	2.0	1.7	1.3	1.2	1.3	1.5	1.7	1.6	1.6	1.7	1.4	1.5	1.1	0.7	0.6	0.2	0.0	0.0	0.0	1.4
Stockpiles	**Units**		**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**	**Year 16**	**Year 17**	**Year 18**	**Year 19**	**Total**
Balance in Stockpile	Mt	n/a	0.059	0.090	0.125	0.204	0.277	0.324	0.352	0.389	0.416	0.447	0.483	0.498	0.501	0.513	0.524	0.525	0.408	0.248	0.088	n/a
Graphite Content	Mt	n/a	0.004	0.005	0.007	0.011	0.014	0.017	0.018	0.020	0.021	0.023	0.025	0.026	0.026	0.026	0.027	0.027	0.021	0.013	0.005	n/a
Stockpile grade at end of year	%	n/a	6.0%	5.8%	5.5%	5.2%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.1%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	n/a
tonnes out during year	Mt	n/a	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.117	0.160	0.160	0.088	0.525
grade of tonnes out	%	n/a	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	5.15%	5.15%	5.15%	5.15%	5.15%
Processed Material	**Units**		**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**	**Year 16**	**Year 17**	**Year 18**	**Year 19**	**Total**
RoM Processed Direct from Mine	Mt	n/a	0.161	0.160	0.159	0.161	0.160	0.159	0.160	0.160	0.160	0.160	0.160	0.160	0.160	0.161	0.160	0.043	0.000	0.000	0.000	2.444
RoM Processed Grade	%	n/a	10.2%	10.2%	9.9%	9.7%	9.6%	9.6%	9.9%	10.4%	10.6%	9.2%	10.1%	11.0%	10.7%	11.1%	10.8%	9.9%	0.0%	0.0%	0.0%	10.2%
From Stockpile	Mt	n/a	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.117	0.160	0.160	0.088	0.525
From Stockpile Grade	%	n/a	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	5.0%	5.0%	5.0%	5.0%	5.0%
Processing Total from Mine & Stockpile	Mt	n/a	0.161	0.160	0.159	0.161	0.160	0.159	0.160	0.160	0.160	0.160	0.160	0.160	0.160	0.161	0.160	0.160	0.160	0.160	0.088	2.969
Processing Total from Mine & Stockpile Grade	%	n/a	10.2%	10.2%	9.9%	9.7%	9.6%	9.6%	9.9%	10.4%	10.6%	9.2%	10.1%	11.0%	10.7%	11.1%	10.8%	6.4%	5.0%	5.0%	5.0%	9.3%
Beneficiated Graphite Rock (concentrate)	t	n/a	15,387	15,252	14,791	14,699	14,431	14,251	14,866	15,533	15,776	13,851	15,135	16,528	16,074	16,725	16,140	9,511	7,533	7,533	4,136	258,152
Recovered Carbon	t	n/a	0.161	0.160	0.159	0.161	0.160	0.159	0.160	0.160	0.160	0.160	0.160	0.160	0.160	0.161	0.160	0.043	0.000	0.000	0.000	2.444

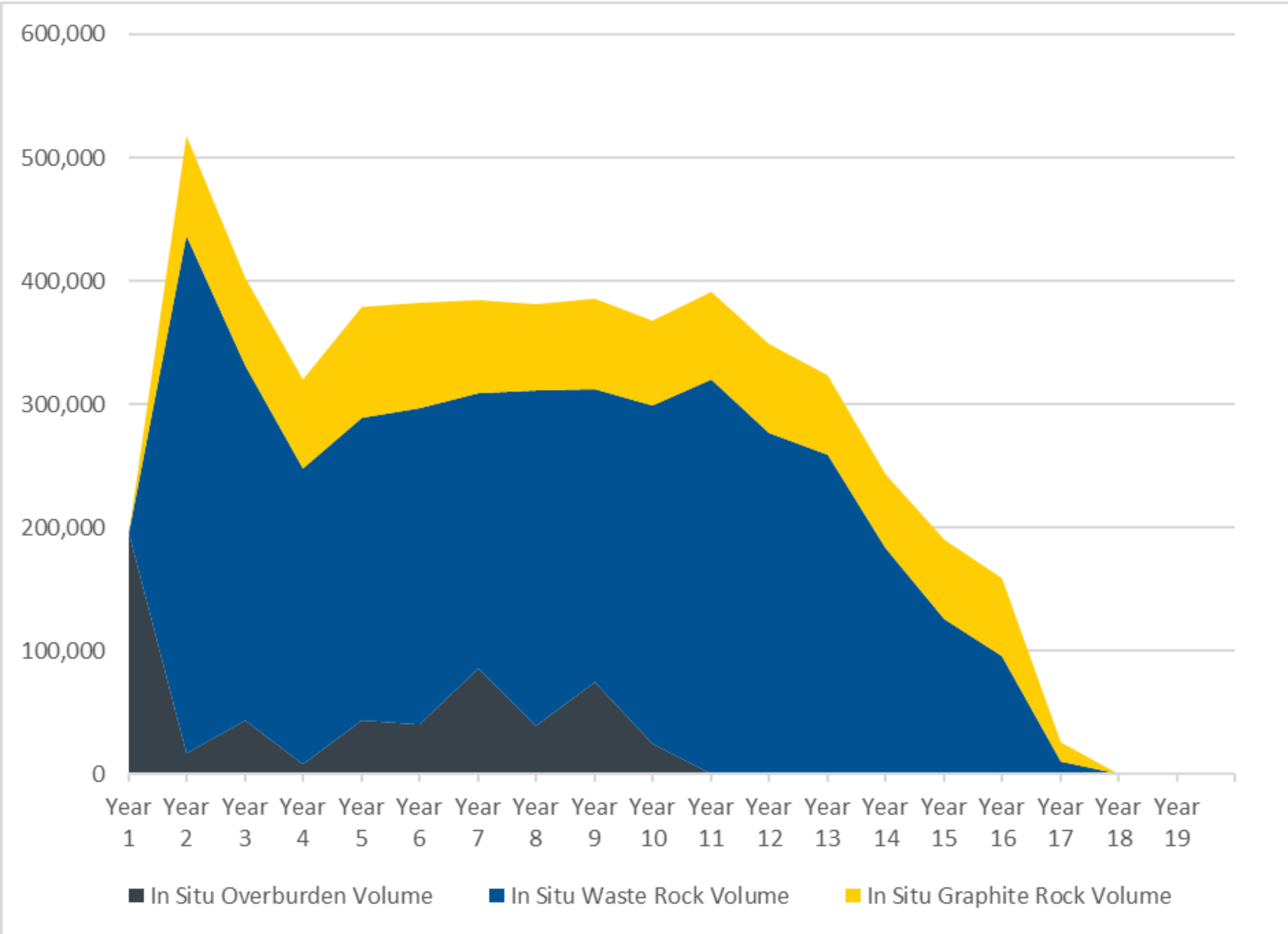


Figure 16-28: Annual Mine Transport Volumes Statistics in BCM

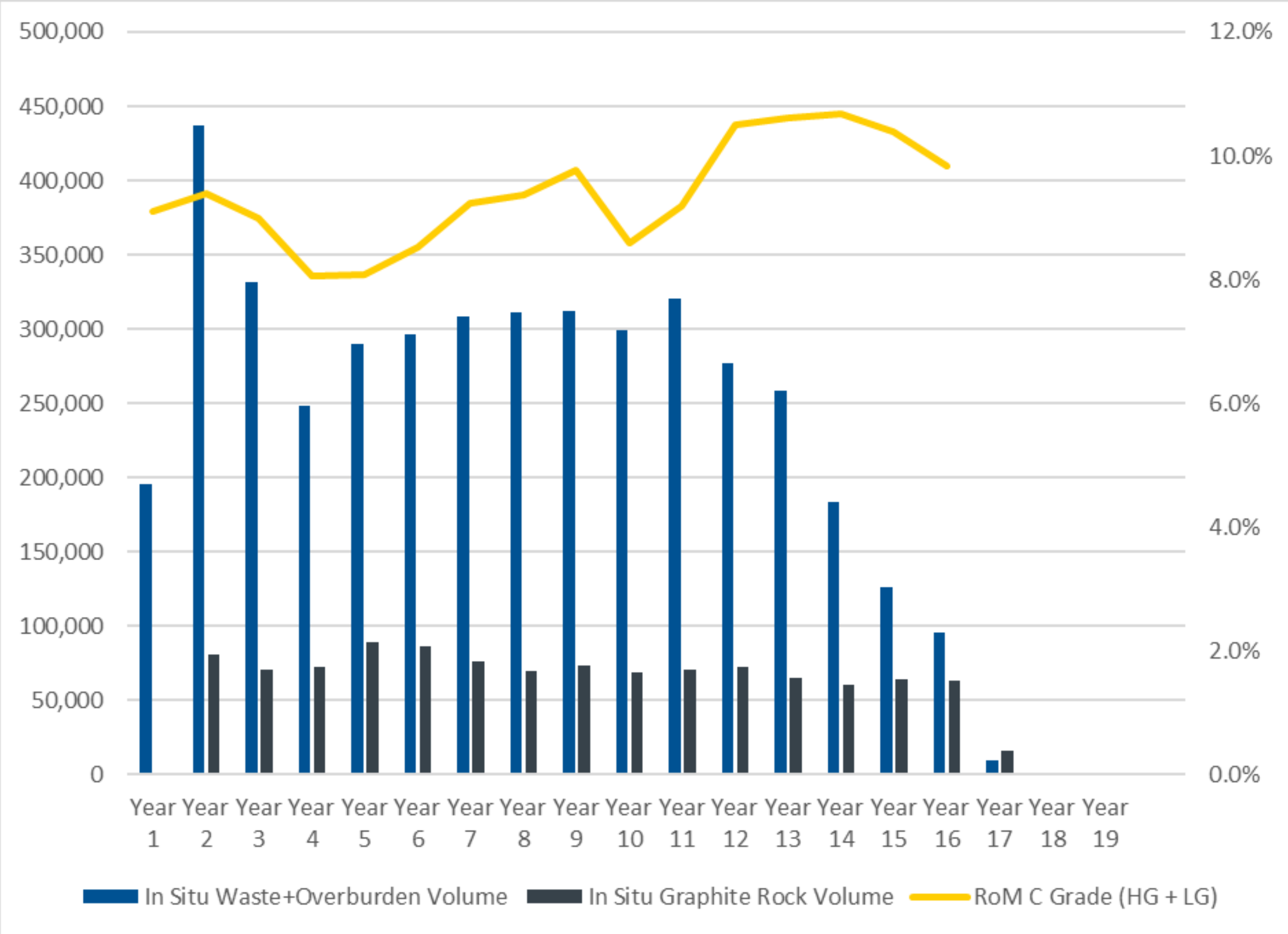


Figure 16-29: In Situ Mine Schedule Production Plan with ROM %C grade

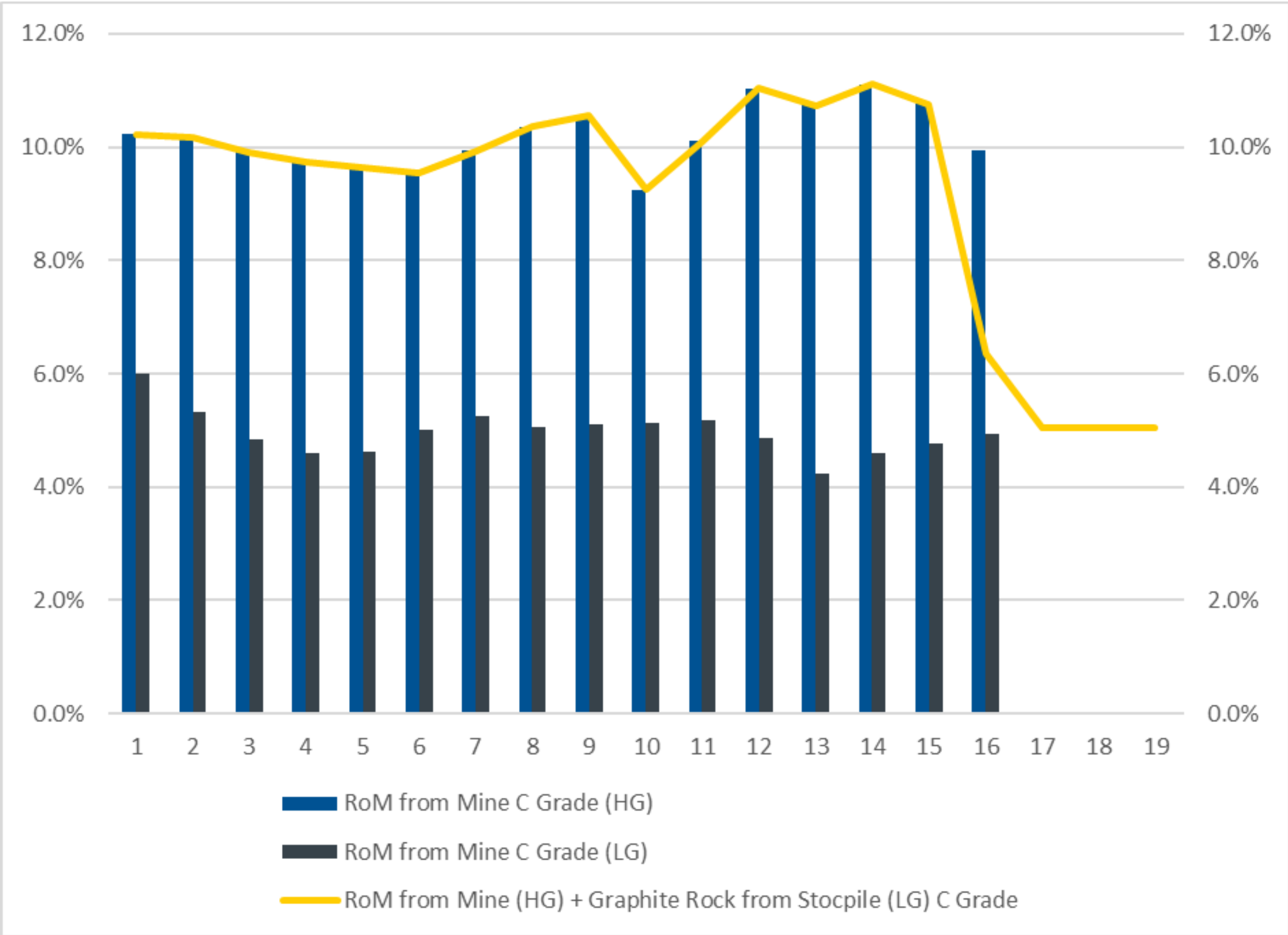


Figure 16-30: Annual Type A and B RoM versus Plant %C Grade Feed in Mine Schedule Production Plan

16.6.1.1 Pit development

The starter pit is located in the relatively high grade, totally or partially eroded overburden area with low strip ratio just west of the old open pit in order to maximise NPV and maintain low capital expenditures.

The mine plan begins with the mining of the east pit at a relatively high stripping ratio of 7.84 (waste tonnes for each tonne of mill feed) to give access to 160 kt of Type A high grade RoM in year one. The low grade Type B material is stockpiled for processing in the 16th year.

The mine plan only mines the east pit during the first three years of the mine plan. The east pit continues to be mined until year 13, whilst the west pit comes into production in year four and continues until year 16. The east pit is mined simultaneously in two sections, an eastern half and western half, which operationally will allow drilling to occur in one half while mining can occur in the opposite half.

Mining of the western pit begins at its eastern extreme and continues in the eastern extremity of the pit until it begins to extend towards the west in year seven. At some points, the western pit has three mini pits within it but by year 13 it is mined with a single contiguous pit bottom which reaches its final depth during year 16.

The supply to the concentrator continues until year 19 when the low grade mill feed from the stockpile is exhausted.

The progress of the pits as per the mine production schedule is illustrated in Figure 16-31 to Figure 16-40.

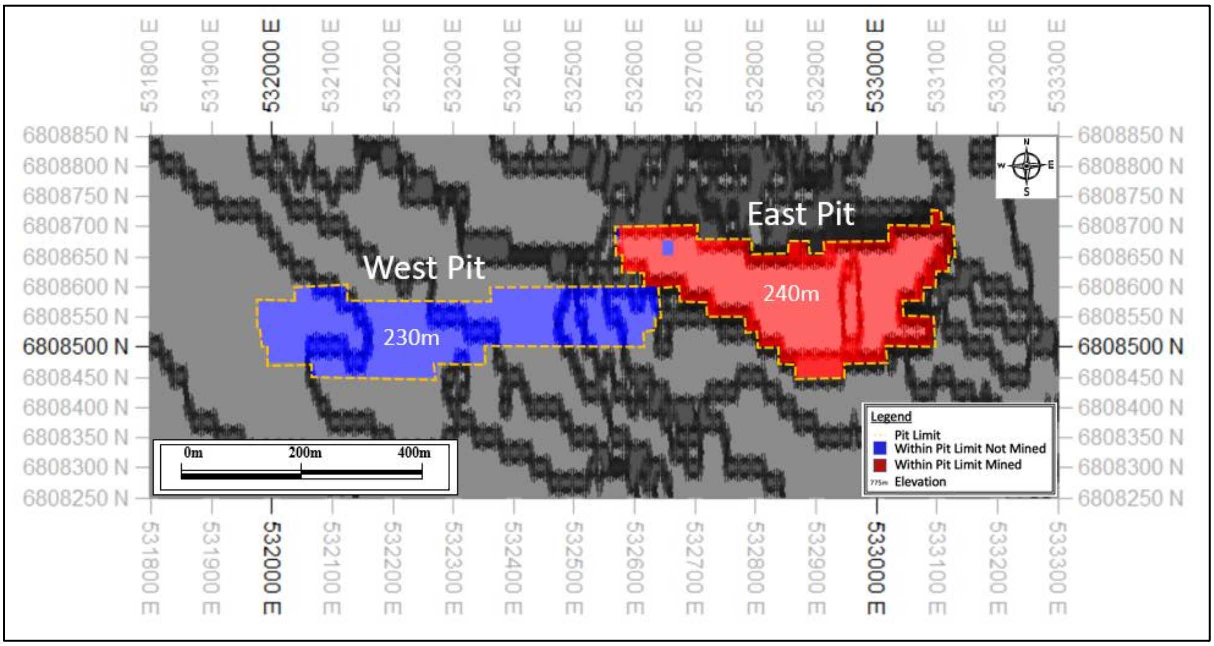


Figure 16-31: Progress of Mine Surface: End of Year 1

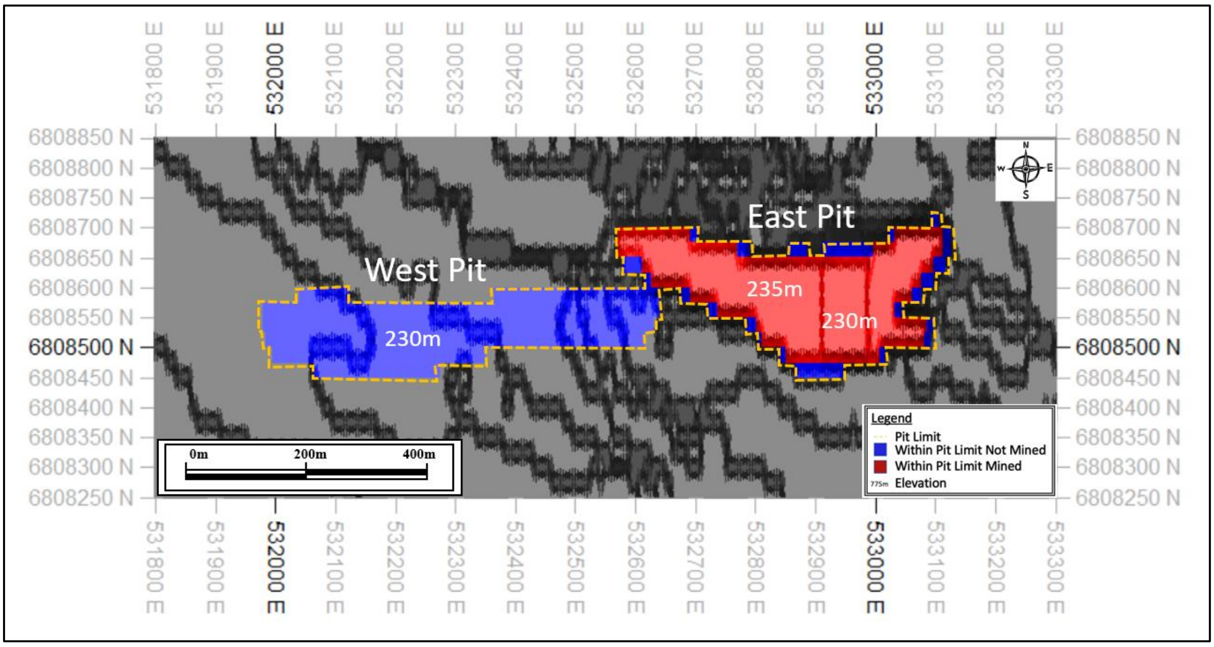


Figure 16-32: Progress of Mine Surface: End of Year 2

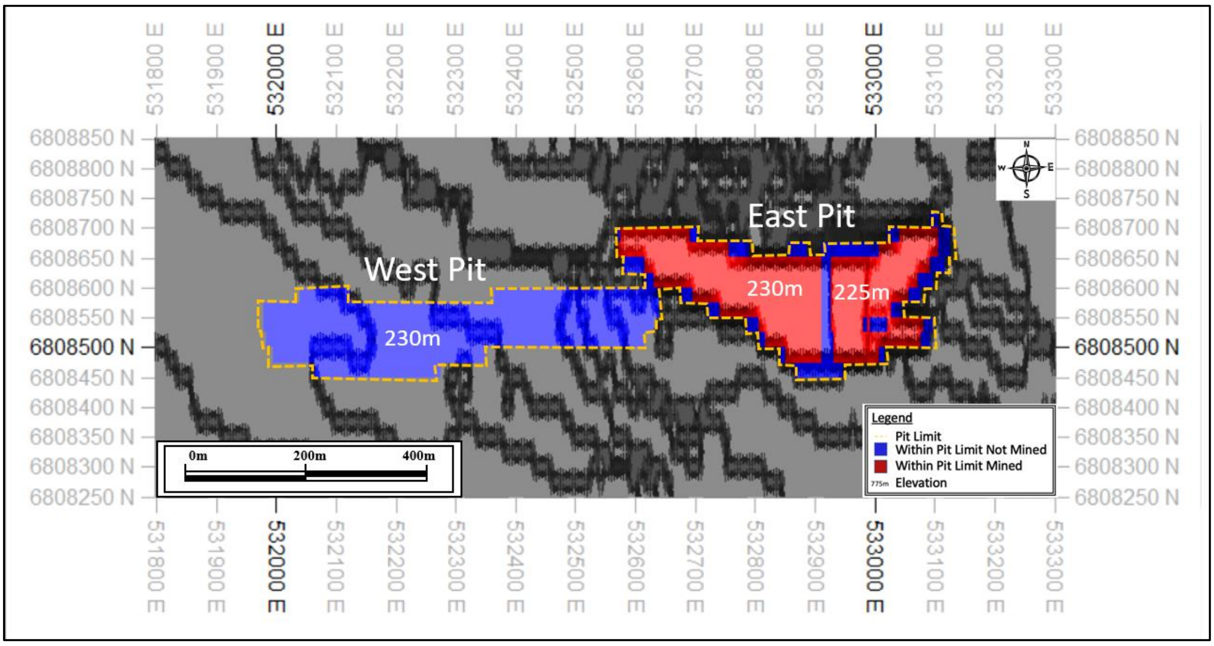


Figure 16-33: Progress of Mine Surface: End of Year 3

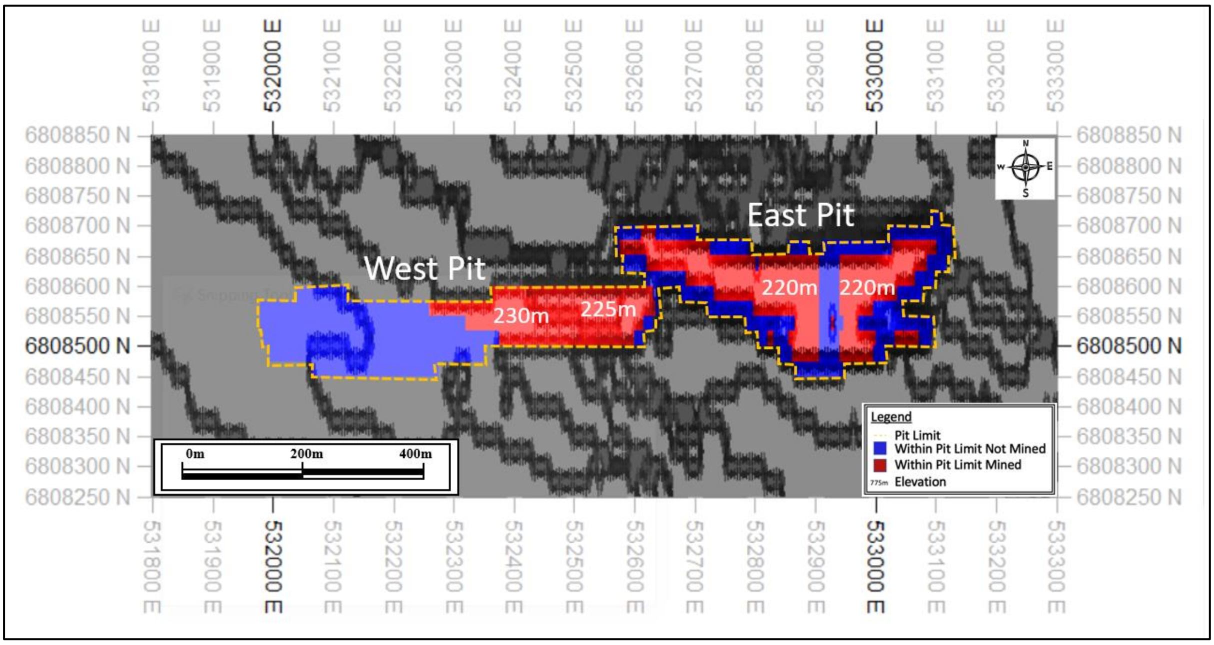


Figure 16-34: Progress of Mine Surface: End of Year 5

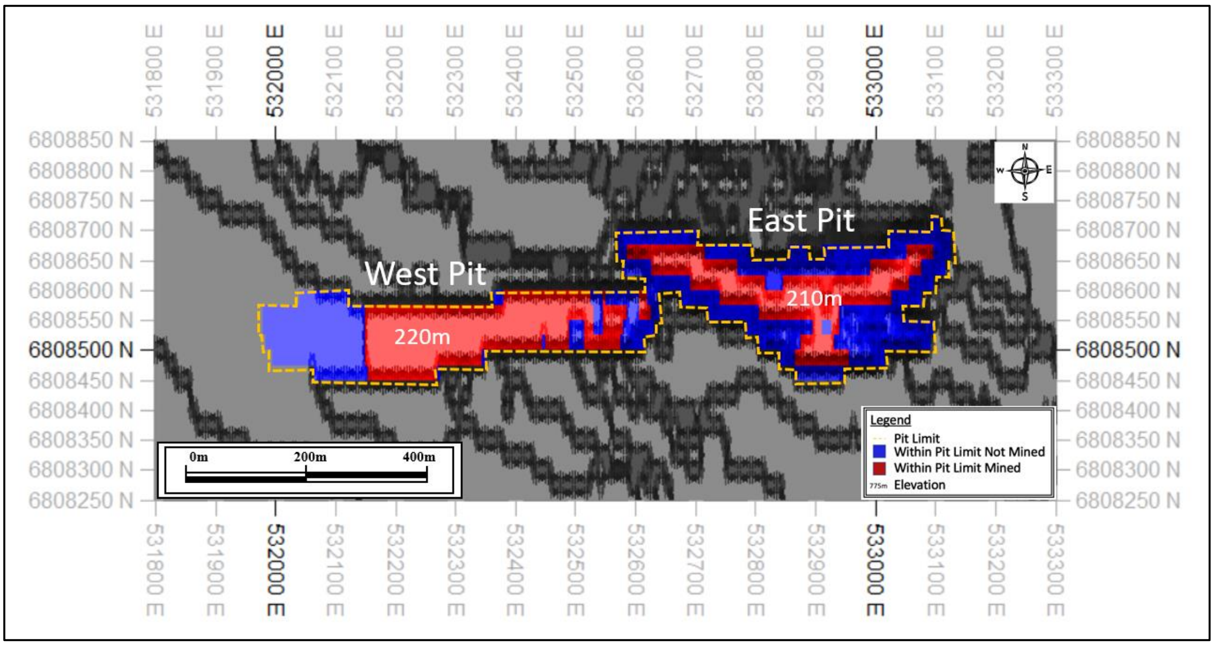


Figure 16-35: Progress of Mine Surface: End of Year 7

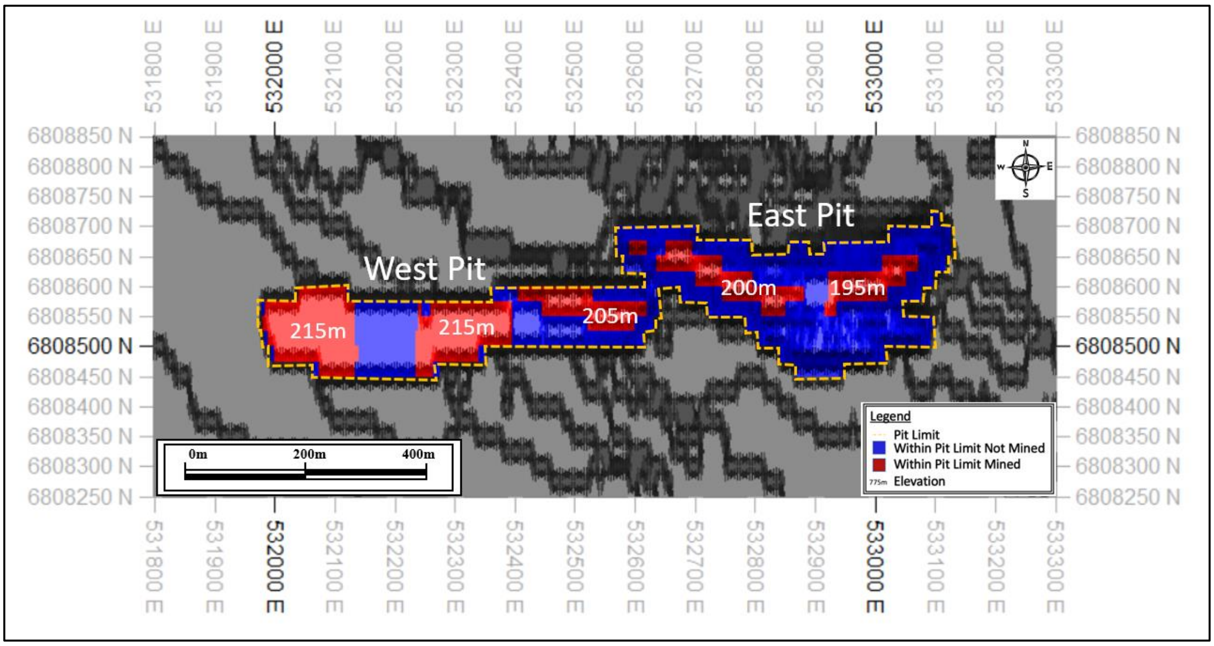


Figure 16-36: Progress of Mine Surface: End of Year 9

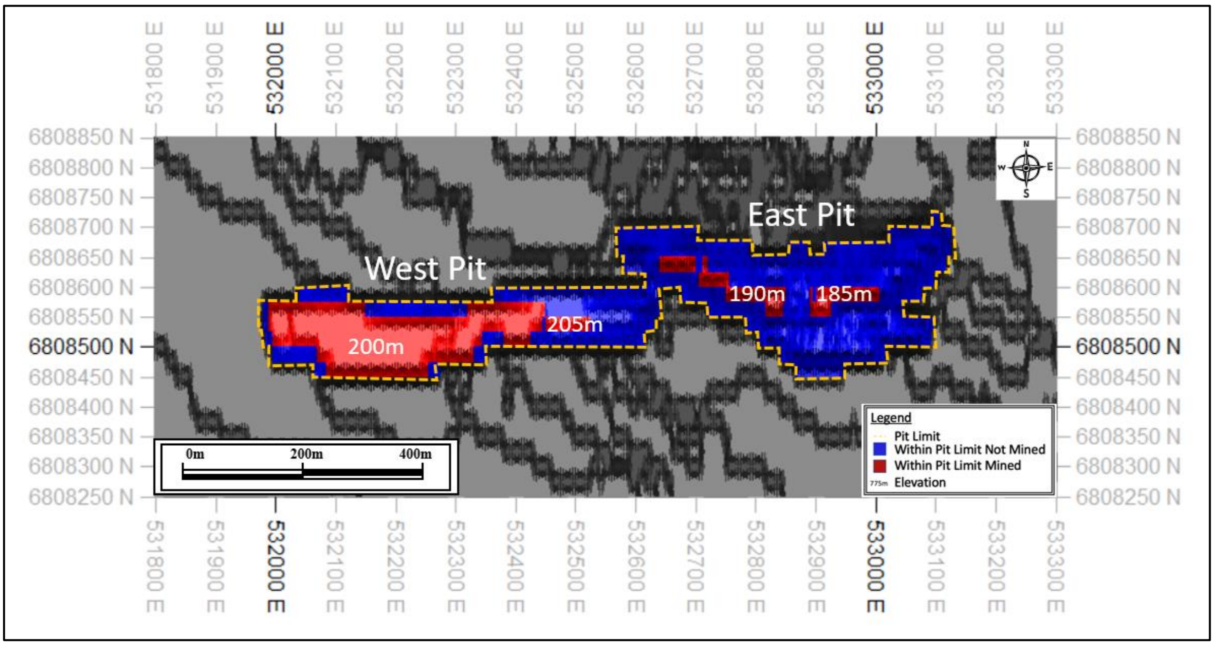


Figure 16-37: Progress of Mine Surface: End of Year 11

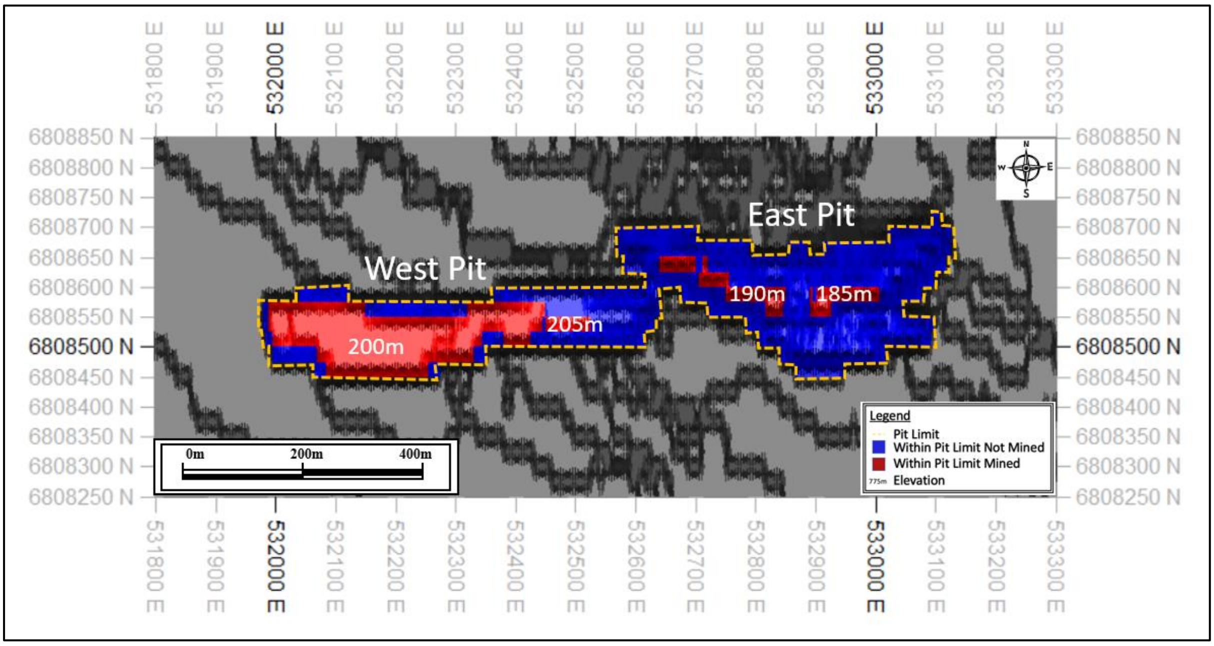


Figure 16-38: Progress of Mine Surface: End of Year 13

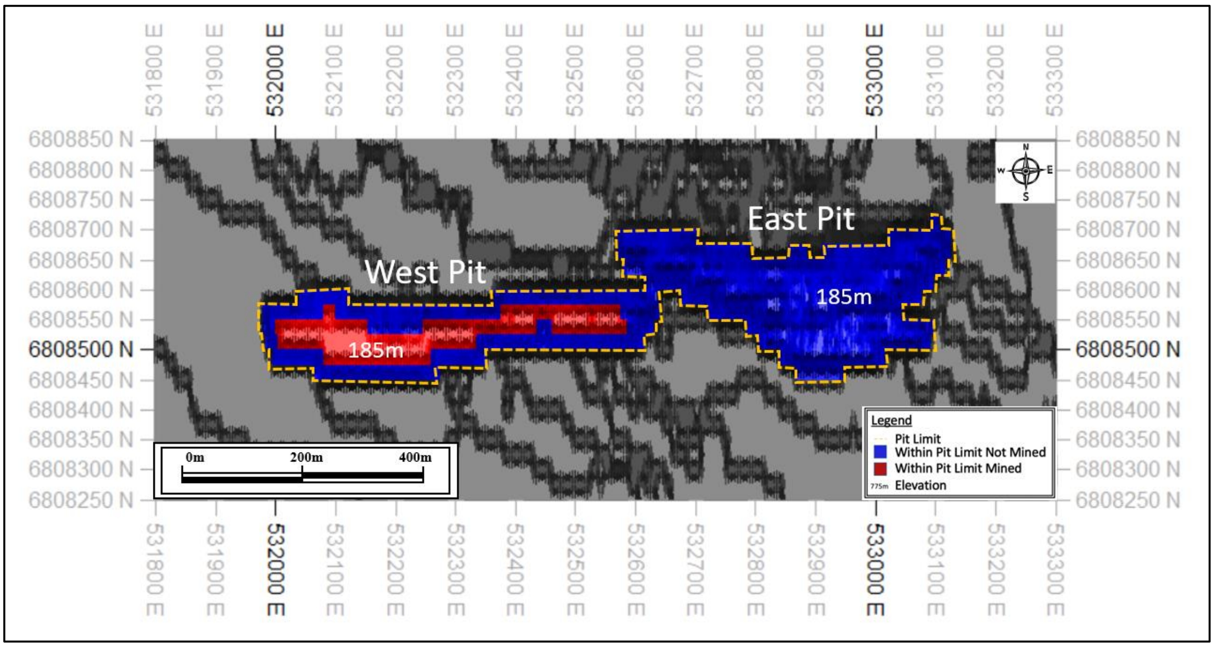


Figure 16-39: Progress of Mine Surface: End of Year 15

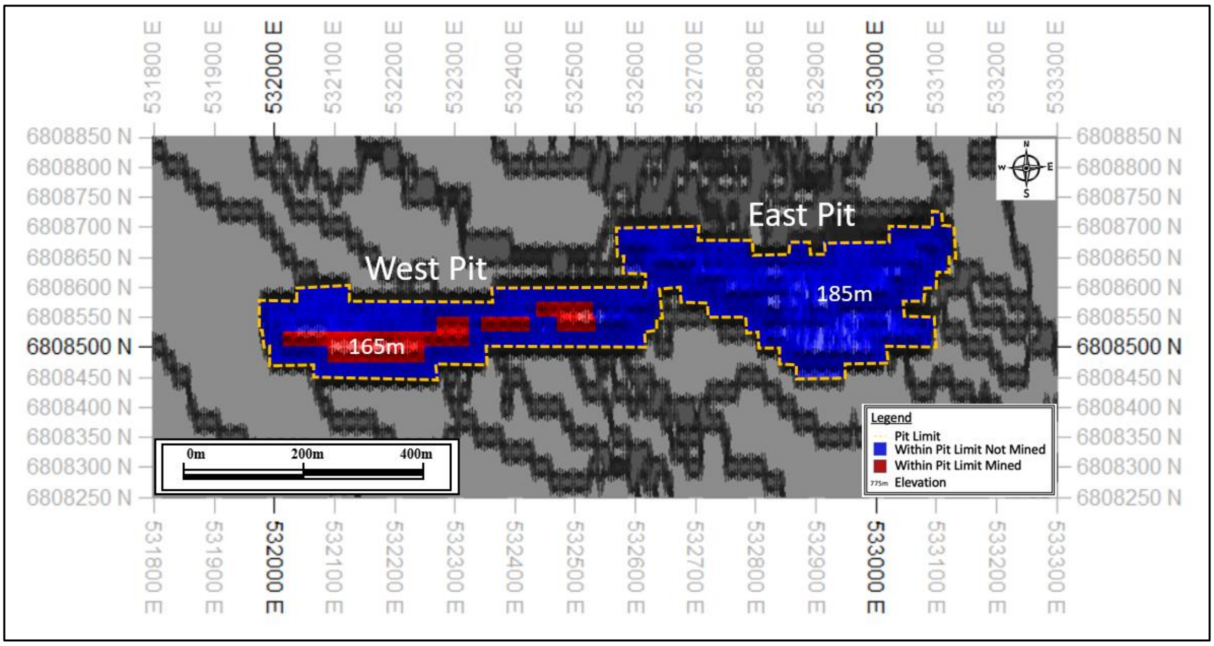


Figure 16-40: Progress of Mine Surface: End of Year 16

16.6.2 Pit slope configuration

During the first few years of mining operation, pit slope design will be one of the major challenges facing Woxna Mine in order to maintain economic performance by simultaneously maximising mineral recovery and keeping waste stripping to a minimum.

This section gives an overview of the terms typically used in the design of pit slope geometries as well as key geotechnical and mining pit slope design factors.

The standard pit slope terminology that is used throughout this report to describe typical geometric arrangement on pit wall of benches and haul road ramps is illustrated in Figure 16-41.

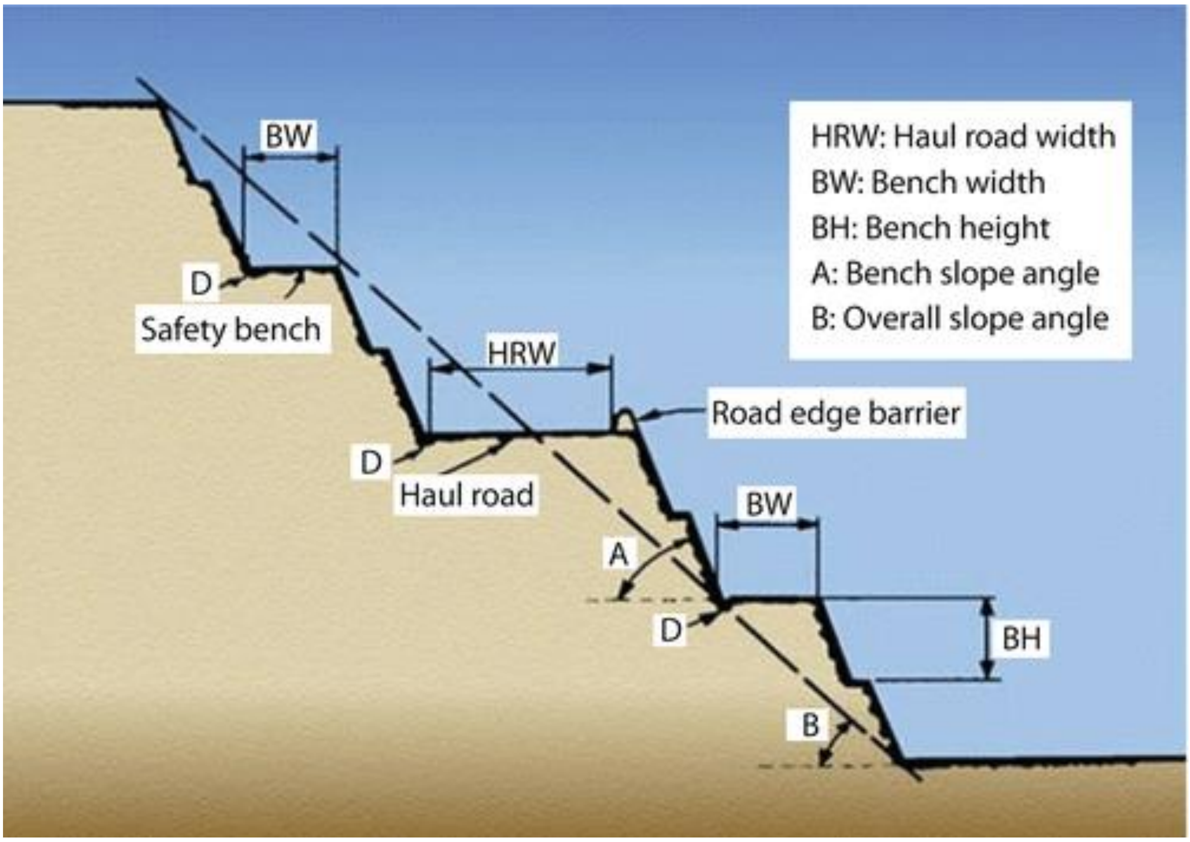


Figure 16-41: Inter-relationships between bench geometry, inter-ramp slope angle, and the overall slope angle. (Wyllie & Mah, 2004)

The configuration of the open-pit wall geometry is clearly influenced by slope angle, inter-ramp angle, and bench geometry (see Figure 16-41). Note that the bench face angles are defined between the toe and crest of each bench, whereas the inter-ramp slope angles between the haul roads/ramps are defined by the line of the bench toes. The overall slope angle is always measured from the toe of the slope to the topmost crest.

16.6.2.1 Bench Geometry

The height of the benches is typically determined by the size of the shovel chosen for the mining operation. The bench face angle is usually selected in such a way as to reduce, to an acceptable level, the amount of material that will likely fall from the face or crest. The bench width is sized to prevent small wedges and blocks from the bench faces falling down the slope and potentially impacting men and equipment. The bench geometry that results from the bench face angle and bench width will ultimately dictate the inter-ramp slope angle as illustrated in Figure 16-42. Double benches are proposed to be used at Woxna Mine to steepen inter-ramp slopes.

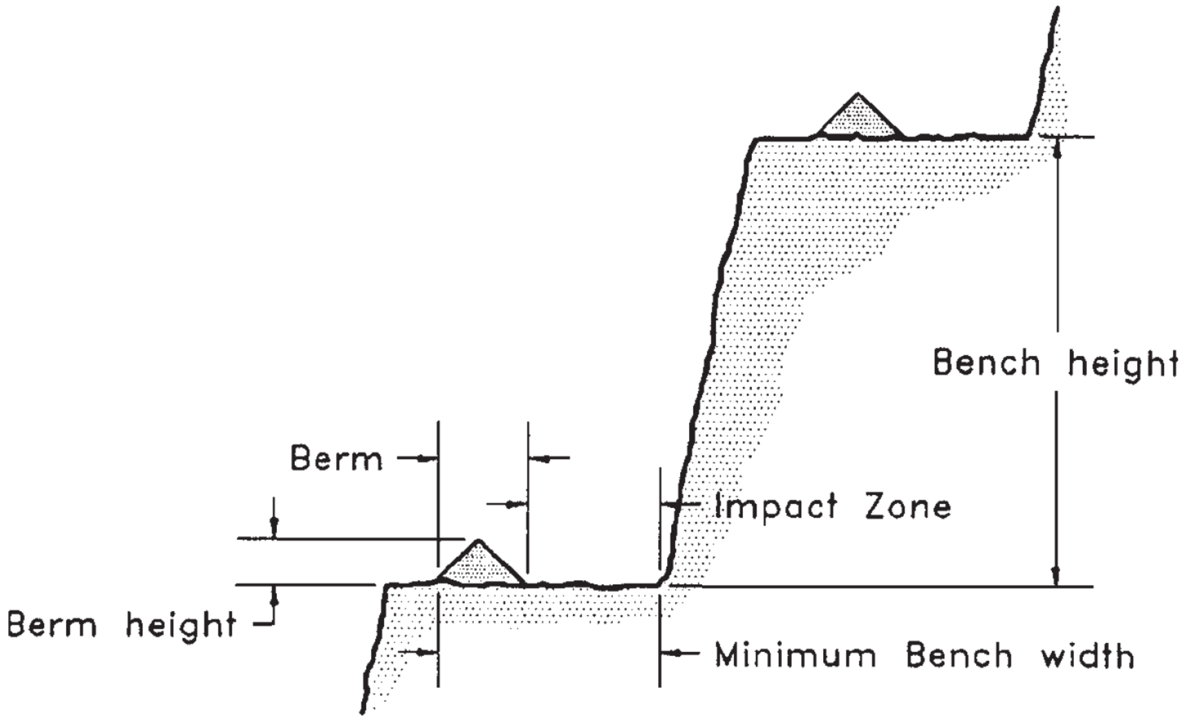


Figure 16-42: Catch bench geometry (originally from Call, 1986) (Kuchta & Hustrulid, 2006).

16.6.2.2 Inter-ramp Slope

The maximum inter-ramp slope angle is typically dictated by the bench geometry. However, it is also necessary to evaluate the potential for multiple bench scale instabilities due to large-scale structural features such as faults, shear zones, bedding planes, foliation etc. In some cases, these persistent features may completely control the achievable inter-ramp angles and the slope may have to be flattened to account for their presence.

16.6.2.3 Overall Slope

The overall slope angle that is achieved for Woxna Mine open-pit is flatter than the maximum inter-ramp angle due to the inclusion of haulage ramps with safety berms as illustrated in Figure 16-43.

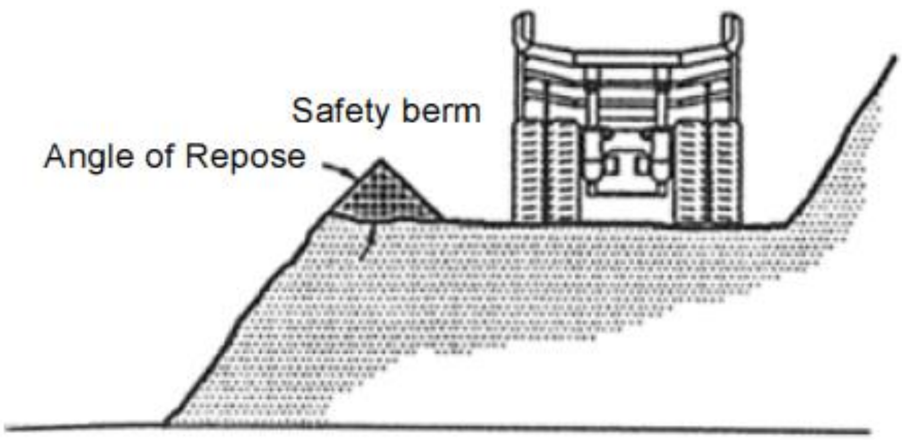


Figure 16-43: Haulage ramps and safety berms at bench edge (Manuel Bustillo Revuelta, Mineral Resources, Springer Textbooks 2017

Other factors that may reduce the overall slope angles are things such as rock mass strength, groundwater pressures, blasting vibration, stress conditions and mine equipment requirements.

16.6.2.4 Pit Slope Design Factors

No slope wall stability analysis was conducted on the Woxna Mine open-pit design which will a be key input into future PFS/DFS studies.

16.6.2.5 Hydrogeological Site Conditions

South and east, as well as partially overlapping the Kringel exploitation concession is located Östermyrorna, which is classified as national interest nature conservation area (kommun, 2012). The Östermyrorna area is ecologically sensitive and described as follows:

"*multiform wetland area with the value of wetland complexes, swamp forest, slightly domed bog, mad at waterways, soli genic and top geological marshes. Much of the area is limestone and has a rich flora.*" *(Bonde, 1998).*

High groundwater pressures and water pressure in tension cracks will reduce rock mass shear strength and may adversely impact slope stability. Depressurisation programmes can reduce water pressure behind the pit walls and allow steeper pit slopes to be developed in the ultimate pit slope walls.

16.6.3 Waste rock storage and management facility

As mining progresses into undisturbed areas the overburden will be removed and stored in a temporary ex-pit storage area. The overburden comprises a volume of 3.5 m thickness over 10 km^2 sq. km excavated at 35 k bank cubic metre per year (BCM) per year. This storage will be temporary as this volume of topsoil will be re-used to rehabilitate the open-pit backfilled area or could also be reutilised as construction material for TSF maintenance in the future.

The volumes of waste rock storage have been estimated from the volume of waste using a swell factor of 1.3. Backfilling will be done without compaction except mining equipment and waste rock weight compaction. The maximal height of these facilities above the ground level will be restricted to 40 m.

The tonnage of overburden to be stored is 1.4 Mt, which will occupy a volume of 0.8 Mm^3 based upon an in situ wet/natural density of 2.3 t/m^3 and a swell factor of 1.3 resulting in a loose or bulk density of 1.79 t/m^3.

Hard rock waste amounts to 10.1 Mt which will occupy a volume of 3.6 Mm^3 based upon an in situ density of 2.7 t/m^3 and a swell factor of 1.3 resulting in a loose or bulk density of 2.08 t/m^3.

16.7 Mining equipment

16.7.1 Selection methodology

MPlan has estimated the loading and hauling equipment necessary for overburden, waste, and RoM mining. Several fleet scenarios were considered for the loading and hauling of the excavated material. Depending on the scenario, the truck and excavator bucket volumetric capacity, payload, or the maximum number of passes were limiting factors.

The following key principles were followed when choosing the appropriate mining fleet:

- at least 2 excavators shall be used for each type of material,
- 4 to 6 passes shall be used to fill the trucks body; and
- the fleets will be standardised for the two pits and will share equipment.

The preliminary estimate of mobile mining equipment from year to year is presented in Table 16-31.

Details of the haulage equipment selected are provided separately in the Mining Supplementary Information Report [10].

16.7.1.1 Rock breaking

Rock breakage at Woxna Mine will be done by drilling and blasting. Top hammer drill has been selected for this task with variable drilling diameter and fleet size according to the planned production rates.

The drilling diameter is function of the bench heights. The average waste thickness of circa 15 m allows 165 mm holes to be drilled, except for the first years of mining where waste thickness is reduced, and 115 mm diameter shall be used. In the graphite+gangue horizon with an average thickness of 5-6 m, the drilling diameter will be 115 mm.

From the planned production requirements, drill and blast estimations have been conducted on parameters such as surface and height to be drilled, number of drillholes, meters to be drilled, and finally the number of drilling rigs required on a yearly basis.

Table 16-31: Mobile mining equipment requirements

	Units	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Average
Tonnes Moved	Mt	1.94	1.07	0.86	1.00	1.02	1.00	1.01	1.01	0.98	1.06	0.94	0.87	0.66	0.51	0.43	0.19	0.16	0.16	0.09	0.79
Average Distance	m	975	1,050	1,125	1,200	1,275	1,350	1,425	1,500	1,575	1,500	1,725	1,800	1,875	1,950	2,025	2,100	500	500	500	1,366
Tonnes x Distance	Mt x m	1,891	1,121	970	1,205	1,296	1,356	1,445	1,515	1,550	1,585	1,627	1,573	1,234	1,001	871	395	80	80	44	1,097
Weekly Rate	t/w	38,783	21,352	17,250	20,086	20,330	20,091	20,276	20,201	19,677	21,128	18,866	17,483	13,163	10,264	8,606	3,757	3,200	3,200	1,757	15,761
Equipment Estimate	**Units**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**	**Y12**	**Y13**	**Y14**	**Y15**	**Y16**	**Y17**	**Y18**	**Y19**	**Average**
Drills	Whole #	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	2
Explosives Truck	Whole #	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shovel	Whole #	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Loader	Whole #	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Trucks	Whole #	5	3	3	4	4	4	4	5	5	5	5	5	4	3	3	2	1	1	1	3
Dozers	Whole #	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor Grader	Whole #	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck	Whole #	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Auxiliary Excavator	Whole #	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

16.7.1.2 Loading

The loading equipment bucket size is selected based on the following two parameters: achieving a good mining selectivity and matching with the estimated mine production schedule. Smaller shovels have will be able to achieve a good selectivity between the different mining zones, faulting areas and types of graphite. Larger shovels are not able to reach the same level of mining selectivity and geology grade control during excavation.

A backhoe shovel excavator is the preferred machinery for the selective mining of the graphite rock. The BCM capacities have been estimated for different bucket sizes with the associated trucks taking into account fill factor, swell factor and cycle time (digging, swigging, waiting and manoeuvring time). The results are compared below.

Table 16-32: Selected 5.0 m^3 Shovel Bucket Characteristics and Estimated Production Rates

	Unit	Base case (40 t payload)		
		Waste	Graphite	Overburden
Bucket size	LCM	5	5	5
In situ Material Bulk density	t/BCM	2.7	2.7	2.3
Swell Factor	-	1.3	1.3	1.3
Bucket Fill Factor	-	0.85	0.85	0.85
Bucket Capacity	BCM	3.3	3.3	3.3
Bucket Payload	t	8.8	8.8	7.5
Nominal Vehicle Payload	T	40	40	40
Number of Passes	no	4	4	5
Actual Truck Payload	T	35.3	35.3	37.6
Vehicle Tonnage Filling Ratio	%	88	88	94
Shovel Total Cycle Time	Min	3.0	3.35	3.5
Shovel Number of Cycles per Hour	No	20	17	17
Output	BCM/h	196	186	195
Output	t/h	705	600	640
Operating Hours per Year (Shovel)	4 620	4,620	4,620	4,620
Production/Year/Shovel	M BCM/a	0.9	0.9	0.9
Production/Year/a	M t/a	3.3	2.8	3.0

Source: MPlan 2021

16.8 Rock Mass Density Considerations

Density tests using the Archimedes principle were carried out systematically on every sample selected for the laboratory tests. No information was provided in regards to the samples being left at ambient temperature and then tested without prior stay in an oven, therefore the measured density is considered as wet/natural.

Bulk density values can be linked to the porosity and the geological nature of the rocks.

17 RECOVERY METHODS

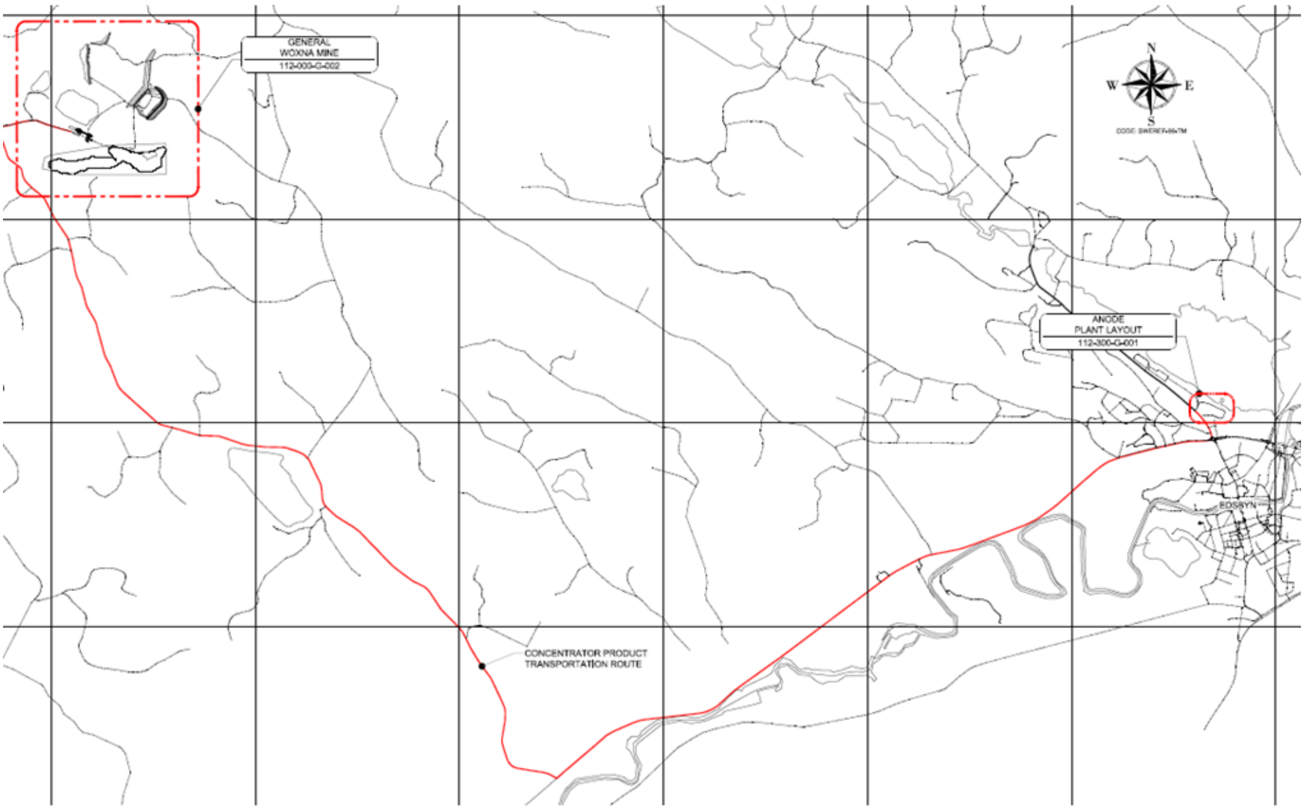
17.1 Design criteria

17.1.1 Basis for design

The basis for the design of the Woxna Concentrator and associated VAP (process design) is the beneficiation of:

- 160,000 tpa of RoM with a grade of approximately 9.2% C by flotation to produce-
- an average of approximately 14,730 tpa of graphite concentrate, grading 92.3% C at the Woxna Concentrator on the mine site, which is then-,

fed to the VAP which will produce an average of approximately 6,604 tpa battery grade CSPG (see Section 2.4) at 99.95% C and an average of approximately 7,479 tpa micronized (by jet mill) graphite at a grade of 93% C. The VAP location for the purpose of this PEA has been selected to be located off the mine site in the nearby town of Edsbyn as shown in Figure 17-1.



Source: Zenito 2021

Figure 17-1: Relative location of the Woxna Concentrator and VAP

The Woxna Concentrator product will undergo several beneficiation stages through the VAP to result in the final CSPG. Sequentially these stages are spheronization to produce 'curled' flakes of graphite; followed by thermal purification and finally coating of the purified, spheronized flakes with additional carbon to produce graphite suitable for use in battery anodes.

The main production criteria for the Woxna Concentrator and VAP are given in Table 17-1.

Table 17-1: Woxna Concentrator and VAP production criteria

Criteria	Value	Unit
Plant life	19	years
Concentrate plant availability	88	%
VAP plant availability	96	%
Woxna Concentrator flotation plant throughput	160,000	tpa
RoM grade	9.2	% C
Woxna Concentrator concentrate production	14,730	tpa
Woxna Concentrator recovery	93.8	%
Woxna Concentrator product grade	92.3	%
Spheronized graphite production	6,629	tpa
Spheronization yield	45	%
Spheronized graphite particle size d_{50}	15	µm
Purified spheronized graphite (Thermal production)	6,629	tpa
Thermal treatment temperature	2,600	°C
Purified spheronized graphite grade	99.95	% C
Coated product - CSPG production	6,604	tpa
Micronized (jet milled) fines production	7,479	tpa
Micronized graphite particle size d_{50}	4	µm
Micronized graphite grade	92.3	% C

The envisaged process design includes the following aspects:

- The basis for the comminution circuit is the existing rod mill. To maximise its throughput the comminution circuit will include a new crushing plant, crushed material stockpile and rod mill classifier.
- The flotation circuit is based on the latest flotation tests which were carried out by BGRIMM.
- The dewatering and bagging circuits are as before.
- Additions are dewatering cyclones in the flotation circuit to increase the slurry density and a clarifier for cleaning the cyclone overflow.
- The existing classification screens for the dry graphite are no longer required.
- The VAP is designed to include milling and spheronization of the graphite flotation concentrate followed by thermal treatment. Fines from the spheronizing are milled again to produce ultra-fine graphite. This will be all new equipment.

17.1.2 Changes to the existing concentrator

The following changes and additions to the existing concentrator are proposed by Zenito to enable the plant to treat 160,000 tpa RoM mineralised material.

- Zenito proposes that a new crushing plant be purchased and be operated by the owner to produce minus 6.3 mm crushed material compared to the previously the outsourced crushing size operation that produced a rod mill feed size of minus 13 mm.
- The crusher will be operated at maximum capacity which is significantly more than that required by the rod mill on an hourly basis, and therefore, will be operated only for one shift per day. A stockpile is used as a buffer between the crushing and milling.
- The existing 260 kW rod mill motor will be replaced by a 360 kW motor so that the rod loading can be maximised.
- A screw classifier will replace the existing cyclone to improve the control of the milling, and improve the particle size distribution to the flotation circuit.

- New scavenger flotation cells will augment the existing scavenger cells.
- In addition to the existing cleaner flotation and regrind circuits, an additional nine cleaner flotation stages with six regrind mills will be installed to replicate the BGRIMM flowsheet.
- A new blower will be required to supply air to the new flotation cells.
- A new, fully automatic, pressure filter will replace the existing filter to maximise the throughput and minimise the moisture in the graphite concentrate cake.
- The existing diesel rotary dryer will be converted to fully electric, which will increase its capacity.
- A new wet low intensity magnetic separator will be installed to remove magnetic minerals (that are sulfur bearing, including pyrrhotite) from the flotation tailings before the tailings is pumped to the Tailings Storage Facility (TSF). These sulfur bearing minerals, including Pyrrhotite, can potentially produce acid water, referred to as the Potentially Acid Generating (PAG) tailings, and will be stored separately from the main tailings.
- Water clarifier

The VAP will comprise new equipment and shall include micronizers and spheronizers, jet mills, furnaces, coating plant and flue gas treatment.

A set of preliminary plant design criteria and mass balance was produced to provide data for sizing the specific unit operations and the main equipment. The criteria were produced from historical production information, testwork and equipment suppliers' information.

17.2 Process description

Block diagrams for the process are provided in Figure 17-2 through to Figure 17-8. The existing equipment is shown in light grey, and the new equipment is shown in blue.

17.2.1 Operating philosophy

The following operating philosophy has been applied:

- Woxna Concentrator crushing plant is operated at full capacity for approximately 8-hours per 24-hours
- The concentrator, from the stockpile to the discharge of the dryer, is operated by three 8-hour shifts per day, 7-days per week, 365-days per year.
- The loading of the dried concentrate is loaded onto trucks on day shift, 7-days per week. The VAP process is operated three 8-hour shifts per day, 7-days per week, 365-days per year.
- Major maintenance is carried out on day shift by contractors and the plant operators perform minor maintenance during the rest of the time.

17.2.2 Crushing and milling

The primary crushing was previously carried out by a mining contractor and secondary by another contractor. It is proposed that Woxna Graphite own and operate the crushing plant.

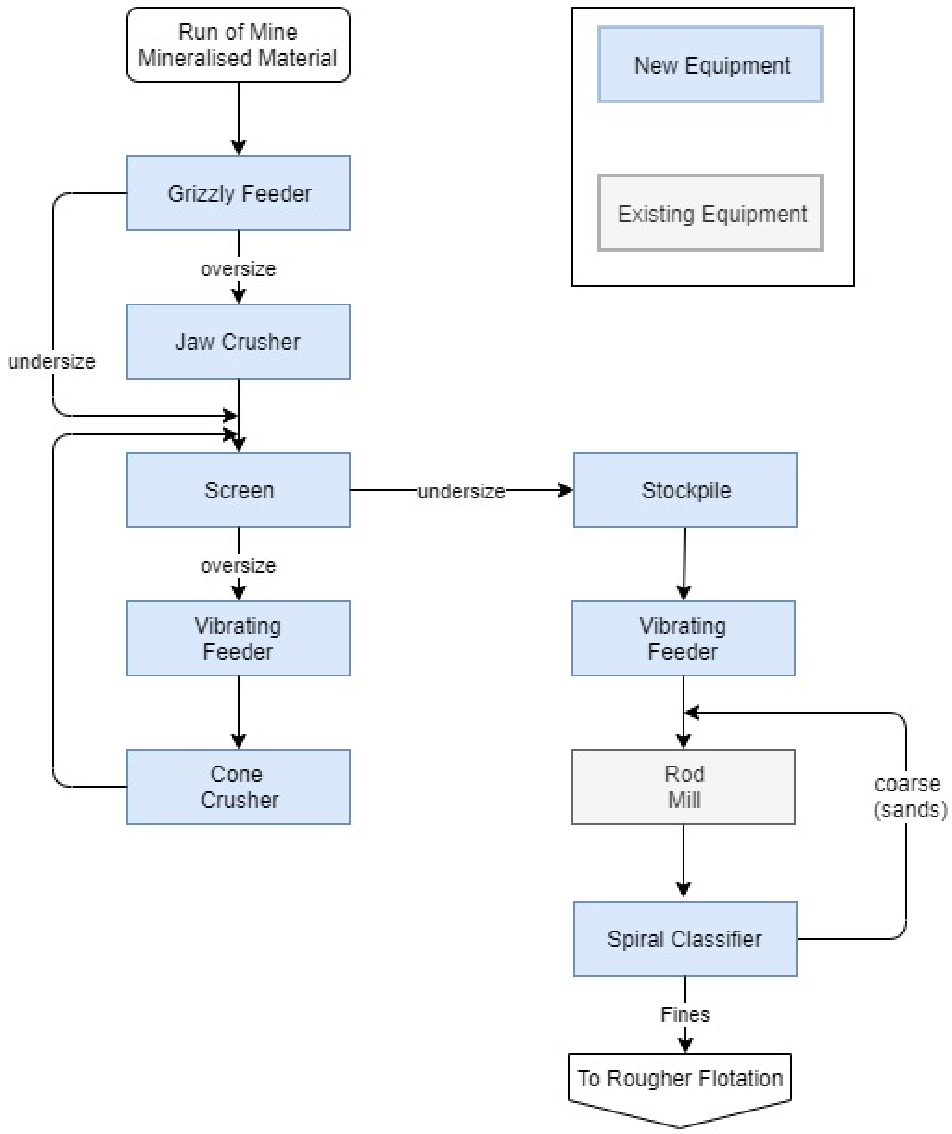


Figure 17-2: Comminution block flow diagram

The design is based on crushing test-work by Metso in 2012. Metso reported that the sample is abrasive, but the crushability is easy. The product from this plant is 80% passing 6.35 mm to maximise the throughput of the rod mill. The crusher operates for one shift per day and feeds onto a stockpile.

The RoM material is fed into the RoM bin directly by a mine haulage truck or from the RoM stockpile by a front-end loader. The RoM material has a top size of 400 mm and is screened by a vibrating grizzly to remove minus 50 mm fraction. The oversize is fed into a jaw crusher where it is crushed to minus 50 mm. The grizzly undersize and jaw crushed products are transferred by conveyor to a triple deck vibrating screen where it is screened at 8 mm. The oversize is conveyed to a bin and then fed by vibrating feeder to a cone crusher. The crushed product is recycled back to the screen. The screen undersize, with 80% passing 6.35 mm, is transferred by conveyor to a stockpile adjacent to the crushing plant. The crushed product is reclaimed by a vibrating feeder and conveyed to the rod mill at a controlled rate.

The milling circuit consists of a rod mill in closed circuit with a spiral classifier. The crushed material from the stockpile is fed onto the rod mill feed conveyor by a variable speed vibrating feeder at a controlled rate of 21 tph. The mass flow is measured by a weightometer on the conveyor belt which controls the feed rate.

Rod milling is used to minimise overgrinding of the graphite. The mill is 2.7 m diameter and 3.75 m long, with a 360 kW motor. The mill discharge is screened by a trommel with the oversize being collected in a scats-bunker for recycling to the mill feed on a batch basis. The mill discharge is diluted and pumped to the screw classifier where to coarse sands, are returned to the rod mill for regrinding and the overflow, the fines, are transferred to the rougher flotation feed tank. The milling circuit produces 80% passing 275 µm.

Water is added to the mill feed to produce a slurry density in the mill of 80% solids.

17.2.3 Flotation and tailings

The rougher flotation circuit consists of roughers and scavengers as shown in Figure 17-3. Conventional mechanical 2.8 m^3 flotation cells with paddles, are arranged in banks of either four cells per bank with feed box-4 cells-discharge box configuration (F-4-D) or three cells F-3-D. The rougher circuit consists of four rougher cells with the concentrate being feed by gravity to the cleaner circuit. The scavenger concentrate is recycled to the rougher and the tailings then pass to second scavenger banks (a and b), to maximise graphite recovery. The concentrate from these two scavengers is recleaned by scavenger bank 3, the concentrate is recycled to the rougher and the tailings back to the first scavenger.

The final tailings from the scavenger circuit are transferred to a wet low intensity magnetic separator (WLIMS) where the magnetic pyrrhotite is removed from the tailings. This pyrrhotite concentrate is dewatered using a vacuum filter and the cake discharged into a bin for storage before being transported to a lined storage area. The tailings from the WLIMS are pumped to the TSF.

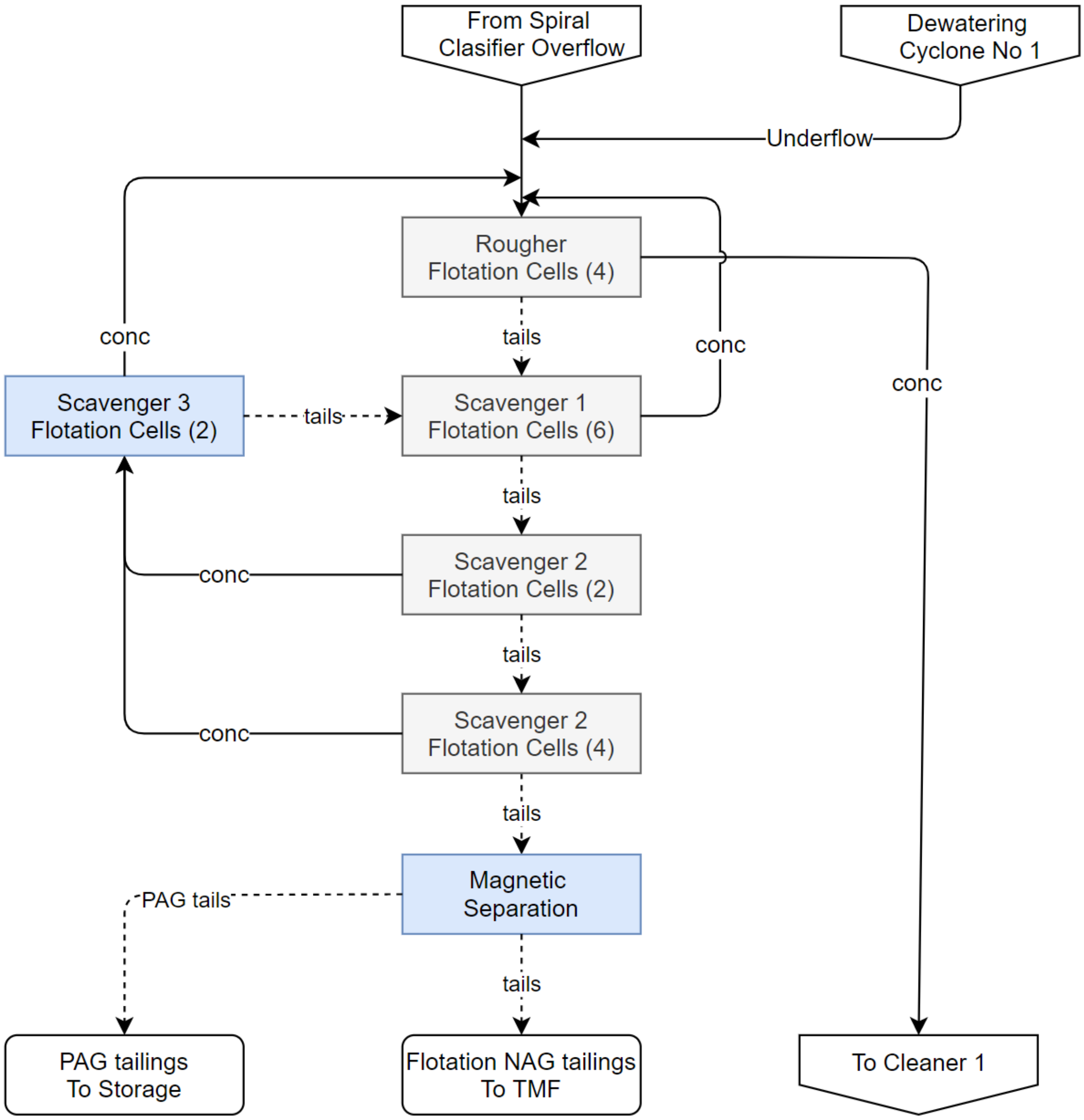


Figure 17-3: Rougher-scavenger flotation block flow diagram

The cleaner circuit has twelve cleaner stages with ten regrind stages as shown in Figure 17-4. The regrinding of the cleaner concentrates liberates the gangue. Conventional mechanical flotation cells, either 2.8 m^3 or 1.4 m^3, are used for all but four cleaner stages, where the existing column flotation cells are used. The mechanical cells are arranged in banks of either four cells per bank with feed box-4 cells-discharge box configuration (F-4-D) or if three cells F-3-D and F-2-D for two cells with the number of cells depending on the retention time required.

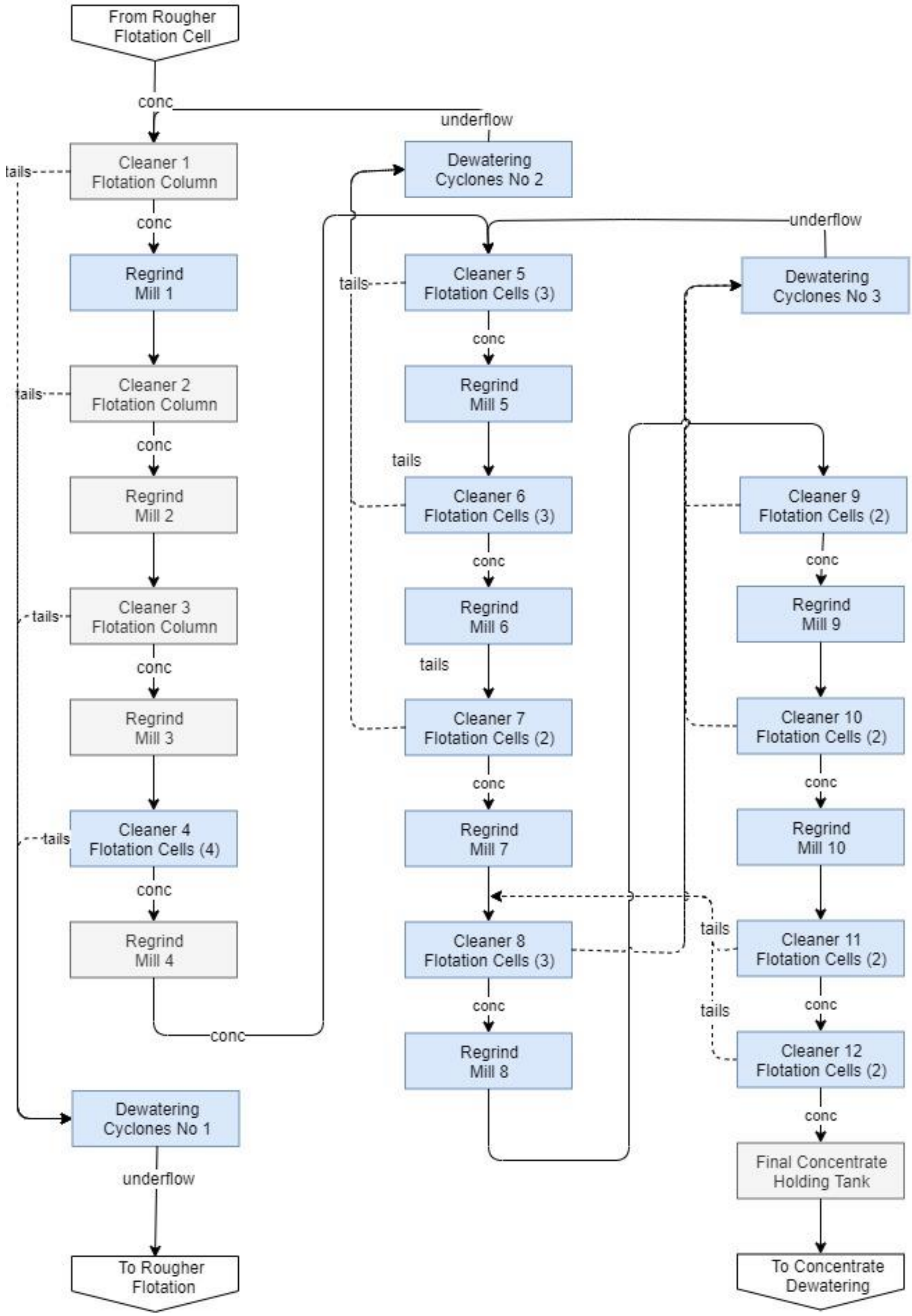


Figure 17-4: Cleaner-regrind flotation block flow diagram

The regrind mills are tower type mills with ceramic media. The regrind mills liberate the gangue from the graphite flakes. The mills have no targeted product particle size but instead operate based upon a 20-minute slurry residence time.

The rougher concentrate is upgraded by a cleaner flotation column. The tailings are recycled back to the rougher and the concentrate is transferred by gravity to a regrind mill to be reground. The product is then pumped to the 2nd cleaner column for further upgrading. This concentrate is then upgraded by two more regrind/cleaner column flotation stages. The tailings from these four cleaner flotation stages are recycled to the rougher flotation. The concentrate from the 4th cleaner stage is then upgraded in six more regrind/flotation circuits to upgrade the concentrate. The concentrate from the 10th cleaner is further cleaned by two stages of cleaning by flotation to produce the final clean graphite concentrate. The tailings from the cleaner flotation are circulated forward to the cleaner stages.

Due to the extensive number of cleaner flotation stages, and the addition of wash water to the flotation cell launders, dewatering cyclones are used at three points in the cleaner circuit to remove water from the cleaner tailings before the tailings are reground. The water in the cyclones' overflow is recycled to the rod mill.

The flotation circuit graphite recovery is 93.8% at a 92.3% C graphite concentrate grade, and a mass pull of the graphite concentrate of approximately 9.8% of rougher feed.

The existing flotation area is shown in Figure 17-5 below.

Figure 17-5: Existing flotation area

17.2.4 Dewatering and drying

The concentrate from the final cleaner is pumped to an agitated holding tank as illustrated in Figure 17-6. From there the concentrate slurry is pumped on a batch basis into a vertical plate filter, where it is dewatered. The concentrate cake drops onto a conveyer and is transferred to the electrical rotary dryer. The product of the dyer is completely dry. The graphite is transferred to a storage silo. The filtrate from the filter is pumped to the clarifier.

The dry graphite is transferred from the storage silo to dry-powder tankers for dispatch to the VAP.

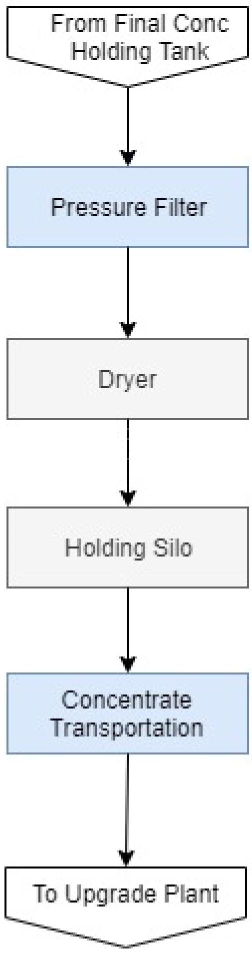


Figure 17-6: Dewatering and drying block flow diagram

The existing pressure filters are shown in Figure 17-7 below.

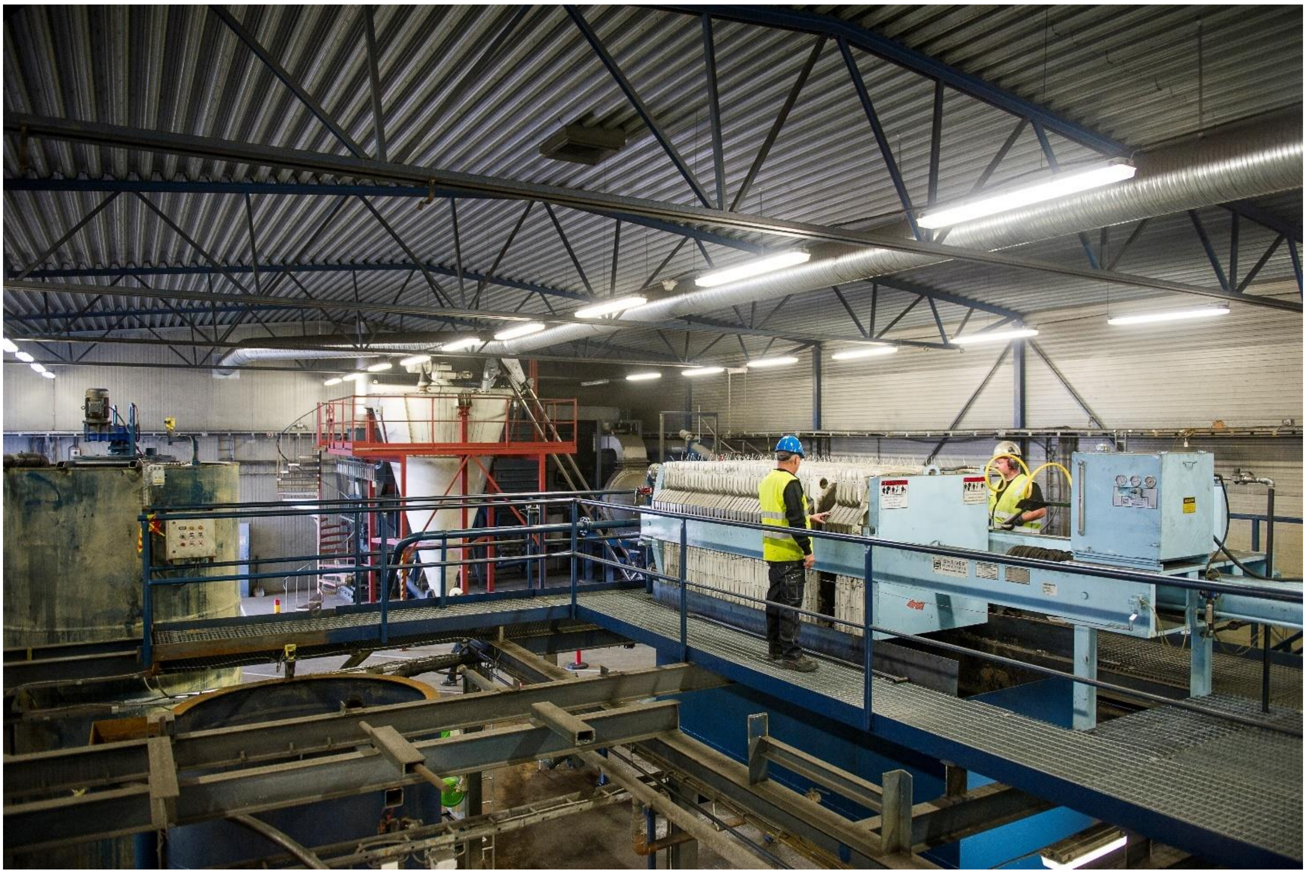


Figure 17-7: Existing filter presses

17.2.5 VAP spheronization, purification and coating

The dry graphite flotation concentrate is upgraded by spheronization, thermal purification and coating to produce battery grade carbon or CSPG as shown in Figure 17-8. A by-product is fine graphite which is micronized in a jet mill.

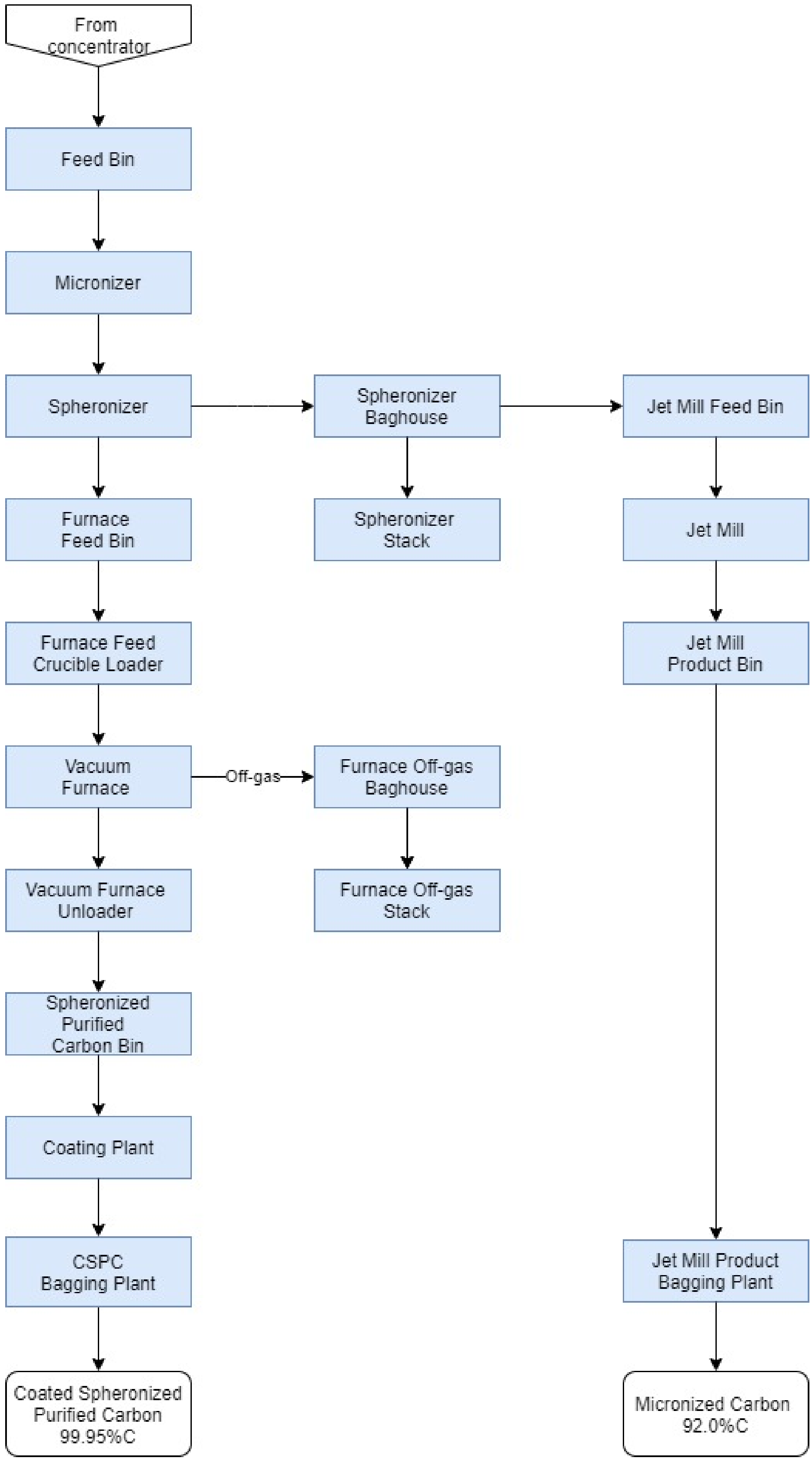


Figure 17-8: VAP block flow diagram

The dried flotation concentrate is transported in tankers from the storage silo to the spheronising and micronization line. The graphite will be finely milled, spheronized, and classified to produce spherical graphite. This is transferred to the purification area storage/feed bins.

Four purification furnaces of the pusher type are installed in parallel. Spheronized graphite is fed from the feed silos into the furnaces which heat the graphite to 2,600 °C and hold it at this temperature for a period of time before cooling the material to remove the impurities. The furnaces are heated electrically. The furnaces are fed and emptied on a semi-continuous basis and the hot gas is recycled through heat recovery circuits.

Testwork conducted to date has shown that purified products of up to 99.99% C are achievable using the thermal process.

The purified graphite is transferred to storage silos and then transferred to the coating plant.

17.2.6 Coating

The area for the coating facility is located within the VAP facility building alongside the balance of the VAP plant.

The purified graphite is processed by a proprietary technology to produce the coated graphite.

17.2.7 Water supply

17.2.7.1 Water supply – Woxna Concentrator

Most of the water needed by the concentrator effectively circulates through the tailings system and back to the concentrator as return water. Any makeup water required by the site may be sourced from boreholes or the Älman River.

The overflows of the dewatering cyclones in the cleaner flotation circuit are recycled to the rod mill discharge as dilution water. Any solids in this water are recycled.

Filtrate from the concentrate filter is pumped to a clarifier to clean the water before transferred to the process water tank.

Tailings decant water is pumped back from the TSF to the clarifier for use in the plant.

The clarifier underflow is pumped to the rod mill. The clarifier overflow is pumped to the process water tank for recycling. Flocculant is added to the water fed to the clarifier to assist settling of solids and the clarity of the overflow water.

The process water is pumped through a ring main. Froth launder spray water is taken off this ring main as well as mill dilution water.

The operations which require clean water, for example, gland service, use filtered fresh water supplied from a borehole.

17.2.7.2 Water supply – VAP

Raw water, for the furnace cooling phase heat exchangers, is available from lake Ullungen which is next to the industrial facility chosen as location for the VAP for the purposes of this PEA. Any potable water for the VAP is sourced from the local Edsbyn municipal water system.

17.2.8 Sampling and analysis

The sample preparation and analysis requirements of the Project will be provided by two separate laboratory facilities, one located at the Woxna Mine site and the other at the VAP site. These analytical facilities are to be used to conduct on-site assay of the mine feed for sulfur and carbon as well as process product analysis for carbon and sulfur content, moisture, and particle size.

17.2.8.1 Woxna Mine site laboratory

The current laboratory at the mine site will be used for flotation control and carbon and sulfur content analysis. Automatic samples are taken of the feed to the spiral classifier overflow, scavenger tailings, final tailings, and final graphite concentrate in the concentrator.

17.2.8.2 VAP site laboratory

A new facility at the VAP site will be constructed to house the laboratory equipment for size analysis, final grade analysis as well as tap density, moisture etc. Automatic samples are taken of the spheronized product, the feed to the furnaces and the purified graphite and the coated product.

17.2.9 Ancillary facilities

Both the Woxna Concentrator and the VAP require additional facilities that provide and control air supply to various points of the process plants as summarised below.

Woxna Concentrator:-

- air for the mechanical flotation cells is provided by air blowers;
- air for the flotation columns is provided by a dedicated air compressor;
- instrument and general plant air are supplied by an air compressor;
- flotation reagents are dosed using peristaltic pumps to the flotation cells from the storage containers. Reagents are purchased and delivered in ready to dose form and are pumped directly from their delivery vessel; and
- an area dedicated to storage of the required chemicals is provided outside the plant

VAP Facility:-

- the atmosphere in the pusher furnaces is an inert agon to prevent the combustion of the graphite;
- the hot off-gas exits the purification and coating equipment and is ducted to a gas handling circuit which will remove the impurities in the gas, vent and scrub excess gasses, and recycle the heat energy; and
- Blowers and compressors for the micronizing and gas handling system are located in a dedicated area within the plant.

17.3 Consumables

17.3.1 Comminution

Crusher wear parts are replaced approximately twice a year.

Additional rods for the rod mill are added based on the rod mill power load.

Rod mill liners will be replaced approximately twice a year.

17.3.2 Flotation reagents

The methyl isobutyl carbinol (MIBC) frother and diesel collector are added to the spiral classifier overflow slurry before being pumped to the rougher flotation. The flotation requires a total of 176 g/t (mill feed) diesel (collector) and 176 g/t MIBC (frother).

The air for the mechanical cells is supplied by an air blower and the air for the columns by an air compressor.

17.3.3 Furnace consumables

Argon gas is used in the furnace to shield the spheronized graphite and prevent oxidation occurring at the high temperatures. The argon gas is also used as a heat transfer medium during the cooling phase of the furnace. An estimated 2,050,000 Sm^3 of argon is consumed per annum by the thermal treatment furnaces.

17.3.4 Coating reagents and consumables

Isopropyl alcohol is utilised as a solvent in the coating process. Pitch tar is used as the source of carbon for the coating. The coating requires 33.7 kg/t of Isopropyl, and 29.4 kg/t of pitch.

Nitrogen gas is used in the carbonization furnace to shield the coated graphite and prevent oxidation. The nitrogen gas is also used as a heat transfer medium during the cooling phase of the furnace. An estimated 3,750,000 Sm^3 of nitrogen is consumed per annum by the coating furnaces.

17.3.5 Energy

All electrical energy will be supplied from the local grid and the electrical infrastructure is discussed in Section 18.2.2 and the associated costs are provided in Section 21.3.3.6. Refer to Section 18 for description of electrical infrastructure. Refer to Section 21.3.3.6 for electrical power consumption and associated costs.

17.4 General arrangement drawings

The concentrator is located at the Woxna mine. The general arrangement drawings for the crushing plant and the concentrator are shown in Figure 17-9, Figure 17-10, and Figure 17-11. The general arrangement drawing for the VAP Facility is shown in Figure 17-12.

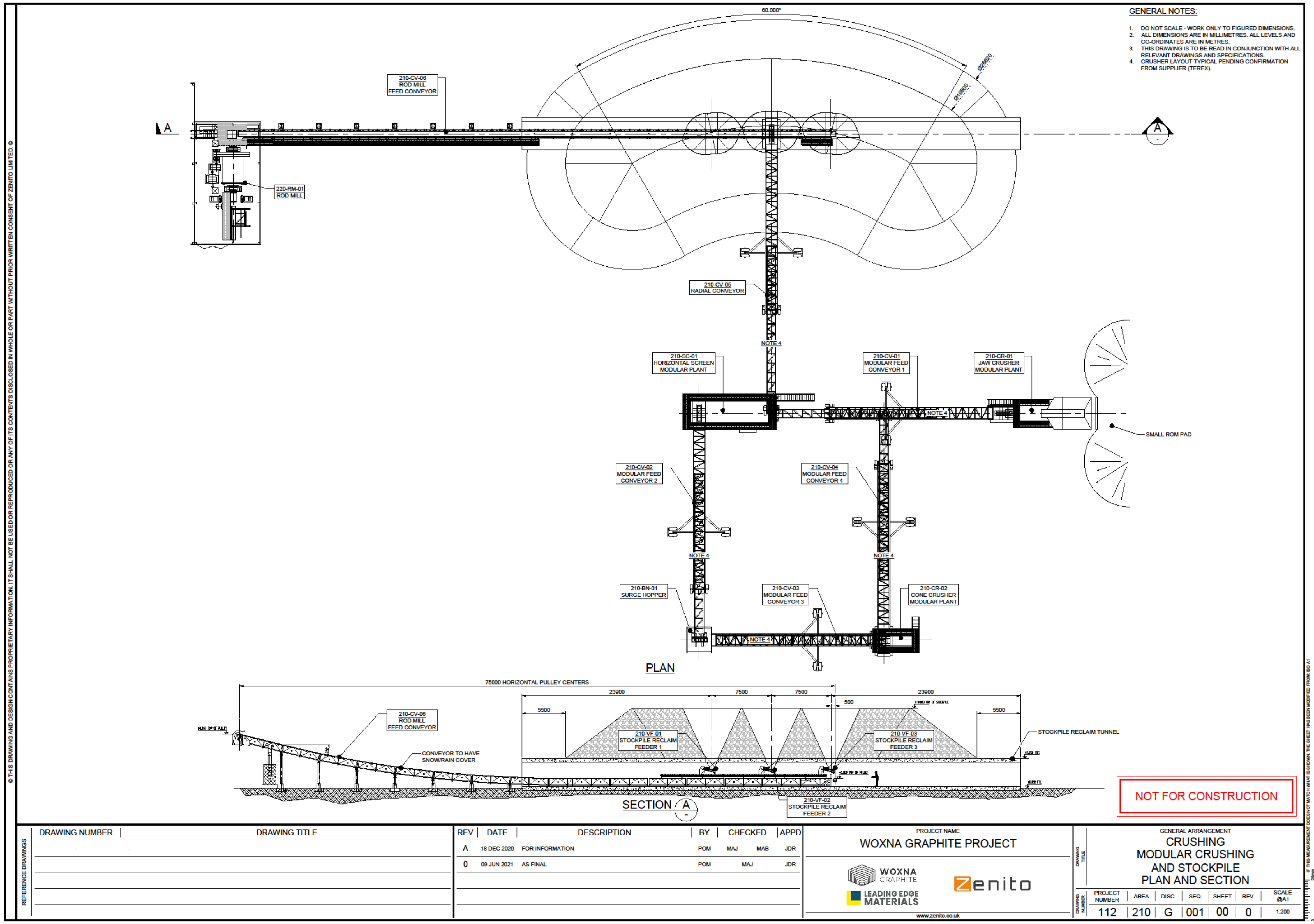


Figure 17-9: General Arrangement - Crushing

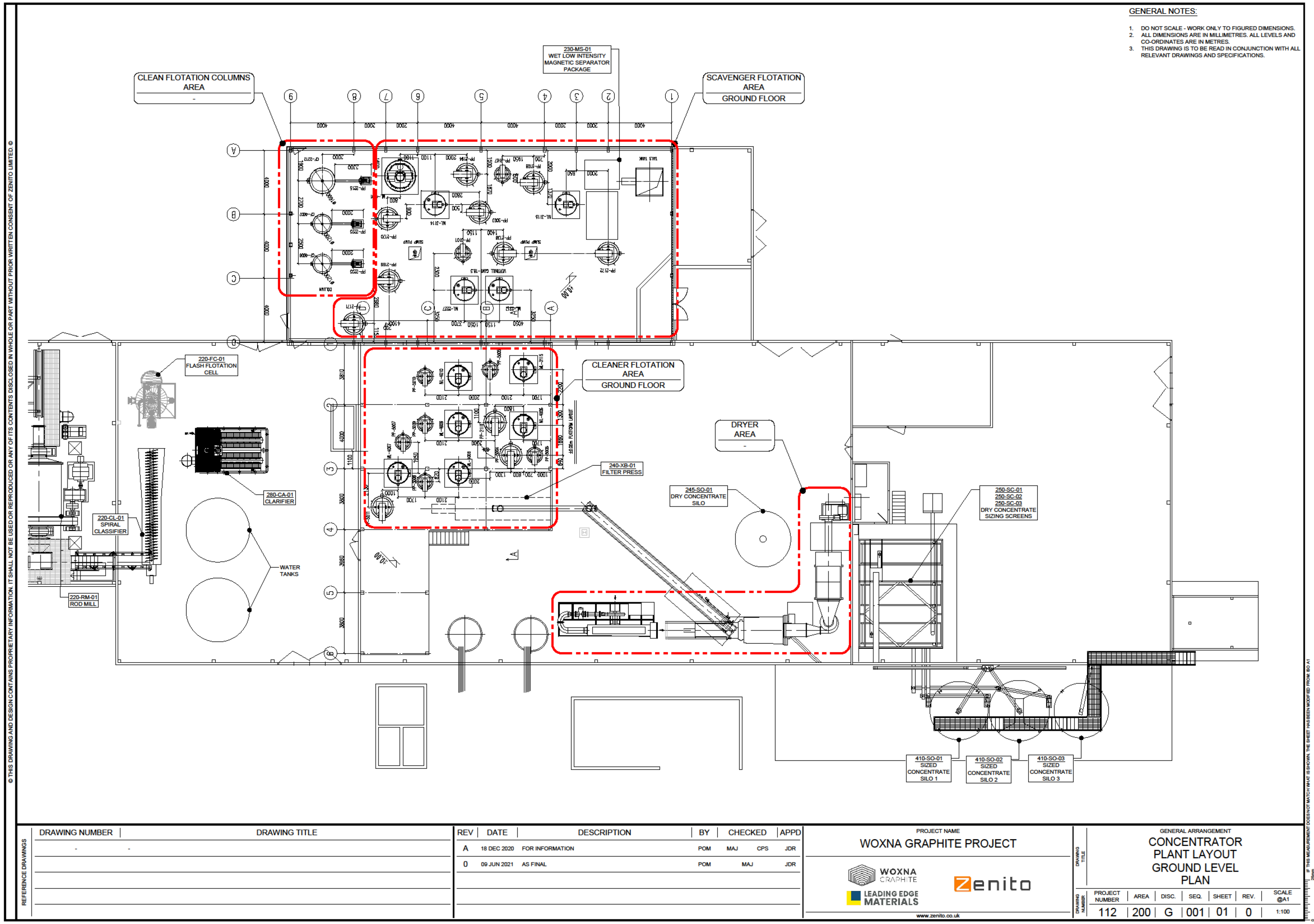


Figure 17-10: General Arrangement – Concentrator – Ground Level

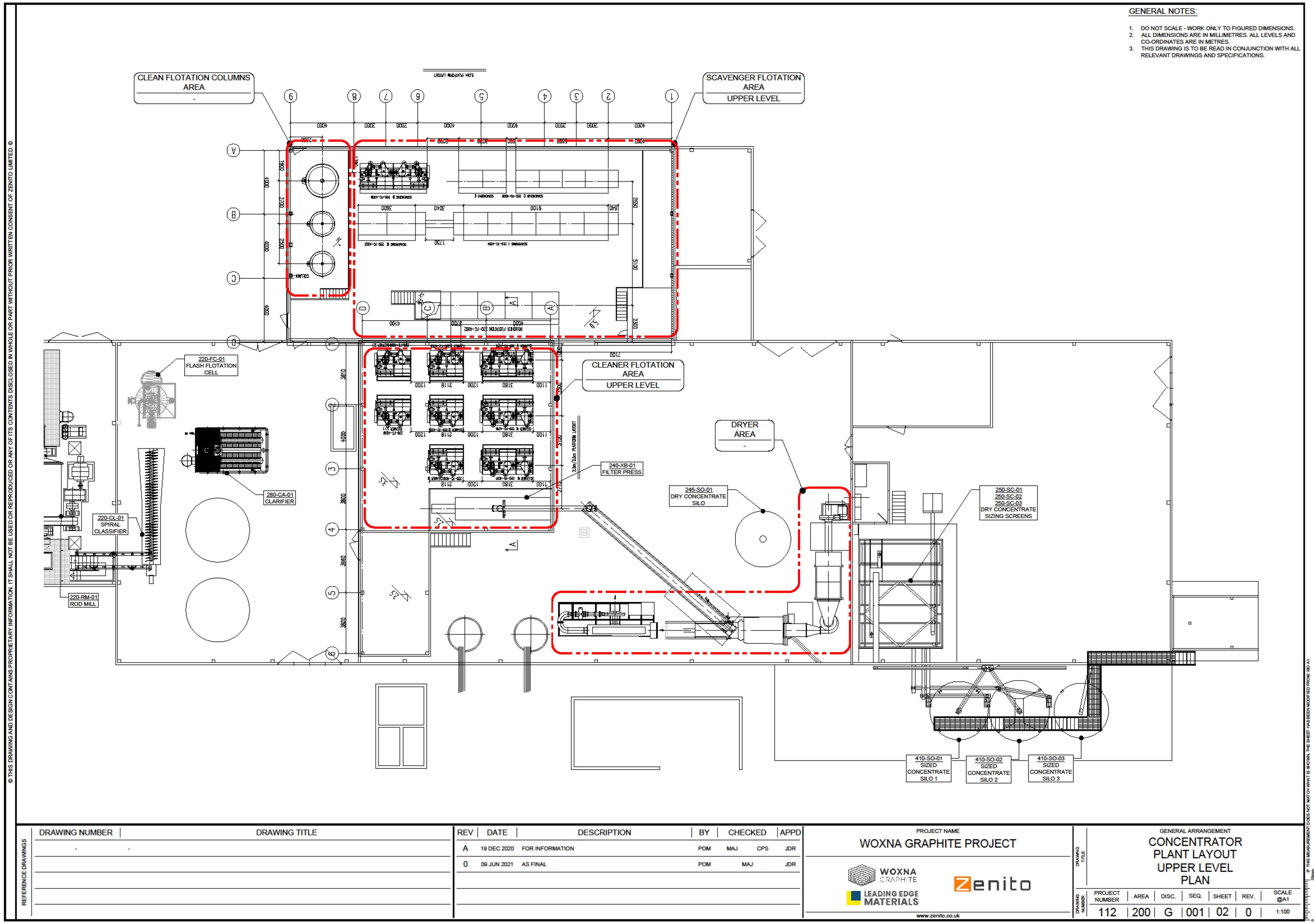


Figure 17-11: General Arrangement – Concentrator – Upper Level

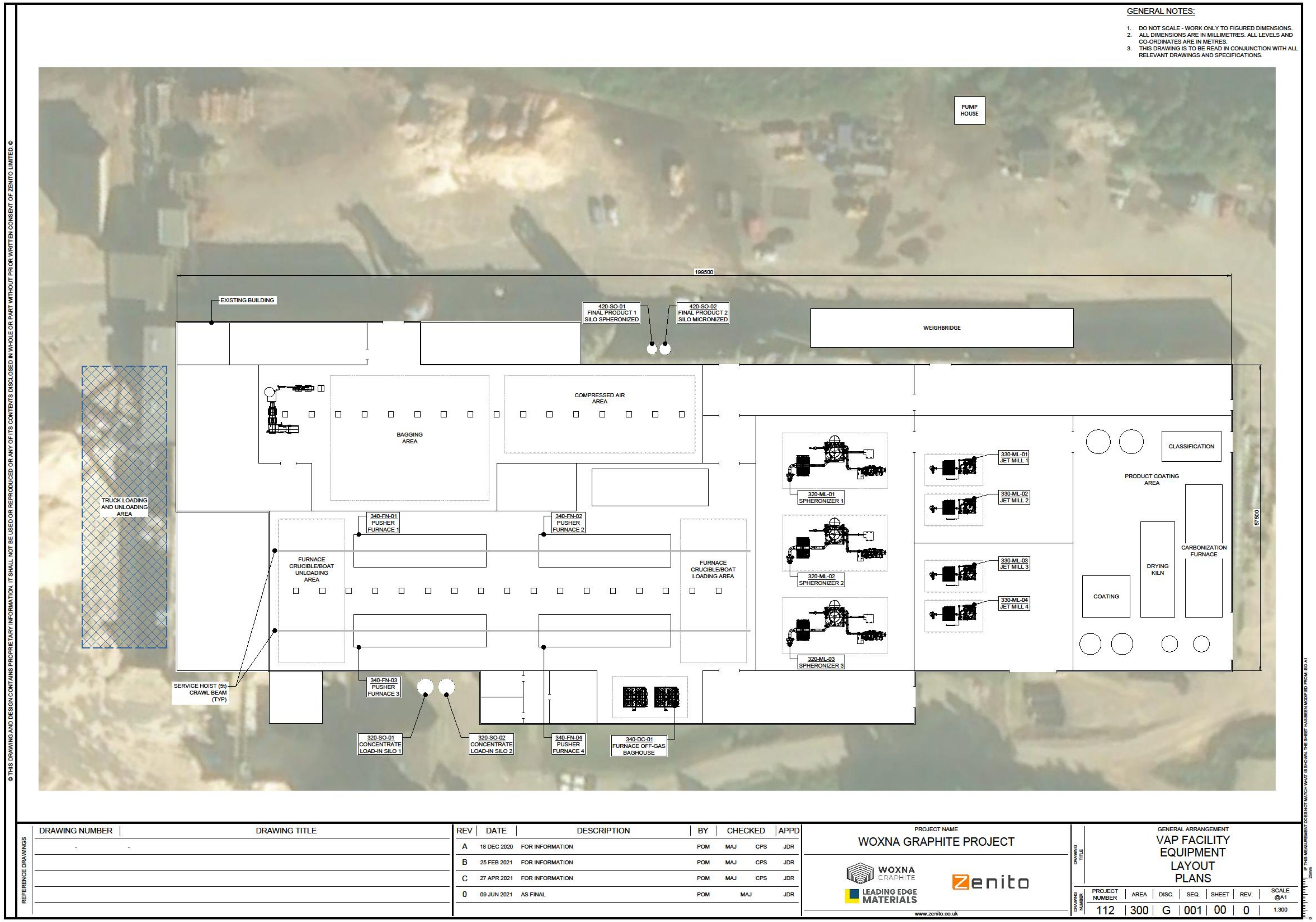


Figure 17-12: General Arrangement – VAP Facility

18 PROJECT INFRASTRUCTURE

The Woxna Mine site has a partially depleted existing open-pit, TSF, waste rock dump areas, mine site roads, clarification ponds and a processing facility as shown in Figure 18-1 and Figure 18-2.

Figure 18-1: Processing plant at the Woxna project site

Figure 18-2: Open pit at the Woxna Mine

18.1.1 Roads

The mine shares the access, of approximately 9.5 km, from national Route 301 to the site gate. The road is of unsealed till/moraine construction and during periods of precipitation/snow and freeze /thaw, requires consistent maintenance. The mine is in a cooperative with the local community that reside along the road and contributes to its upkeep. The financial contribution is apportioned to share of traffic, and a suitable allowance has been made in the operating cost estimate to cover this.

The current access and transports to and from the mine has been agreed on within the current environmental permit however, it has always been a concern for residents along this road residing in Älmesbo village. Therefore, the mine has investigated an alternative access road.

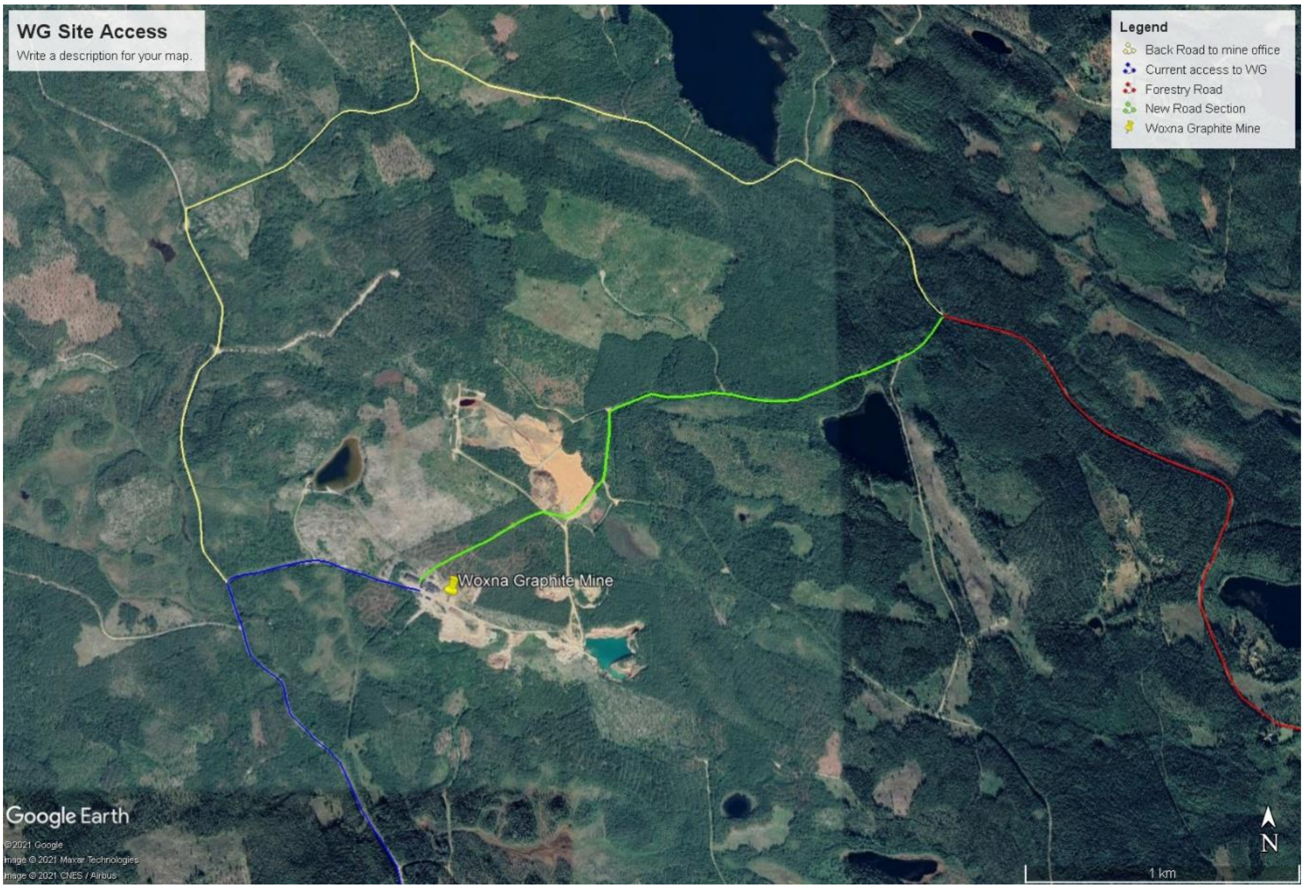


Figure 18-3: Project roads

The alternative road shown in Figure 18-3 consist of two parts. Firstly, to construct a new road section shown in green with a total distance of 2.55 km from the mine to an existing forestry road shown in red. The second part of the road towards the main road 301 is well maintained as a forestry road and farm access road. However, this road will require upgrading to allow for the addition traffic. The cost for construction of the new road section is estimated at USD 265,000.

The current forestry road section shown in red is 8.6 km long and is currently maintained by Edsbyn's Västra Samfällighetsförening (Road Association. The cost for the upgrade of this road could be shared between the Road Association and Woxna Graphite. As this is a state registered road, similar to the current road, Woxna Graphite may have the road upgraded at a reduced cost after negotiation with authorities.

Additional advantages of the new evaluated road, it will reduce traffic in Älmesbo village with social, environmental and safety benefits to the local community. Environmental advantages as the route will be shorter to the VAP facility as well as operational advantage for piping to tailings facility. It is recommended that as a next step a more formal trade-off study is to be performed between the current permitted road solution and the newly evaluated alternative.

Site roads for access and maintenance experience low levels of traffic and are of basic unsealed till/moraine construction. Maintenance is performed on an as needs basis using mine and plant equipment and contributes a negligible cost to operations.

Haul roads will be constructed and maintained by the mining contractor and included in its operating cost.

18.2 Concentrator site services

18.2.1 Fuel supply, storage, and distribution

Current diesel storage on site is a tanked facility, positioned adjacent to the main building within a bunded area. The primary consumer of diesel, the concentrate dryer, will be converted to electric heating as an operating cost savings measure.

The mining contactor will provide and operate its own diesel storage facility.

18.2.2 Utilities

18.2.2.1 Power distribution

There are currently two 1,000 kVA transformers installed in a Transformer / medium voltage (MV) switchgear / low voltage (LV) switchgear building located close to the plant.

The transformers are of different vector groups (T1 = Dy 0; T2 = Dy 11) and can therefore not be connected to the bus in parallel. Attempts to do this in the past have resulted in the bus feeder breakers tripping.

The transformer rooms are undersized and have inadequate ventilation or air filtration. Part of the transformer building is a common MV (12 kV) and LV (400 V) room.

A new MV 1,900 kVA substation is to be installed and supplied in a kiosk type enclosure, including MV switchgear with spare feeder to allow for future expansion. This would remove the need for the MV switchgear in the existing location thus separating the LV and MV switchgear.

Power will be distributed throughout the process plant on existing and new circuits from the plant main LV switchgear assembly, which will remain in the existing substation building. Reticulation throughout the plant will be via radial circuits at 400 V.

There is an additional electrical room inside the plant building and opposite the MV/LV building which houses two PFC (Power Factor Correction) panels. This is assumed to be sufficient for the new and existing loads and no allowance has been made for additional PFC. Allowance is made for testing and refurbishment.

18.2.2.2 Motor Control Centers (MCC)s and Motors

From breakers in the existing LV distribution room a number of existing starter panels (approximately 20) are fed throughout the process plant infrastructure. These panels shall be tested and refurbished where required prior to re-commissioning.

A new MCC room will be installed to house the motor starters for the new process loads. Variable speed drives (VSDs) where required will be supplied as part of the MCC package.

The mill motors and drives are to be replaced. A VSD has also been included for the rod mill.

Basic motor tests on all other existing plant motors will be required prior to re-commissioning.

At present, the motors can only be started from the Programmable Logic Controller (PLC) / Supervisory Control and Data Acquisition (SCADA). There are no field start or field emergency stops installed at the motors. However, every motor has a local isolator. Some local isolators are located in positions which are not easily or safely accessible. The only way a motor can be stopped in the field is by activating the local isolator. The motor starter circuit must be modified for emergency stops and start-up warnings (where required). Installation of an emergency stop near each motor and associated cabling is required as a minimum however, local control stations are allowed for to facilitate plant operation by a small crew.

18.2.2.3 Cable sizing and selection

All new cables will be sized and selected according to Swedish or equivalent international standards. All new power and control cables will be steel wire armour cables for mechanical protection.

18.2.2.4 Emergency power

No allowance has been made for emergency power.

18.2.2.5 Control and instrumentation

The plant motors are all started and stopped from the PLC/SCADA system. Currently there are no field start or stop pushbuttons in the plant. Therefore, the PLC is an important requirement in order to operate the plant.

The old, damaged PLC was replaced in 2014 with an up-to-date PLC.

The flotation columns level control loops are using outdated instrumentation (Fisher Bailey Porter). These are to be replaced with ABB Instrumentation.

18.2.2.6 Vendor packages

The proposed crushing plant package will be vendor supplied with on-board electrical switchgear, motor starters and controls. A spare LV feeder breaker in the LV distribution panel will be utilised to supply this package

The pressure filter will have its own control supplied as a vendor package. This will be for the control of all functions of the filter as well as quoted accessories and process control functions.

There is a drying plant with a Triplex dryer which is supplied by 'Gebrüder Pfeiffer AG Kaiserslautern' as well as a packing plant. Both plants are vendor packages. The local control panels of these vendor packages are supplied from the LV room.

The graphite packing plant local control panel was to be upgraded to PLC control in 2014.

18.2.3 Communications

The mobile network coverage at the Project area is good, and an internet connection is currently established.

18.2.4 Buildings

18.2.4.1 Workshops and warehouses

A warehouse/storage facility of approximately 950 m^2 is located adjacent to the process plant. This area will be sufficient for process plant spares and graphite product storage. This building is enclosed and lockable, though not heated.

Due to the manning structure, most maintenance will be performed either in-situ or in a contractors' facility in the nearby town, and therefore, a large, dedicated maintenance area with cranage is not required on site. There is some space adjacent to the gravity equipment that has been used for small maintenance projects. This area should remain the location of these works, as well as tool storage, as this area is part of the heated building.

The mining contractor will provide and operate their own maintenance facilities.

18.2.4.2 Office facilities and accommodation

There is an office that is currently in use by the mine staff of approximately 200 m². Several semi- portable office sections have been installed expanding the office space to approximately 300 m². The condition of the office is fair, however with a longer term view a new office will be required, although no allowance has been made for this in the PEA.

Potable water is currently brought in by water tanker to a small on-site storage facility. This will continue during operations due to the low cost and adequate service.

There is a small accommodation block of approximately 120 m² adjacent to the administration building. It has four rooms and a shared kitchen facility. It is currently utilised for short stays by visitors. It is not envisaged that rotation staff will require this accommodation for shift purposes, and it will remain in service for intermittent use only.

The milling building has a recently refurbished bathhouse to support the intended crew size.

18.2.4.3 Laboratory

A prefabricated laboratory building has been installed adjacent to the process plant building so as to be free from contamination of dust and vibration from the plant. The lab is capable of processing approximately 10 mining samples, 16 plant samples, and 50 product samples per day for size analysis, carbon content, sulfur content and moisture content.

18.3 VAP Facility

18.3.1 Utilities

18.3.1.1 Power supply and distribution

Current power supply to VAP location selected for the purpose of this PEA is 2 MW and includes a transformer installation and switchgear. An additional 7.6 MW is required for the VAP facility, a total of 9.6 MW. Additional transformer rooms are available at the facility to upgrade the power supply. To meet the increased power demand, an additional 3.5 km cable will be installed from the main substation to the VAP facility at a budgeted cost of USD 1.7 million. New transformers and switchgear will be installed.

All areas in the proposed VAP facility are equipped with lights and 220 V power supply. Additionally, all the major areas are equipped with 400 V 3-phase power points. Existing cable racking is installed throughout the premises to facilitate additional power installation.

18.3.1.2 Emergency power

No allowance has been made for emergency power.

18.3.2 Water

Raw water, for the furnace cooling phase heat exchangers, is available from lake Ullungen which is next to the VAP.

Raw water was extracted from the lake during previous operations at the facility. An existing pump house is installed next to the lake with water piping however, the pumps have been removed. New pumps and piping will be installed to ensure sufficient supply.

Potable water for the VAP is supplied from the local Edsbyn municipal water supply.

18.3.3 Buildings

The existing VAP facility building has a usable area of approximately 11,500 m^2. The building is in good condition, and ongoing maintenance of the facility has been carried out after the shutdown of previous operations.

18.3.3.1 Workshops and warehouses

The existing facility allows for ample workshop space, as well as storage facilities in the basement of the main building.

18.3.3.2 Office facilities

The existing building is equipped with several offices, spaced throughout the premises for operational control. Change house and ablution facilities are in place and could accommodate a total of approximately 90 employees with separate space for male and female personnel. There are three separate areas equipped as lunchrooms for personnel.

19 MARKET STUDIES AND CONTRACTS

For the purposes of this section the Qualified Person has relied on information pertaining to market studies and material contracts provided by Leading Edge Materials Corp with the sources referenced within the section including:

- Benchmark Mineral Intelligence, "Uncoated & Coated Spherical Graphite Market Overview for Leading Edge Materials Corporation," 2020.
- Leading European Graphite specialist consultancy (Name of consultant withheld due to confidentiality), "Market Information on Micronized Graphite from the Woxna Mine," 2021.

The Qualified Person has reviewed the information provided by Leading Edge Materials Corp believes this information to be accurate and adequate for use in this PEA technical report.

19.1 Introduction to graphite

It is the unique structure of graphite, that separates it from any other mineral ever used in industrial and commercial applications. It has opened the possible technological advances we are currently experiencing and foresee within the near future. The remarkable properties of graphite are being harnessed and used in a variety of distinctly different applications, from uses in batteries, fire resistant buildings to nuclear applications.

The significant interest for this assessment is graphite's importance as a key component in batteries, which is creating expected exponential growth of flake graphite demand. The market drivers experienced globally and the EU are to transform into a cleaner, circular and energy sustainable society.

19.1.1 Properties of graphite

Some of the key properties which sets graphite apart include:

- Exceptional conductor of heat and electricity;
- Ability to maintain its stability and strength up to temperatures in excess of 3600 °C;
- High corrosion resistance due the chemically inert nature;
- Natural lubricant as a result of the layered structure;
- High natural strength and stiffness;
- Incredibly lightweight reinforcing agent;
- Insoluble or hydrophobic; and
- Expandability.

19.1.2 Deposit types

Both natural and artificially made graphite (synthetic graphite) exist. Naturally occurring graphite can be found in three forms: flake, vein and amorphous. Synthetic graphite is produced using by-products from oil refining as feedstock.

19.1.3 Flake

Flake graphite is a mineral created by geological conditions of immense heat and pressure generated during the stages of metamorphism of carbon-rich shales and limestone material. Flake graphite was disseminated typically through rocks such as marble, schists and gneisses in these metamorphic formations, often in pocket pods or distributed throughout the parent matrix. Flake graphite is the most commonly occurring type of natural graphite found.

Structurally, flake graphite is made up of an infinite array of carbon atoms. One of these carbon atom layers in isolation is referred to as graphene. Within each single graphene layer one carbon atom is connected to three strongly bonded hexagonal lattice structures. While the links between graphene layers are weak enabling each to slide over one another. Flake graphite consists of free electrons that are able to move without restrictions through the planar structural layers empowering the graphite to become a conductor of electricity, heat and even prone to light absorption.

Classification:

Naturally extracted flake graphite is classified by the carbon content percentage, more commonly referred to as 'grade' ("Cg" – carbon grade or "TGC" – total graphitic carbon), and the flake size (small, medium, large, jumbo, super jumbo).

The concentrated product after processing is classified according to the carbon purity percentage in relation to the percentage of impurities still present in the concentrated product.

19.1.4 Vein

Vein graphite was formed by the transport of solid graphitic carbon from subterranean environments through high temperature fluids depositing the carbon into formation fractures creating crystalline veins. Vein graphite is one of the rarer forms of naturally occurring graphite although the highest in carbon content and most commonly found in Sri Lankan deposits. Due to the lack of common occurrence vein graphite does not appear in too many applications.

19.1.5 Amorphous

Amorphous graphite is the result of organic anthracite coal undergoing extreme conditions of metamorphic heat and pressure in a vastly different process opposed to the other deposit types. This happens through the process whereby the heat destroys the organic molecules and dissipates the other elements such as oxygen, nitrogen and sulfur with only carbon remaining and crystallising as a fine-grained mineral graphite. Due to the microcrystalline structure, fine flake size and the lowest purity of all graphite types, the market value and value-added applications for example batteries is fairly limited.

19.1.6 Synthetic (vs natural)

Synthetic graphite: is produced using by-products from oil refining, such as calcined petroleum coke and coal tar pitch, as a precursor material. The by-product is milled, pressed and then baked at high temperatures for several weeks to graphitize the material. Varied applications range from steel furnacing, graphite electrodes, reinforced plastics, nuclear reactors and batteries.

The shortfalls for synthetic graphite are:

- The cost to produce synthetic graphite is significantly higher. While naturally occurring graphite is cheaper and offers supply growth potential;
- Power demand to run the processing plants;
- Deeply reliant on the oil industry for petroleum by-product supply security;
- Increasing environmental pressures on the oil industry;
- Oil price slumps;
- Natural graphite purification technology improvements with environmental advantages increases competitiveness in the market.; and
- Natural purified flake graphite provides better electrical and thermal conductivity;

19.1.7 Mining and processing methods

The preferred way of mining naturally occurring flake graphite is through drill and blast techniques in an open pit extraction setting. Generally, the overburden stripping ratio and the hardness of the rock formation has a direct impact on the cost of mining. Softer formations can be found in dry or tropical settings where the oxidised zones are deeply weathered and open up to easier free dig techniques.

Once the graphite-rich rock ore has been unearthed, the graphite needs to be extracted and separated from gangue in the form of a concentrate (typically +95% TGC) for industrial useable specifications. A key target historically throughout the processing stage is to preserve the flake size therefore, technical care needs to be maintained during the crushing, grinding and flotation stages so as to liberate flake graphite and remove waste. At this stage the natural graphite concentrate may be sold to an industrial manufacturing end-user that would process the concentrate even further into a required form dependent on the final application.

For battery use the flake graphite needs to be sized, shaped, purified and coated to achieve the required electrochemical performance characteristics demanded by the battery producers. Sizing and shaping is performed by using various milling and shaping equipment. Purification has traditionally been done by using strong acid leaching whereas the general market trend is moving towards more sustainable purification methods like thermal purification. The final step is to add a surface modification to the spherical purified graphite particles in order to increase conductivity, increase cycle life of the battery and improve safety.

19.2 Graphite end-markets

The vast majority of the graphite supply for battery applications has traditionally been from the synthetic graphite producers supplying a narrow group of end-users. The market share in the energy storage sector for the use of natural flake graphite is continuously increasing, driven by technological improvements, increased amount of research & development therefore, more sustainable product for end-users.

19.2.1 Traditional markets

- Steel and Iron-casting industry – recarburises (assist in hardening the steel composition);
- Foundries – variety of metal cast moulding, sand casting (enabling the shaping and release of metals);
- Lubricant producers – air compressors, ball bearings, oil lubricants, railway (acting as a natural dry lubricant);
- Automotive – brake pads, clutches, drums (highly resistant friction material);
- Electrodes – electric arc furnace steel production (only capable from synthetic graphite properties); and
- Refractories – linings of furnaces, incinerators, reactors, kilns, as an additive in magnesia-carbon and alumina-carbon brick refractories (assisting in high thermal resistance).

19.2.2 Growth markets

- Battery anode materials – Used in lithium-ion batteries for electrical vehicles and other commercial & device energy storage applications (high thermal and electrically conductive capabilities);
- Expandable graphite – fire retardant cladding in buildings, paints, electronics and other coated products (assisting in high thermal resistance);
- Fuel cell bipolar plates and
- Graphene production.

19.2.2.1 Battery materials

19.2.2.1.1 Value chain

Natural flake graphite is becoming increasingly important in the European battery supply chain as the current supply of the material is mainly from China. To ensure healthy competitiveness and expanding the geographical supply chain additional sources and projects are emerging for the supply of natural graphite.

As previously mentioned in the value-added chain, the graphite concentrate is spheronized and purified to a minimum of 99.95% TGC. The purified material is then coated with a carbon mix to enhance the conductive properties of the anode feedstock. The anode producer will typically make a blend of this feedstock with synthetic graphite and other additives, designed to the requirements of the battery cell producer. This anode material is either sold as a mixed powder or applied directly on copper foils and used in conjunction with the cathode and electrolyte and other materials in the process of cell production. The EV automaker will then purchase the cells from the battery producer for inclusion in its vehicles.

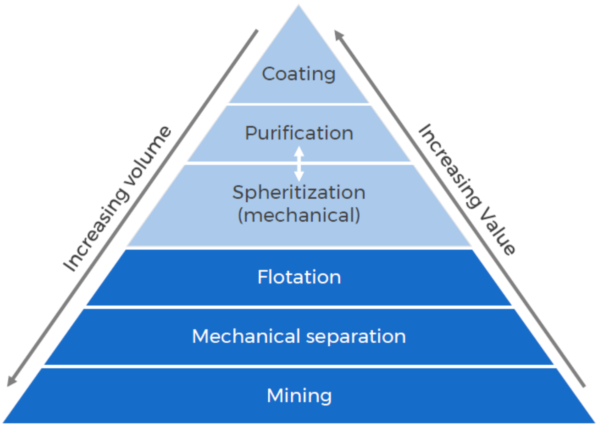


Figure 19-1: Value-add along the value chain (Source: Benchmark Mineral Intelligence, 2021)

19.3 Battery Market Demand

19.3.1 Current demand

Flake graphite demand by end-use/market in 2020. The Li-ion battery market currently accounts for around a quarter (202,617 tonnes) of the natural flake demand.

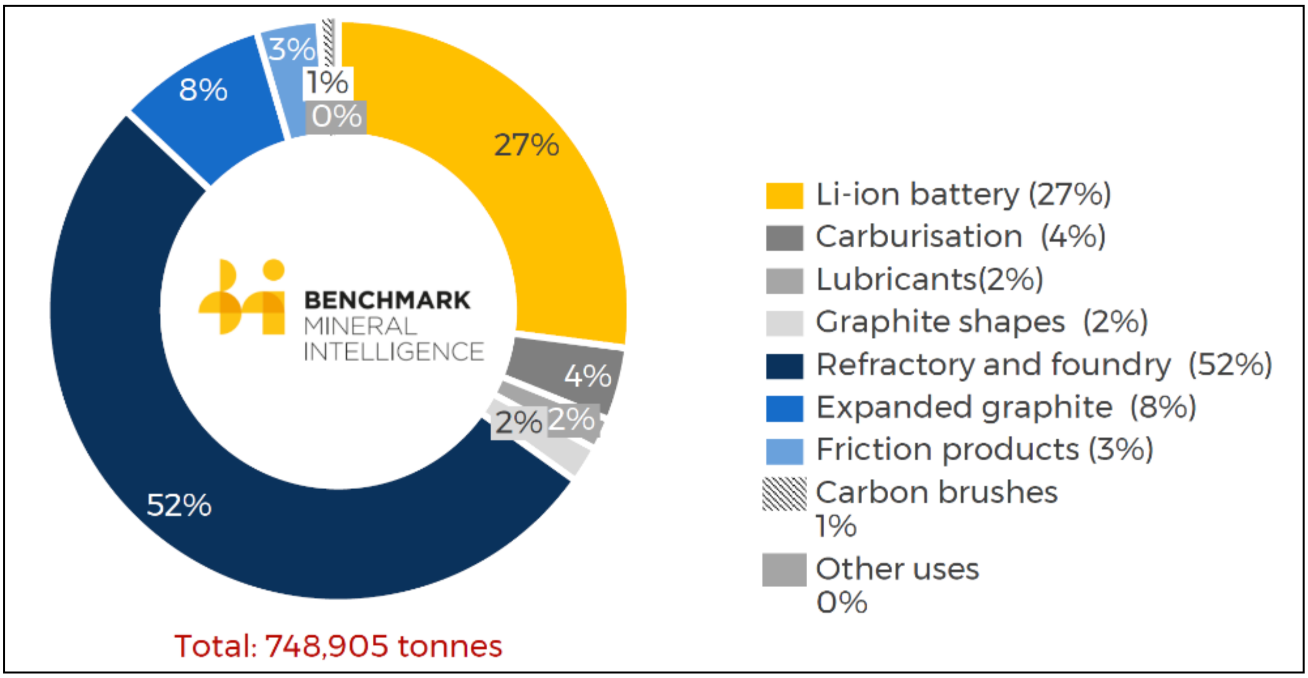


Figure 19-2: Flake graphite demand by end-use/market 2020, (Benchmark Mineral Intelligence, 2021)

19.3.2 Future battery market demand

19.3.2.1 Demand drivers

The end-use categories leading demand growth for batteries are transportation, energy storage systems (ESS) and portable devices.

- Transportation – Commercial, public and private electric vehicles uptake is increasing rapidly;
- ESS – Spurred by the growth of renewable technology rollouts, environmental public pressure, EU and Global agendas; and
- Portables – Smaller segment of the market, yet still driven by consumer innovations moving towards smarter interconnected systems and faster connectivity.

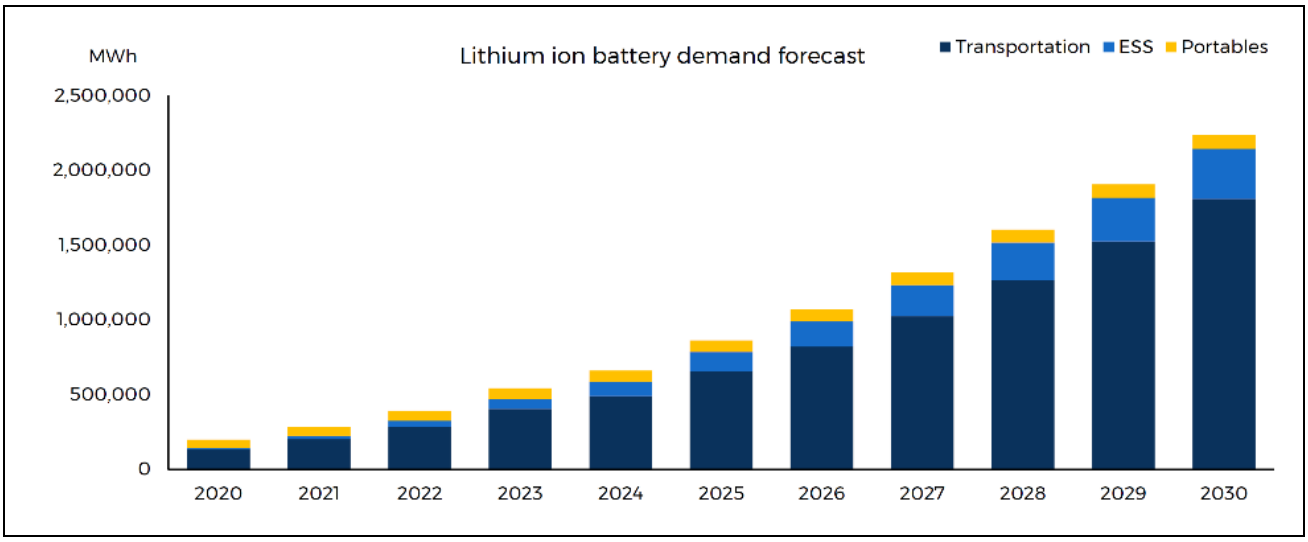


Figure 19-3: Lithium-ion battery demand forecast (Mwh), (Benchmark Mineral Intelligence, 2021)

The demand for these end-use categories and required associated materials are in turn driven by a number of macro factors:

- Environmental – As nations around the world commit to decarbonize political will to support low carbon footprint technologies. The production in the anode supply chain is under pressure to increasingly develop environmentally friendly purification methods, challenging a large portion of the producers in China not controlled by western environmental standards;
- Legislative – The EV demand is in turn driven by legislative pressure on internal combustion engines, creating a swift new demand for battery materials;
- Employment – Creating local jobs and indirect community development;
- Trade: Increased trade between EU member states including opening up other international markets with European supplied products;
- Security – Strategies in the form of European sourced critical minerals for security and defence technology; and
- Dependency – Europe and the globe have experienced the bottle-neck consequential risks of full reliance on one superpower nation during times of pandemics, trade embargos, trade wars, acts of aggression, ESG concerns and technological growth.

19.3.2.2 Global graphite demand

Natural flake graphite demand for anode applications is currently around 200,000 tonnes, expected to reach 1.1 million tonnes in the next five years, and 2.8 million tonnes in the next ten years resulting in an expected CAGR of over 17% over the next decade.

The global customers for flake graphite for battery use are Spherical Purified Graphite producers, currently all located in China, but there are some integrated operations planned outside of China. The tonnage needed for anode material is around 2.5 tonnes of flake graphite per tonne of anode. Currently the Spherical Graphite production capacity matches demand from the anode sector with close to all current capacity (>90%) and growth being based in China. This is however expected to change by 2025 when demand starts to outpace forecasted supply. Outside of China companies are targeting to launch operations to close this supply gap.

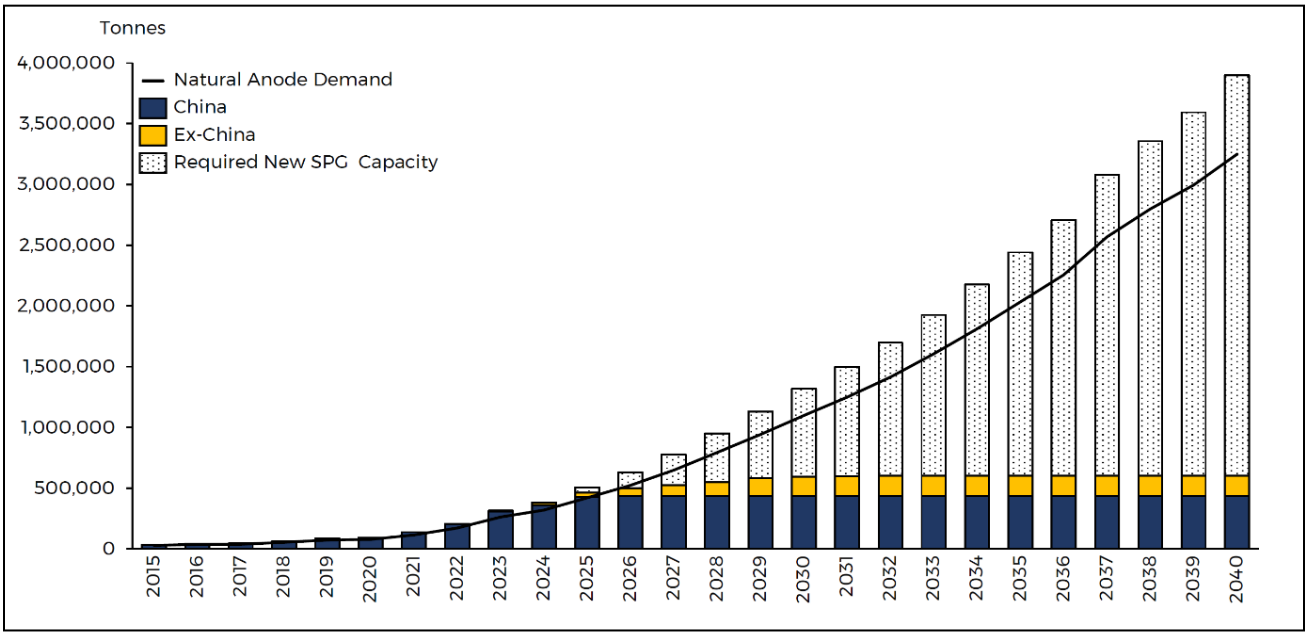


Figure 19-4: Spherical Graphite Capacity & Natural Anode Demand Forecast (tonnes) – 2015-2040, (Benchmark Mineral Intelligence, 2021)

19.3.2.3 EU demand

The forecasted demand in Europe is, at the time of writing in mid-July, a planned battery production capacity of over 1,000 GWh by 2030, which would need approximately 1,000 ktpa of anode material.

EU legislative demand:

- The European Union legislation is introducing a "battery passport initiative" to ensure responsible and sustainable mineral sourcing.
- EU members are each setting national goals for the end of sale or registration of vehicles powered by fossil fuels, with near term targets such as 2030 or 2040.
- EU aims to reduce emissions by 55% before 2030 and climate neutrality by 2050, supporting this by investing €1 trillion into the European Investment Plan.

19.4 Supply

19.4.1 Spherical graphite

19.4.1.1 Current supply

China dominates the natural graphite raw material supply, with Spherical Graphite supply currently 100% China dominated. Supported by developed flake feedstocks for the production of anode precursors at current demand levels. Production costs for graphite concentrate typically range from 400 to 700 USD/t depending on the region of production, which sets the majority of their production towards the low end of current cost curves for flake graphite.

China competes with their ready-built infrastructure of spheronizing facilities which operate in regions with subsidised energy costs, cheap use of harmful acid purification, meaning their opex for value-adding processes is very competitive.

The major downfall for the Chinese dominance in supply of Spherical Graphite is the varied upstream sourcing of concentrate. This affects product inconsistency, quality assurances and the loss of new international market confidence and growth. In contrast to the west, ESG concerns is not top of mind for Chinese producers, repeatedly sourcing concentrate from small scale operations operating under unsatisfactory western social & environmental standards.

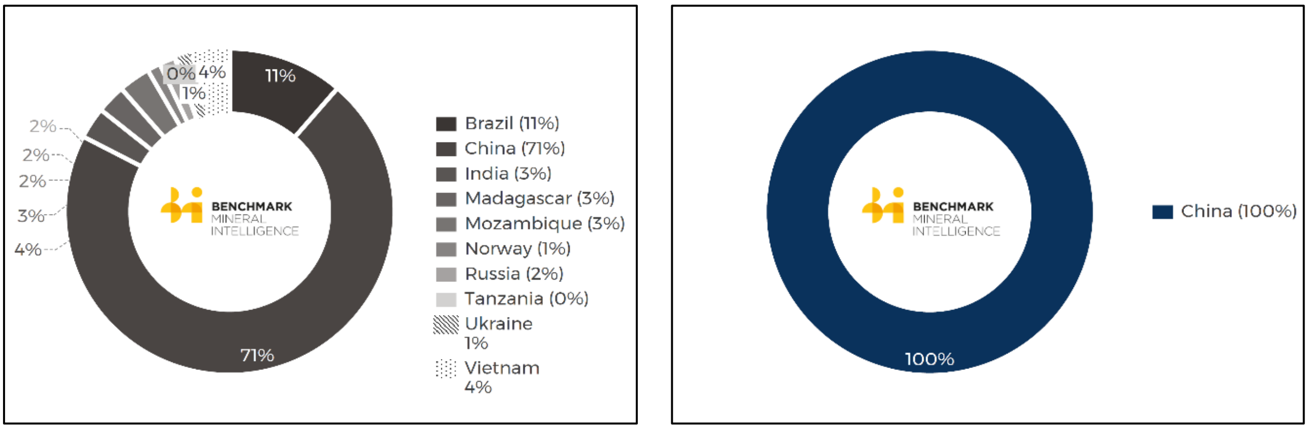


Figure 19-5: Graphite raw material supply 2020(left) and Spherical Graphite supply 2020(right), (Benchmark Mineral Intelligence, 2021)

According to Benchmark Mineral Intelligence, the capabilities for European sourced graphite products currently accounts for only, 3% of extraction/mining, 0% chemical processing, 0% anode production capacity, although plans for ramping up construction are in place.

19.4.1.2 Future supply

In global terms, the forecasted supply from China and Brazil will remain static, while realistic expansions along East Africa and Madagascar will prove to be a major supplier, mostly flake graphite concentrate. North America and Europe are commencing with planned projects, both intending to channel focus towards the value-adding downstream processes into their projects.

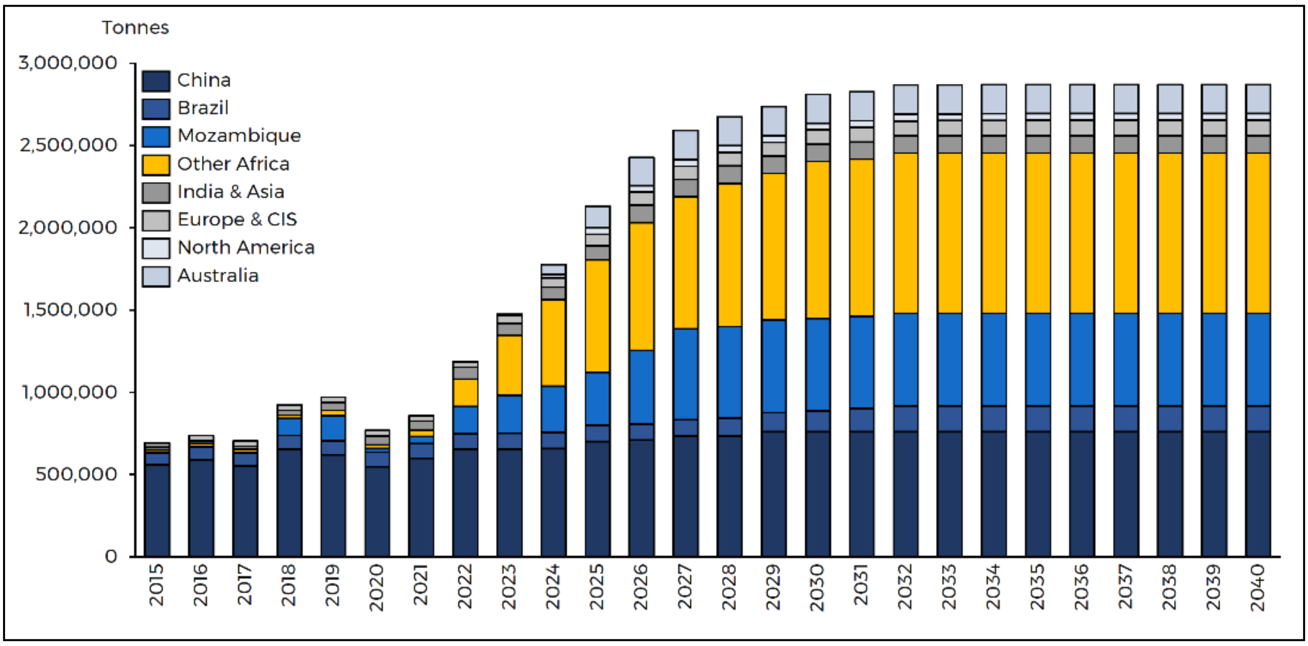


Figure 19-6: Forecasted Flake Graphite supply by region (tonnes), (Benchmark Mineral Intelligence, 2021)

Benchmark Mineral Intelligence has indicated that there is a combination guiding the future supply: Increasing levels of growth in the lithium-ion batteries market coupled with the balancing out and encouraging growth of the global steel and oil industry. This recipe is resulting in an increased global demand for large flake natural graphite, thereby de-stocking inventories and driving demand i.e., pricing.

19.4.2 Micronized graphite

According to a marketing report on European micronized graphite markets, the consumption of natural micronized graphite in Europe is as shown in Table 19-1 below. The table is divided into two categories: products below 12 micron and those above 12 micron. This is also a rough distinction for products usually being micronized with impact mills (>12 micron) and material usually being micronized with jet mills (<12 micron).

Additionally, the data is divided according to the carbon content (C). Grades with a carbon content below 96% do not need purified feed materials, whereas grades requiring a carbon content above 96% are usually produced by using chemically or thermally purified feed materials.

Table 19-1: Consumption of micronized carbon in Europe (Market Information on Micronized Graphite from the Woxna Mine, 2021)

Particle Size		d50 ≥ 12 micron	d50 < 12 micron	Sum
Mill type		Impact mill	Jet mill	
Carbon Content	%	85 -95	85 -95	
Consumption	kilo-tonnes	9	10	19

The total consumption of micronized graphite with carbon content between 85 and 95% in Europe reaches approximately 19,000 tonnes. The majority of this material is produced in Europeprimarily due to the high quality requirements and a closer customer-supplier relationship.

19.5 Pricing

19.5.1 Natural flake graphite concentrate 94-95% C

The Company strategy does not include the supply of flake graphite concentrate to the market, however prices are provided below to assist in the understanding of the benefits for value-added materials. The following prices are for the various concentrate mesh sizes Freight on Board (FoB) to China for 2021:

- USD 1,431(+50 mesh)
- USD 892 (+80 mesh)
- USD 654 (+100 mesh)
- USD 577 (-100 mesh)

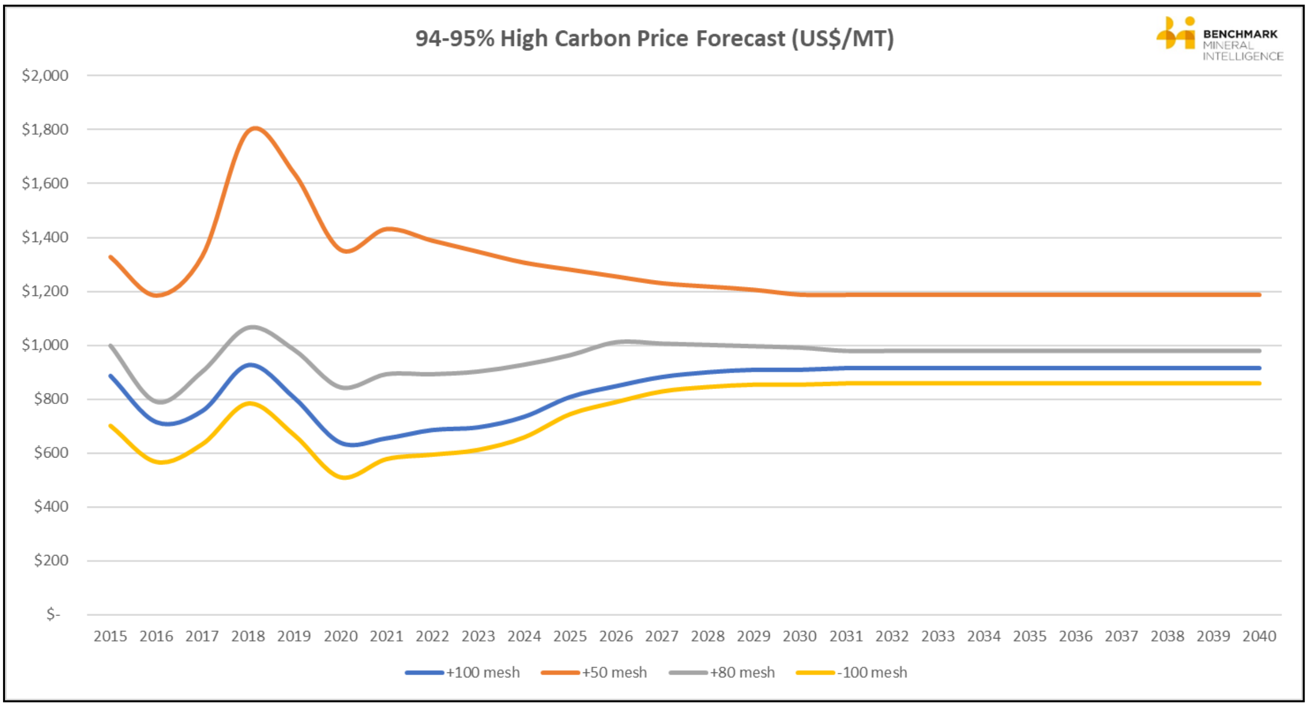


Figure 19-7: 94-95% high carbon price forecast (USD), (Benchmark Mineral Intelligence, 2021)

19.5.2 Spherical Purified Graphite (SPG)

As the majority of anode producers are based in China, the pricing is normally set between a small number of Chinese suppliers (around 30) and buyers.

A key component in Spherical Graphite pricing is the influence of the micron sizing which dictates the energy density constraints for the battery anode material. Larger micron sizes (>25 micron) typically prohibit use into higher value EV applications. There is a premium placed on smaller micron grades (<25 micron) that are able to show and maintain their structural integrity throughout the life of application. The smaller size allows greater energy density and offsets some of the higher costs required to achieve these properties without the need of using other feedstocks.

Benchmark Mineral Intelligence experiences the current spread within size fractions between minimum and maximum prices seen in the market to be around USD 500, and this forecast is to remain in future. Current conservative pricing averages for 2021 stand at:

- USD 3,650 (10 micron)
- USD 2,700 (15 micron)

- USD 1,779 (micronized fines >99.9%)

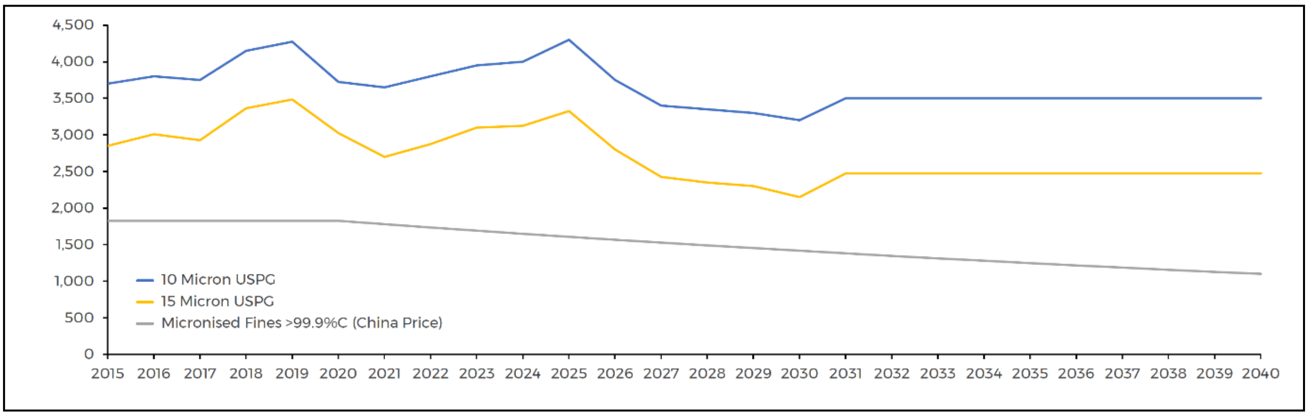


Figure 19-8: USPG and Micronized Fine price forecast (USD), (Benchmark Mineral Intelligence, 2021)

19.5.3 Coated Spherical Purified Graphite (CSPG)

CSPG material is rarely traded in the open market with the majority of material is consumed internally at the anode producer site within their own scope of variable specifics such as cell chemistry, electrolyte, cell form, quantity and customer requirements.

Contractual agreements are long term for materials based on highly integrated models between seller and buyer. As such there is a wide range in prices for this material, application dependent. The highest being for military and space applications, followed by Tier 1 cell producers, then consumer goods and lower end EV. It is important to note that qualification for material approval on the higher-level pricing is only achieved by a Tier 1 Coated Spherical Purified Graphite anode producer.

Benchmark Mineral Intelligence shows there is currently a larger spread for CSPG between minimum and maximum prices seen in the market to be around 5,000 USD/t. Current conservative pricing average for 2021 stands at 9,500 USD/t (Tier 1 EV)

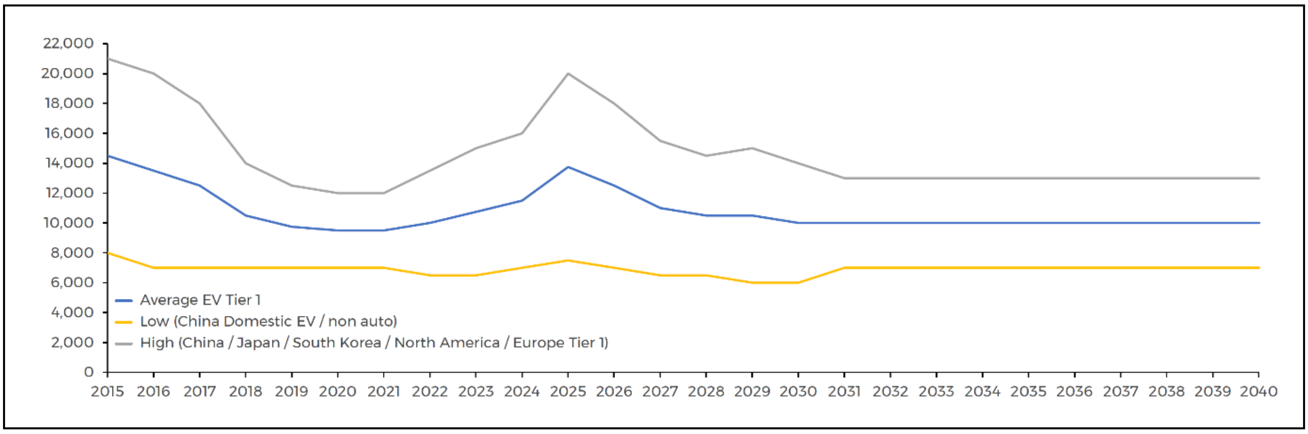


Figure 19-9: CSPG price forecast (USD), (Benchmark Mineral Intelligence, 2021)

19.5.4 Spheronized purified graphite pricing forecast overview:

Table 19-2: Current and forecasted pricing for uncoated and coated Spherical Graphite 2021-2040, (Pricing extracted from Benchmark Mineral Intelligence, 2021)

	2021 (USD/t)	2025 (USD/t)	2030 (USD/t)	2035 (USD/t)	2040 (USD/t)
USPG (15 micron) - Average	2,700	3,325	2,150	2,475	2,475
USPG (10 micron) - Average	3,650	4,300	3,200	3,500	3,500
CSPG (EV Tier 1) - Average	9,500	13,750	10,000	10,000	10,000

19.5.5 Micronized graphite pricing

A leading European graphite consultancy conducted jet mill micronizing test work with Woxna feed material in 2020 [3]. It was shown, that with both the flotation plant concentrate and spheronized fines materials, a properly jet milled micronized graphite product could be produced.

The current pricing for micronized flake graphite was provided and is displayed in Table 19-3 below.

Table 19-3: Current pricing for micronized graphite <96% C (Market Information on Micronized Graphite from the Woxna Mine, 2021)

Particle Size		d50 ≥ 12 µm	d50 < 12 µm	Sum
Mill type		Impact mill	Jet mill	
Carbon Content		C < 96%	C < 96%	
min. price	USD/t	960	1,690	
max. price	USD/t	2,710	4,210	
average price	USD/t	1,440	2,530	
quantity	kt	9	10	**19**
total value	**M USD**	**13.0**	**25.3**	**38.3**

The total market value of micronized natural graphite with carbon content below 96% is above USD 38 million. The growth rates for these products have been strong over the past few decades.

All the Woxna micronized graphite is produced from the fines created as a by-product during spheronization and would be a 'jet milled d50 < 12 µm' product.

Testwork indicates that the carbon content of the woxna micronized graphite is 92.3% C. The annual woxna production of a 'jet milled d50 <12 µm' product is 6.6 kt. Given that 10 kt is currently consumed annually in Europe, the Woxna production may impact pricing. When considering these factors and the specialist consultant report, a price of 1,200 USD/t has been specified for the micronized graphite Woxna product. This value is considered to be conservative refering to the pricing in Table 19-3 for a 'jet milled d50 < 12 µm' product.

As the project progresses into the next stages of feasability study, it is recommended further discussions with potential customers are engaged with the aim of agreeing off-take contracts in support of, and to confirm, this pricing.

19.6 Market positioning

Leading Edge Material’s through its subsidiary Woxna Graphite AB in Sweden is strategically positioning itself as an European natural graphite anode material producer by taking advantage of introducing value-added graphite products. Adding a coated spherical purified graphite product offers the opportunity to supply EU battery cell production directly. Moreover, the Woxna CSPG offers an EU sustainably sourced product with less supply-chain risk.

The Woxna Graphite Anode project is strategically placed within the EU where the planned battery production capacity is expected to result in a dramatic increase in demand for graphite anode materials in the region with a strong preference for secure and sustainable supply. The European Commission is actively pursuing policy objectives to promote an increased self-reliance on raw materials that are required for the green transition and the Company is an active participant in industry alliances such as the European Raw Materials Alliance.

Woxna Graphite, thanks to its location in Sweden has access to low-cost and low carbon footprint hydropower allowing the company to choose a thermal purification process for the project that offers the advantage of a significantly reduced environmental footprint of its products compared with current global supply alternatives.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental studies

20.1.1 Environmental setting

The Project area is located in the municipality of Ovanåker (approximately 10 km) north-west of the town Edsbyn. Ovanåker is part of the County of Gävleborg.

The Project today includes an open-pit, TSF, waste rock piles, a clarification pond (CP) Uxatjärn, a process plant with an associated laboratory, an office building and a sleeping barrack. No mining activities are currently undertaken.

The immediate environs are hilly with a number of wetlands surrounding. The area is sparsely populated and the distance to the closest residential building is approximately 500-meters to the south-east of the open-pit and 600-meters to the west of the process plant. There are also a number of residential buildings along the connecting road to the mine, especially at the village Övre Ölmesbo. The residents here are affected by transportation to and from the mine and potentially traffic noise.

The Kringel deposit is classified as a “national interest” for mineral resources, see Figure 20-1. To the south of the open-pit is a “national interest” for nature conservation which partly overlaps the deposit. Within this national interest a number of wetlands are present. One of these wetlands (Östermyrorna) has been classified as being of very significant environmental value. The wetlands are judged to be sensitive to water loss (loss of surface water and lowering of the groundwater table). The latest inventories of vascular plants in 2019, indicate that conditions are becoming dryer in Östermyrorna.

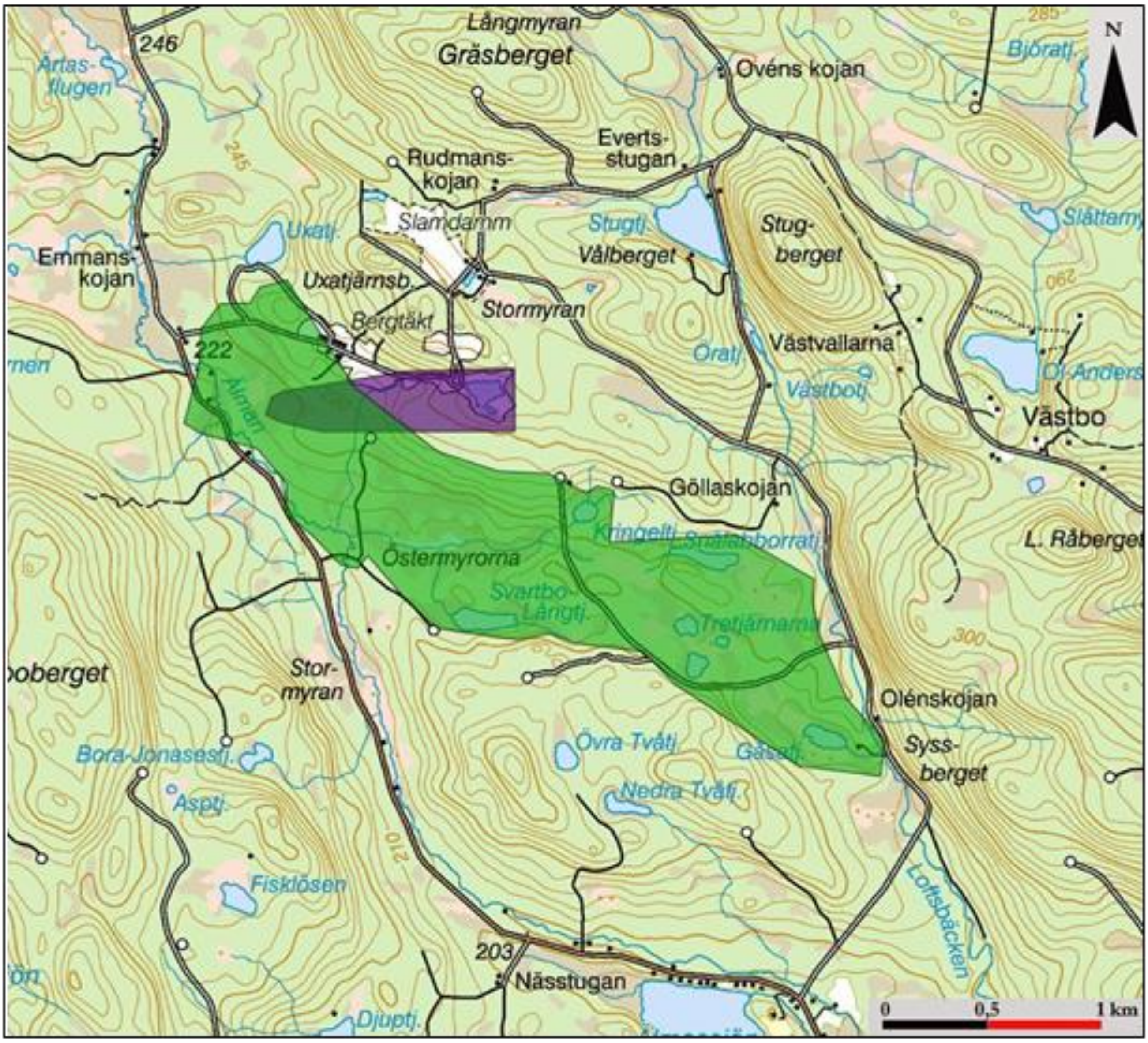


Figure 20-1: Two areas of national interest: nature conservation (green) and mineral resources (purple). As can be seen the national interests partly overlap.

Water from the mining area is mainly discharged into the watercourse (river) Älman which flow west of the site in a southern direction.

Älman has been classified in accordance with the Water Framework Directive (WFD). The current ecological status is" moderate" and the Environmental Quality Standards (EQS) has been classified as "good" in year 2021 since the Project does not affect the the quality factors for fish/fauna or the connectivity factors (fish migration barriers in Älvan. The chemical contamination status is "lower than good", mainly because of atmospheric deposition of mercury and brominated diphenylether.

Some drainage from the eastern part of the TSF discharges towards the watercourse Loftsbäcken to the southeast of the mining area. Loftsbäcken has also been classified in accordance with the WFD to have a "moderate" ecological status and "lower than good" chemical contamination status. The negative environmental impacts in Loftsbäcken are related to fish migration barriers and atmospheric deposition as in the case for Älman.

Älman and Loftsbäcken converge approximately 5 km to the south of the Project area. According to stream flow data from the Swedish Meteorological and Hydrological Institute (SMHI) the average yearly stream flows at this position are 0.3 m^3/s in Älman and 0.15 m^3/s in Loftsbäcken.

In the 2017 Land Use Plan, the Project area is designated for mining activities as shown in Figure 20-3. The plan also states that new buildings or other measures that may prevent the development of mining activities shall not be allowed, and that the exploitation of the deposit has to consider impacts on the nature interests.

20.1.2 Summary of environmental studies

Several environmental studies have been performed at the Project the last 15–20 years. Some of these studies have been performed as part of the monitoring programme (see Section 20.1.3) and these studies cover but are not limited to the following:

- Inventories of vascular plants and vascular cryptogams at nearby wetlands (the latest done in 2019).
- Inventory of nature values in the environs (the latest done in 2013).
- Investigations in watercourses nearby the Project area regarding bottom fauna, water chemistry, sediments (metal content) and fish (the latest done in 2012). The study in 2012 indicated a local biological impact in Älman downstream from the Project based on the results from the bottom fauna investigation.
- Measurement of groundwater levels at the wetland Östermyrorna (ongoing).
- In 2020 ÅF Pöyry (AFRY) prepared a new waste management plan on behalf of Woxna Graphite. The plan contains results from performed waste characterisation work (both tailings and waste rock) reports, metal concentrations, leachate tests (shake tests), acid base counting and humidity cell tests. Some of the tests were done some years ago e.g. the humid cell tests (in 2012) and the tests of the waste rock (in 2012). According to test data the tailings cannot be classified as inert waste but as non-hazardous. The waste rock comprises of two different types where one of them (Type A) is classified as inert waste and the other as non-hazardous waste (Type B) because of high sulfur concentrations. The overburden is classified as non-hazardous material. The main chemical issue with the tailings is elevated concentrations of barium, nickel, zinc, and sulphate together with a low pH-value.

In 2016 an environmental impact assessment (EIA) report was prepared by ÅF Pöyry as part of the application for an exploitation concession over the Project area. The EIA does not contain any calculation of the impact on groundwater levels from the open-pits (two pits are illustrated in the EIA).

20.1.3 Current monitoring/control programme

Woxna Graphite has developed an environmental monitoring programme related to the activities at the Project and the results of the latest April 2018 programme covers the following:

- Sampling and analyses of surface water in Älman and in Uxabäcken (the small creek at the outlet from the clarification pond). pH, conductivity, and visual control of colour/suspended substances are checked every month. Twice a year sampling and analyses of sulphate and metals are undertaken. If the analyses indicate discrepancy from normal, additional sampling points are included.
- Sampling and analyses of surface water in Loftsbäcken and the wetland Stormyran (south-east of the TSF). pH, conductivity, and visual control of colour/suspended substances are checked every month. Twice a year sampling and analyses of sulphate and metals are done. If the analyses indicate discrepancy from normal, additional sampling points are included.
- Measurement of groundwater level in two groundwater pipes located in the wetland Östermyrorna, once a month. Sampling and analyses of surface water in Östermyrorna twice a year (pH, alkalinity, conductivity, calcium, magnesium, sodium, potassium, sulphate, and chloride).
- Precipitation is measured daily during spring, summer, and autumn. The pH-value in the rainwater (or melted snow) is measured every week.

- Sampling and analyses of water from the clarification pond (CP). The pH-value is measured daily. Once a month, analyses of metals and suspended solids is done. Mineral oil and nitrate are analysed once a month, but only when mining activities takes place.
- Inventory of plants at seven wetland locations is done every third year.
- Investigation of bottom fauna at two locations in the watercourse Älman is done every third year. A fish survey is done at the same locations every third year.
- Noise measurement is performed at the closest residential building when the production restarts or in connection with the periodical inspection.
- Dust monitoring in air is planned to be performed every third year.

Periodical inspection by external expertise will be performed every third year. A first-time inspection will also be performed when the full production restarts.

20.1.4 Need for additional environmental studies

A review of performed investigations, inventories and studies has been made. It is obvious that some of the work performed needs to be updated since the level of detail is not sufficient for an EIA required for an application according to the Environmental Code. On the other hand, there are many studies performed that probably can be used as they are or used after minor changes. In the table below the need for additional studies is presented.

Table 20-1: Environmental Studies needed

Studied aspect	Comment
Site localisation study for waste rock dumps and TSF	Even if no major changes of the footprints are planned, a localisation study has to be prepared to prove that the chosen locations are the most suitable from an environmental perspective. The number of alternative locations that need to be assessed can be discussed. Often the County Administration can give advice on this matter during the consultation process, see Section 20.2.4.2.
Waste rock and tailings characterisation	Some of the data from the performed characterisation can be reused. However recent applications have shown that the authorities now demand characterisations done according to the CEN-standard, hence it is recommended to perform new characterisations using new tailings and drill core material. If a fraction of the tailings is separated prior to deposition on the TSF, the characterisation should represent the tailings that actually will be placed on the TSF as well as the separated part. An updated waste management plan would be needed for the application.
Hydrogeological investigation	Probably field tests including pump tests and flow logging need to be performed in order to understand the impact on the surrounding area close the open pits. The work should include installation of groundwater pipes in the surroundings for the purpose of monitoring groundwater levels. From collected data, as well as existing data, the influence area should be calculated and reported in the EIA report. A detailed and updated water balance covering the site is required.
Surface water chemistry	More elements and substances need to be analysed in surface waters because of the decided EQS for surface waters (water framework directive). Regarding metals, the samples must be filtered in accordance with the regulation from Swedish Agency for Marine and Water Management. The water balance covering the site would be required for this work.
Nature value inventory	The inventories performed so far are probably sufficient for a new application, but an update should be considered in case new information on nature values becomes present.

Studied aspect	Comment
Bottom fauna investigation, sediment sampling and fish inventory.	This is covered by the monitoring program and the data so far is most likely sufficient for a new application. However, it is not unlikely that the authorities would require an update.
Investigation of ground vibrations and noise	Ground vibration is most likely not an important issue at Kringel due to the distance to any residential buildings. The matter needs anyhow to be assessed in a future EIA report. Traffic noise from trucks on the connecting road to Kringel needs to be investigated, assessed, and reported in the EIA report.
Process and mine pit water treatment	If there are any difficulties in complying with the EQS-values, water treatment studies should be done in order to identify techniques needed to achieve an acceptable impact on the watercourses.
Mine Closure	An updated closure plan adapted for the final shape and layout of the facilities needs to be prepared. The plan will form a part of a new permit application and will include calculated closure costs as well as a bond.

Besides the work described in Table 20-1 above, also other studies/documents are needed to prepare a complete EIA report for an application according to the Environmental Code e.g. status report, waste management plan. The estimated costs for all work, additional studies, and reports, including the work in Table 20-1, are presented in Table 21-8 in Section 21.

20.1.5 Processing in the town of Edsbyn

Woxna Graphite plans to further beneficiate the flotation concentrate produced at the Woxna Concentrator in the VAP located in the town of Edsbyn that has been selected for the purpose of this PEA. An industrial building is available that was previously used for plywood fabrication. The decision to use this VAP facility in this PEA was motivated by existing power supply, existing industrial facilities and logistics. The building is located on the property Norra Edsbyn 10:75 close to the lake of Ullungen in the central parts of the town, see Figure 20-2. Because of the adjacent lake it should be possible to extract water from it for process/cooling use. According to stream flow data from SMHI, the average yearly outflow from the lake is 5,800 m^3/h.

Figure 20-2: Location of the industrial building in Edsbyn marked with a red ellipse.

The VAP facility includes an installed high-temperature furnace. Such a plant will probably not imply any significant environmental local impact, but an environmental permit will be needed for the operation. The permitting of the plant can be included in the same permit application as that for the mining operation. To include the Edsbyn plant in the permit application for the mine will reduce the overall cost for the permitting process.

20.2 Permitting

20.2.1 Environmental permitting

Woxna Graphite holds a permit according to the Environmental Protection Act granted by the Licensing Board for Environmental Protection (sv Koncessionsnämnden för miljöskydd) on 17 September 1992. The Environmental Protection Act is an old law that no longer exists, however the permit is still currently valid.

The permit gives Woxna Graphite the right to carry out mining operation including exploitation and processing of 100,000 tpa RoM annually. The permit includes 13 permit conditions. Below some of the most important permit conditions are summarised below:

- Condition 1: Unless otherwise provided for in this permit, the works shall be operated mainly in accordance with what the company stated or undertook in the application and in the remainder of the case.
- Condition 2: Process wastewater, open pit water and contaminated stormwater shall be discharged to the TSF.
- Condition 3: Target values regarding concentrations of contaminants in the water that is released from the CP to the watercourse Älman. The target values cover the sum of metals (Zn, Cu, Pb, Cr, Cd and As)

<0.5 mg/L, suspended solids <15 mg/L, nitrate-nitrogen <4 mg/L, mineral oil <0.5 mg/L and pH-value >6.5.

- Condition 6: Dust emissions from crushing and sieving equipment and other equipment shall as a target value, not exceed 20 mg/Nm^3.
- Condition 9: Noise limits at residential areas. Equivalent levels: 50 dB(A) daytime weekdays, 40 dB(A) night-time (22-07) and 45 dB(A) other time. Maximum noise levels night-time shall not exceed 55 dB(A).
- Condition 12: Closure. When the mining activities have ceased, affected areas shall be remediated. A remediation plan shall, in consultation with the supervising authority, be prepared. The Licensing Board gave the supervising Board the rights to issue detailed provisions on remediation.
- Condition 13: Financial security. The company shall provide financial security for fulfilment of the future remediation. The amount shall be approved by the supervising authority. [Comment: In 2019 the financial security to cover future remediation costs totalled 500,000 SEK]

Compliance with the permit conditions is reported in the annual environmental report.

Besides the granted permit above, the supervising authority (County Administration Board) has issued a number of decisions regarding the mining operations. In 2008 the authority decided that the water level in the open pit must not exceed 295.5 m and if this level is exceeded, the water shall be pumped to the TSF. In 2019 the authority decided on the dam safety classification. Other formal decisions and approvals have also been made by the supervising authority between 2008 and 2019, but these are mainly closed.

Woxna Graphite also holds a permit according to the Water Act that was granted by the Water Court (sv Vattendomstolen) on 15 October 1992. It should here be pointed out that the Water Act is an old law that no longer exists, but the permit is currently valid. The permit covers several rights that can be summarised as follows.

Permit to:

- Build a tailings storage facility (TSF) between Uxatjärn and Stugtjärn by construction of a downstream dam with bottom outlet and overflow drain and an upstream dam and a screen dam.
- Build a CP in Uxatjärn with a bottom outlet and overflow drain.
- Regulate the water level in the CP in accordance with certain conditions (see further below).
- Discharge process- and mine water to the TSF.
- Pump water (up to 90 m^3/h) from the CP to the process plant.
- Construct a pump station at the watercourse Älman.
- Pump water from Älman at a maximum rate of 45 m^3/h to be used as process water.

The permit conditions regulate operation of the CP and the pumping from the watercourse Älman. Regarding the CP, the water level can vary between +233.5 m and +231.4 m and a level scale shall be placed in the pond that clearly marks these levels. In 2013 the Environmental Court changed this condition and allowed the level to vary freely if the level +233.5 m is not exceeded.

Regarding Älman, the permit holder shall make notes of the running times for the pump and at occasions when the stream flow in Älman is low, the water abstraction is not allowed to cause any drying in the watercourse.

20.2.2 Mining concessions

Woxna Graphite holds an exploitation concession for the Kringel deposit that is valid until 3 November 2041. The concession area is shown in Figure 20-3 below.

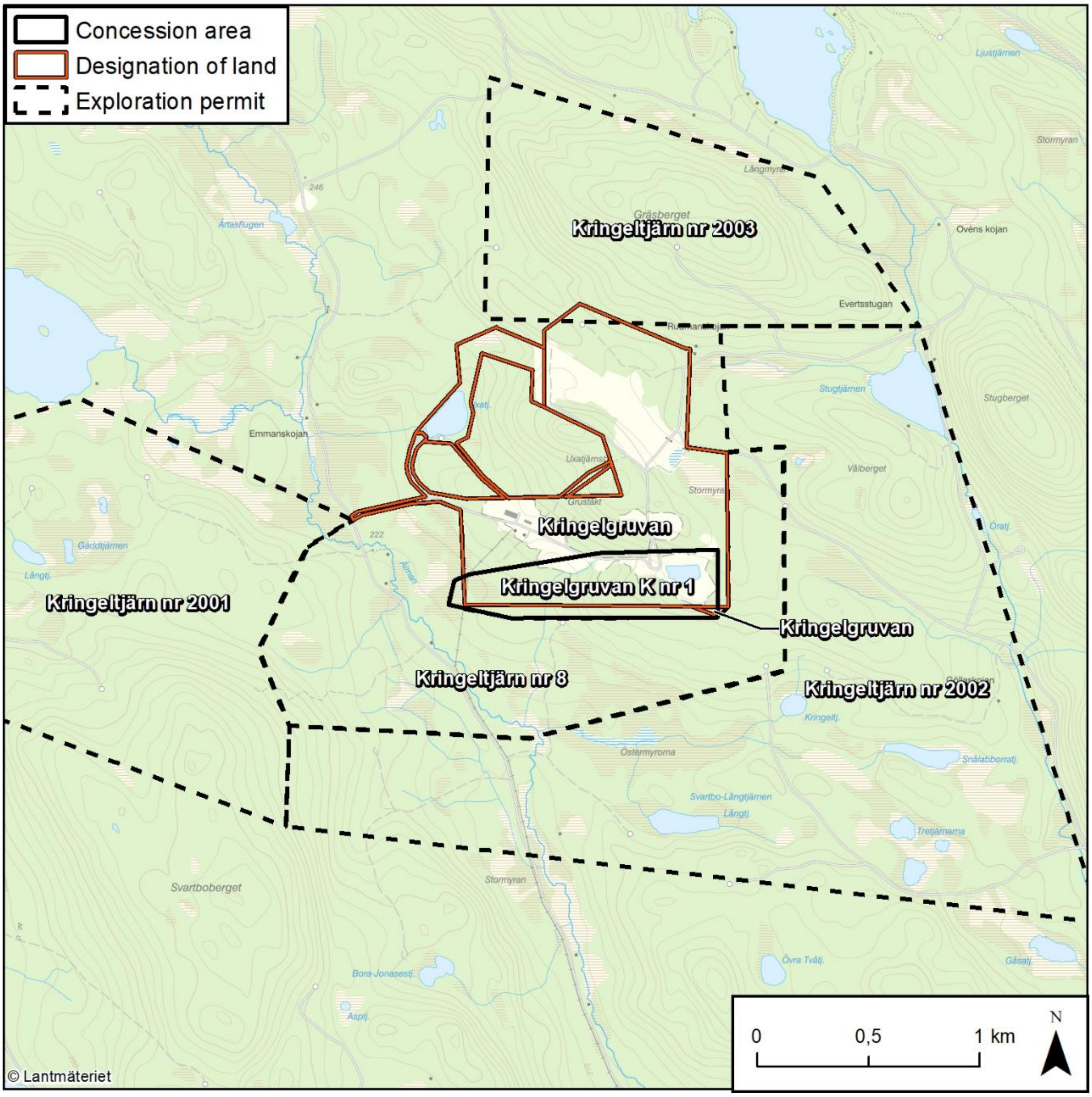


Figure 20-3: Exploitation Concession, exploration licence, and land allocation (Golder 2021)

20.2.3 Requirement for new permits

The two existing permits according to the Environmental Protection Act and the Water Act were issued under old and superseded legislation and cannot be altered or changed. The new Project design parameters include increased mining production and an extended open-pit compared with the current permitted level (100,000-tpa), which would require a new permit in accordance with the current Environmental Code. Since the current permits cannot be altered or changed under now defunct legislation, a new environmental permit would be preferred by the environmental authorities once full-scale mining activities are restarted. In Section 20.2.4.2 the permitting process is summarised.

20.2.4 Regulatory framework

20.2.4.1 General

The Swedish Environmental Code (SFS 1998:808) provides the legal environmental framework for environmental matters. The Environmental Code comprises 33 different chapters dealing with different aspects as provisions concerning management of land and water areas, environmental quality standards, environmental impact statements, protection of nature and species, provisions concerning environmentally hazardous activities and health protection, contaminated land, water operations, chemical products, waste and producer responsibility, consideration of cases and matters, supervisions and charges, and penalties.

As part of the framework there are also a number of ordinances issued under the Environmental Code. Such ordinances mainly specify or clarify the rules or objectives stipulated in the Environmental Code.

The Swedish Environmental Protection Agency (SEPA) and the Swedish Agency for Marine and Water Management (SAMWM) publishes with the support of the Environmental Code regulations and guidelines on specific matters like industrial and construction noise, waste management and treatment, protection of species and key biotopes, environmental reporting, control/monitoring activities and classification - EQS for surface waters. Corresponding EQS for groundwater are published by the Geological Survey of Sweden. Regulations related to handling and storage of flammable gases and liquids are published by the Swedish Civil Contingencies Agency (SCCA).

20.2.4.2 Environmental permitting process

The first step in the permitting process is consultation with the County Administration Board (CAB), the Local Environmental Department (LED) and potentially affected private individuals (Chapter 6 in the Environmental Code). This consultation is most often two separate meetings, one with the CAB and LED and one with the local stakeholders. Prior to the consultation the applicant has to prepare a consultation document that covers planned localizations for the activities, the extent of the planned operations, preliminary designs and the foreseen environmental impacts from all activities. If the planned activities are expected to impose significant impacts, consultation shall also be carried out with other national authorities, municipalities, environmental organisations (NGO's) and the public. The purpose of the consultation is to obtain viewpoints to consider in the EIA report. The CAB has a key role to guide the applicant regarding the extent of the EIA.

After finalising the EIA report and the technical description of the activities and facilities, a formal permit application (legal) is prepared. All reports, drawings and documents are thereafter submitted to the permitting authority (Environmental Court or the Permit office at the CAB). The actual EIA is just a report with several appendices, usually with the following main sections:

1) Administrative information
2) Introduction including purpose and any limitations
3) Legislation
4) Planning matters
5) The nature and extent of the operations
6) The no-go option
7) Site and area description
8) Protected areas and areas of national interest (land and water)
9) Cultural and socio-economic conditions
10) Effects of pending operations (impact assessment)
11) Rehabilitation
12) Best available technologies
13) References
14) Attachments and appendices

An application also includes a detailed technical description, drawings, lay-out etc.

In the next step the permitting authority sends the full application to the consultation bodies for viewpoints if supplementary information or data is required to regard the application as complete. The public gets the same opportunity to respond as the consultation bodies. Requests for additional information and/or clarification are thereafter sent to the applicant who gives the opportunity to submit additional information to the permitting authority.

When the permitting authority has judged the permit application to be complete, announcement of the application and the EIA report is done. After announcement consultation bodies, experts, organizations, and private individuals are asked to provide their opinion on the complete application within a certain time. Opinions are sent by the permitting authority to the applicant who gets the opportunity to response to the provided opinions. If the Environmental Court is the permitting authority, the Court decides upon a date for court hearings after consultation with the applicant. The hearings normally take place in local premises in the area of the project site (c-3-5 days), making it possible for local stakeholders to attend and giving the Court a possibility to perform a site visit together with the applicant, consultation bodies, stakeholders and others. At the end of the hearings the Court informs when a decision on the application can be expected. The decision made by the Court can be appealed to the Environmental Court of Appeal and a decision made by the CAB can be appealed to the Environmental Court. The actual process is summarized in Figure 20-4.

Figure 20-4: Swedish Permitting System

*Should be conducted as early in the process as needed for the company to be able to take opinions/information into account when planning the operations localization, potential process design choices etc. Is the basis for CAB's decision regarding significant environmental impact.

**If considered a significant environmental impact a specific EIA should be conducted.

***A specific EIA usually require more underlying investigations of surrounding environment (for example endangered/threatened species, ecological and chemical status in water recipients etc.).

****Only for District court. Usually not conducted for applications handled by CAB.

20.3 Waste management

20.3.1 Background and objectives

Woxna Graphite is planning to resume and expand mining operations at the Kringel deposit. As part of the multidisciplinary team undertaking the PEA, Golder has been requested by LEM to undertake the conceptual design for the LoP TSF for the Project.

The overall scope of work of the present study is to undertake the conceptual design of the TSF over the 19-years planned LoP, including:

- Conceptual study, including maps plans and key conclusion of the technical work, highlighting one option for future tailings management to purse during the next stage of project development;
- recommendations on Roadmap to restart, highlighting key items required for the next phases of the development (field investigations, test work, preparatory work if necessary);
- Risk and opportunities analysis; and
- Cost estimate at conceptual level (±50%) for the project development, construction, operation and closure of the tailings facilities, clarification pond and appurtenant structures.

Previous operations have used a single tailings facility. For the PEA design, there are separate tailings facilities, a facility for the non-acid generating (NAG) tailings, and a facility for the potential-acid generating (PAG) tailings.

20.3.2 Description of the Project TSF

A review of all the available existing data about the TSF and the CP has been undertaken. Among all the information available, Golder reviewed the following:

- historical documentation (permit application,);
- topographical data;
- hydrogeological, hydrological, and geological data;
- previous technical tailings design, studies and construction documents including:
  - site investigation data;
  - tailings characterisation data; and
  - stability & Seepage analysis.
- current plans for processing, mining, overall site water management;

In addition to the desktop study review, a site visit was undertaken by a Golder team comprising a senior Tailings Engineer, a senior Environmental Engineer and a senior Hydrogeologist over two days at the start of the project to check the current status of the facilities and discuss with the owners' team on the previous operation as well as the current plans going forward.

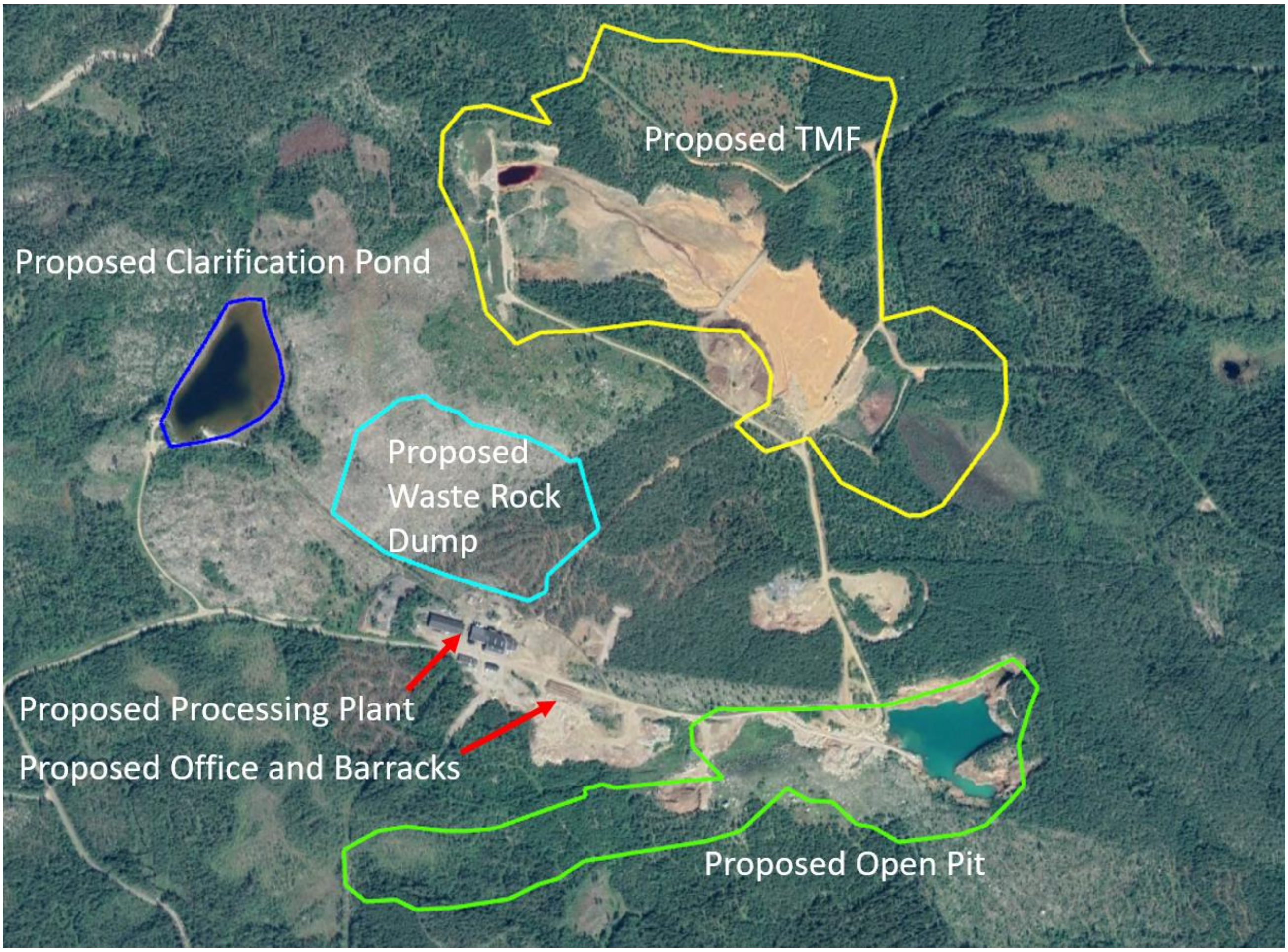


Figure 20-5: Overview of the Proposed Site Layout (Golder 2021)

20.3.3 Existing TSF description

The current TSF consists of the three dams, Northwest (NW) Dam, Southwest (SW) Dam, and the East (E) Dam, shown on Figure 20-6 below.

Figure 20-6: Overview of the tailings storage facility showing the three dams (TCS 2014 [11]).

The existing TSF was constructed prior and during the previous mine operation under earlier owners and has been inactive since 2001. The three dams provide containment for the tailings which were deposited from the south abutment of the E Dam, with the resulting tailings beach sloping towards the west dams and supernatant flowing also towards the NW Dam. A spillway is currently located at the NW Damfor discharge of water to the clarification pond, see Figure 20-5 and Figure 20-6.

During the latter stage of the mine operation, an intermediate Berm was constructed to deposit tailings from approximately the middle of the tailings deposition area.

The current volume of tailings in the TSF is estimated to be 1.0 Mm^3 and the total area is approximately 10 ha [12]. The depth of the tailings within the TSF varies between approximately 0.5 and 17 m and the average depth estimated to be 5 m [13]. The parameters of these three dams are summarised in Table 20-2.

Table 20-2: Summary of the three existing TSFs ([12]; [11]).

Description		Unit	SW Dam	NW Dam	East Dam
Crest		m	+275.6	+271.5	+276.4
Water pond level – normal case		m	+269.5	+269.5	
Water pond level of water level – extreme case		m	n/a	+207.5	
Stability Factor of Safety -	normal case	-	2.0	1.7	n/a*
	extreme case	-	1.4	1.4	n/a*
	Oversized leakage	-	1.9	1.6	n/a*
Elevation of tailings		m	none	none	+274

Description	Unit	SW Dam	NW Dam	East Dam
Type of construction of Embankment Dams		Rockfill starter dam with moraine core	Rockfill starter dam with moraine core	Homogeneous earth-fill starter dam with central moraine core
Downstream slope	V:H	min 1:3	min 1:3	approx. 1:1.25
Upstream slope	V:H	min 1:2	n/a*	n/a*

*n/a = not available

The E Dam is constructed with a central moraine core, and moraine support fill on the upstream and downstream sides. The E Dam has a crest height of +276.4 m, with tailings placed to a level of +274 m against the upstream slope.

The NW and SW Dams are constructed with a central low permeability moraine core, and crushed shale rockfill support fill on the upstream and downstream sides (according to the DTU manual [14]). It is understood that no filters are present between the moraine core and rockfill to prevent internal erosion of the moraine core. The tailings pond is in direct contact with the NW Dam. The pond is not in contact with the SW Dam as the upstream toe of the SW Dam is at a higher elevation than the typical pond level.

Significant seepage on the downstream side of existing embankments has been reported.

Pictures of the conditions of the different facilities are shown in APPENDIX A. Significant amounts of rehabilitation and reinforcement work are necessary to improve the conditions of the embankments prior to restart and expansion of the mine.

20.3.4 Site conditions

20.3.4.1 Topography

The topography nearby the mining site is relatively undulating with several wetlands surrounding the premises. The area is sparsely populated, and the closest residential building is approximately 500 meters to the south-east of the open pit and 600 meters to the west of the Woxna Concentrator.

The mining area is at an elevation of about +300 mamsl. Data on ground-surface topography are available from a topographical survey that was performed by Lantmäteriet in 2012 (Golder 2013), see Figure 20-7.

As shown on Figure 20-7, the elevation of the tailings surface contained in TSF is 270–276 m.

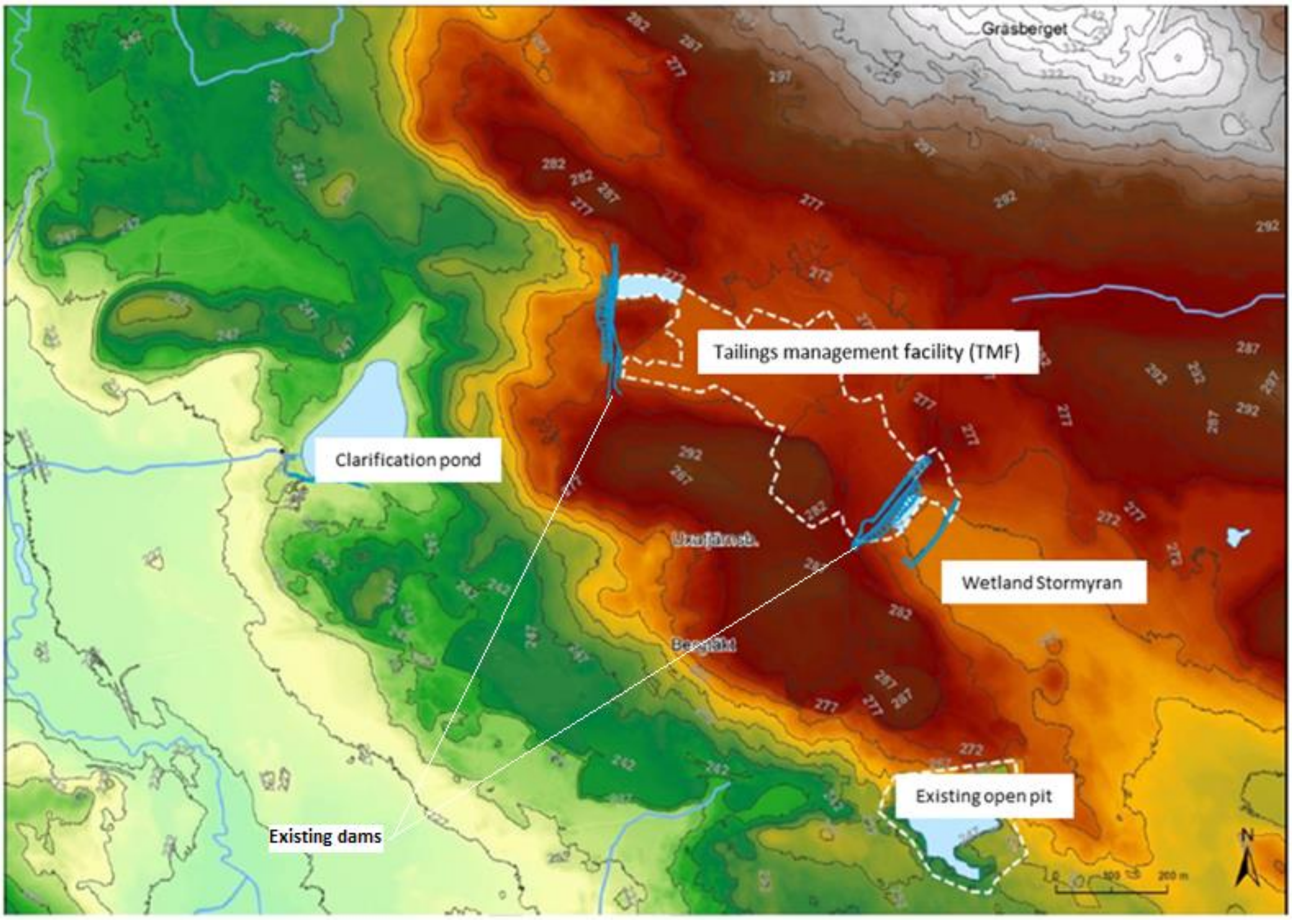


Figure 20-7: 2012 Ground-surface topography (Golder 2013)

The open-pit and the CP are located in the catchment area of the Älmån river, whereas the TSF is located within the catchment area of the Loftsbäcken river.

Figure 20-8 shows the boundaries of the catchment areas of facilities and structures at Kringel according to the delineations of Golder (2013). The calculated catchment areas are presented in Table 20-3.

This scoping study has been undertaken on the basis that two tailings streams will be produced from the plant. One tailings stream is called the Non-acid Generating (NAG), while the other is the Potential Acid Generating (PAG). The NAG tailings will represent 90% of the tailings production and will be deposited on the existing tailings area, while the PAG tailings represent 10% of the total tailings production and will be deposited in a separate TSF built purposely.

Table 20-3: Catchment areas of the planned open pit, TSFs, and CP (Golder 2013)

Object	Area (m^2)	Catchment area (m^2)
Open pit (including the existing open pit)*	155,000	512,500
Tailings storage facility (TSF NAG and TSF PAG)	312,500	995,000
Clarification pond (Uxatjärn)	25,000	310,000

**Calculated based on Zenito's design (2021)*

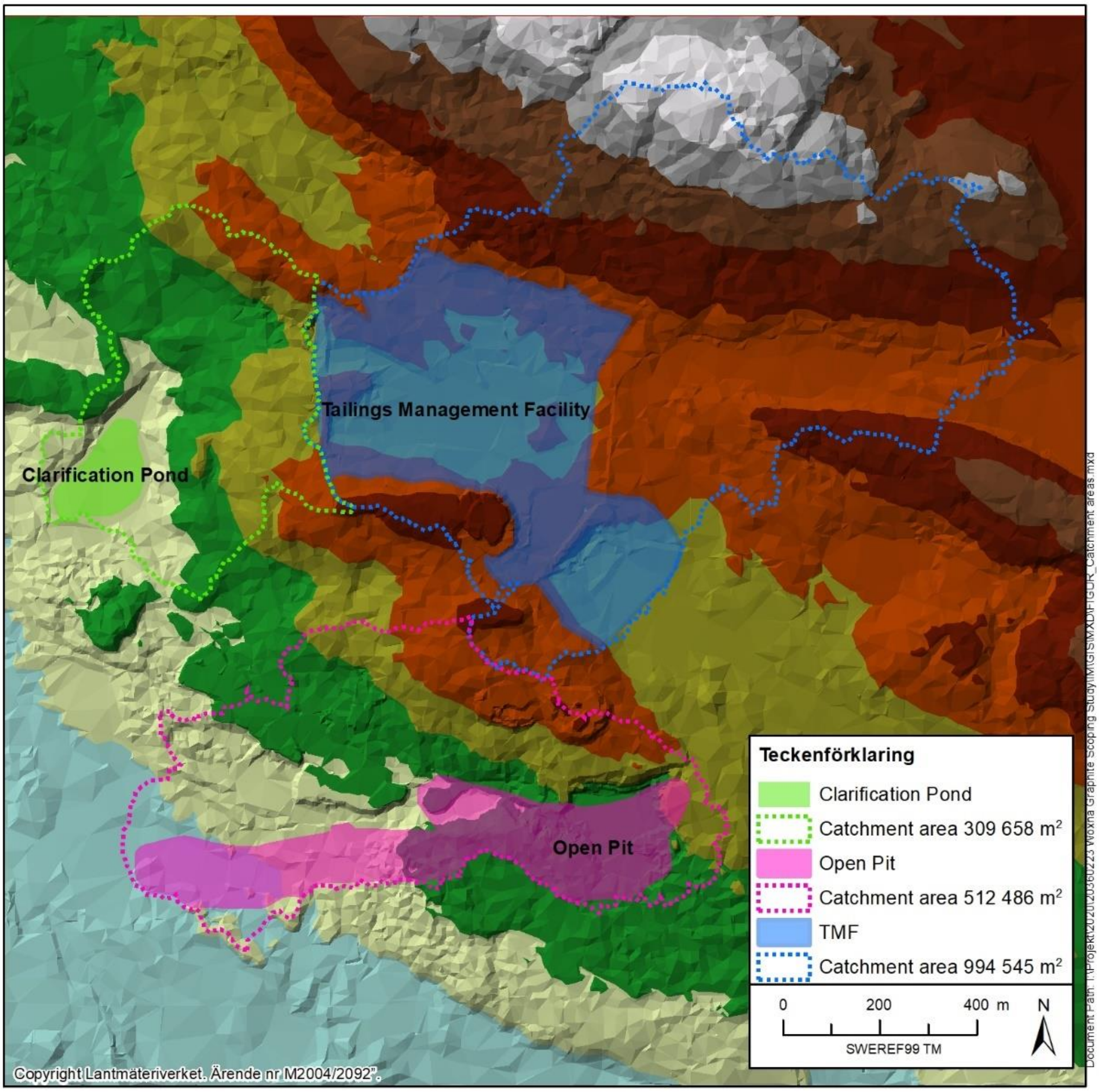


Figure 20-8: Catchment areas of the planned NAG and PAG TMFs, clarification pond and open pit (Golder 2013).

20.3.4.2 Geology and hydrogeology

20.3.4.2.1 Regolith depth, types, and stratigraphy

Glacial till is the dominant type of regolith in the investigated area. West of the Project there is a hilly stretch of glacial till surrounding a ridge that extends in the NNW-SSE direction about 1 km from the western side of the TSF.

The moraine generally has a high content of silt, sand and gravel, and low/missing clay content. There are a large amount of boulders present on a mine site area both, on and below ground surface; the proportion of boulders in the regolith is about 20–40%. The moraine depth around the existing open pit varies between 1 and 4 m, and an average depth across the mine site of approximately 2.5 m (Mattsson, 2011).

There are number of wetlands in the surrounding area, one is Stormyran which is located on the eastern site of the existing TSF, see Figure 20-7 and Figure 20-9.

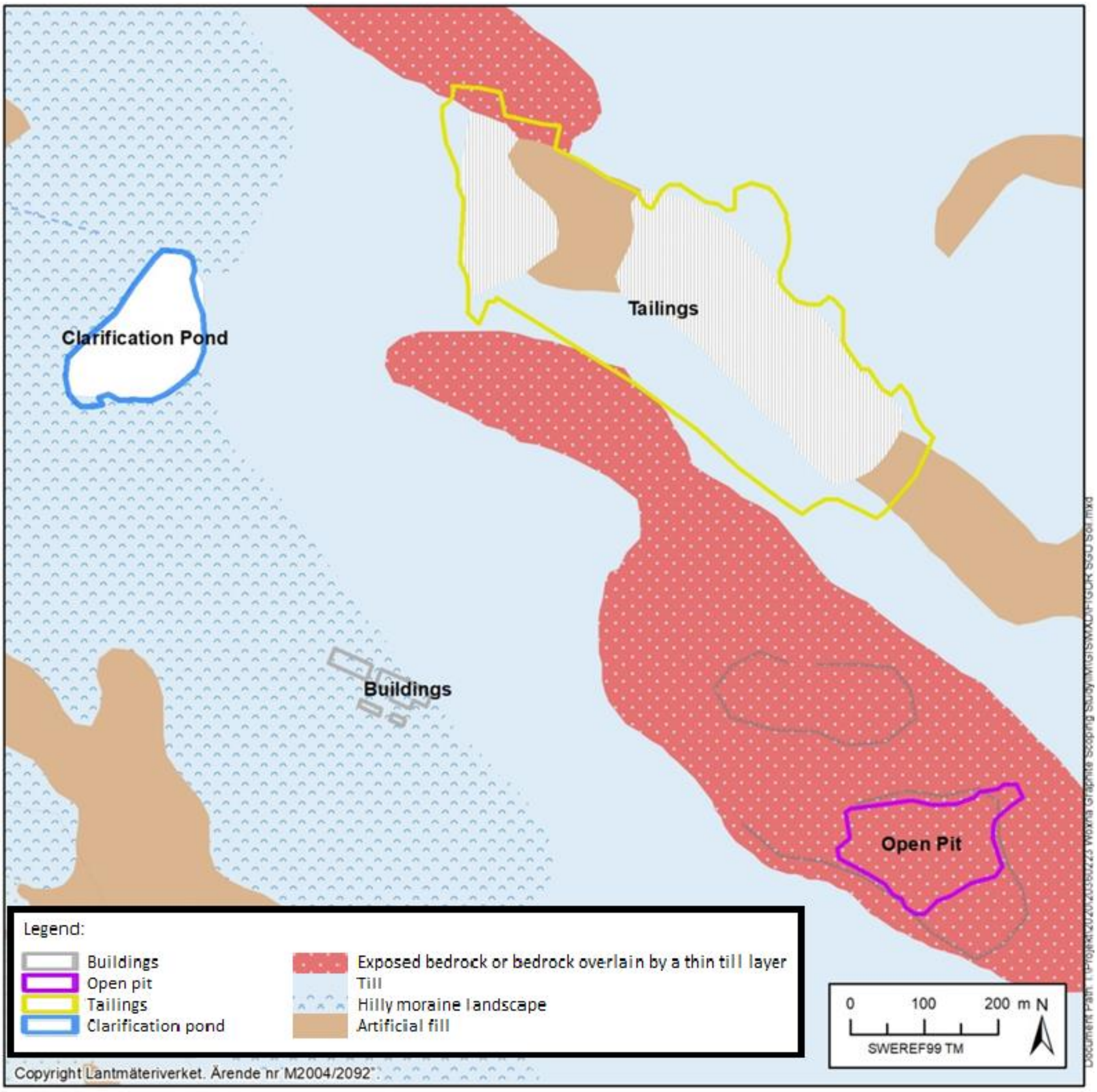


Figure 20-9: Quaternary geology of the investigated area (Golder 2013).

The moraine in the area has been investigated in several trial pits by Sweco GeoLab (2012). In two of these (PG33 and PG34), the hydraulic conductivity was measured in the laboratory to be between $2x10^{-8}$ and $1x10^{-7}$ m/s. The hydraulic conductivity in the upper part of the subsoil and in the existing dams is estimated at $6x10^{-8}$ m/s, estimated from these test values.

20.3.4.3 Meteorology

The meteorological data, e.g. air temperature, precipitation, wind speed and direction and potential evaporation, have been used for the water balance model in the Kringel site. These data are overall obtained from two SMHI (2020) meteorological stations:

- Edsbyn meteorological station (SMHI id 115230) which was in operation from 1941-01-01 to 1995-12-31; and
- Edsbyn A meteorological station (SMHI id 115220), which is in operation from 1995-10-01 to the present.

Further details regarding the meteorological data can be found in the report summarizing the water balance modelling (Golder, 2021).

20.3.5 Basis of design and design criteria

The TSF design criteria are based on the Swedish dam safety guidelines for the mining industry, GruvRidas (SveMin 2012). Where required, international dam safety guidelines have also been referred to, to meet the best available practices for mine storage facilities: These include:

- Canadian Dam Association (CDA) and the Mining Association of Canada (MAC) guidelines;
- Global Industry Standard on Tailings Management (GISTM); and
- European Commission (EC) Reference Document on Best Available Techniques for Management of Tailings and Waste-Rock in Mining Activities.

20.3.5.1 Embankment construction methods

There are three main construction methods for tailings dams (excluding that related to dry stacking) which are: upstream; downstream, and centreline. A simplified diagram is shown in Figure 20-10. The three types of embankment have different cross-sectional areas and will require different quantities of engineered fill, which is assessed in subsequent sections.

- Upstream Construction, which has the lowest construction volume, relies on the strength of previously deposited tailings, and is limited by allowable rates of rise.
- Centreline construction also relies on the tailings strength, but to a lesser degree, can generally have a higher rate of rise.
- Downstream construction does not rely on the tailings strength, are generally the more robust solution but also requiring the larger volume of construction material.

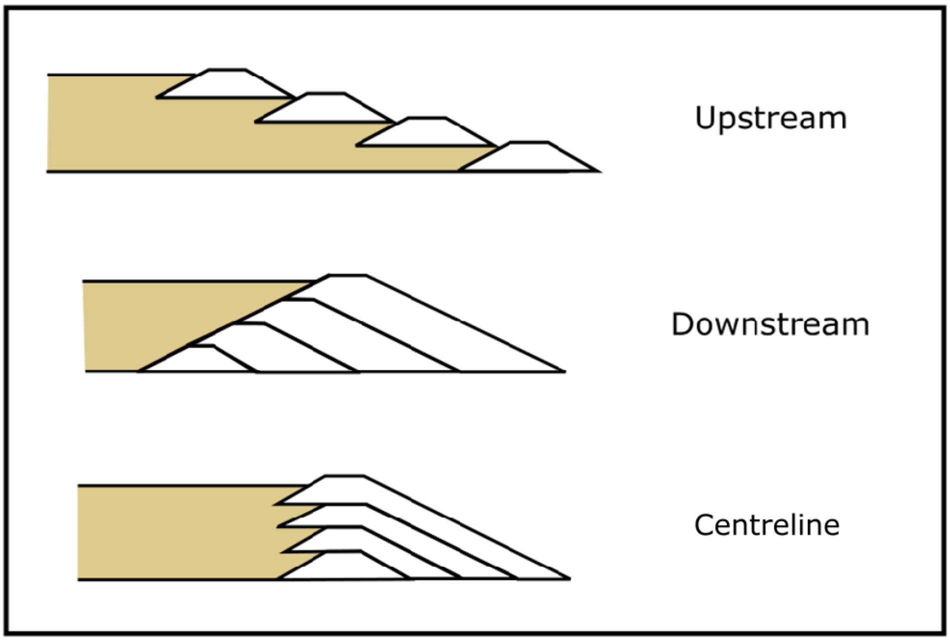


Figure 20-10: Simplified diagram of the three main embankment construction methods (Golder 2020)

This study combines two methods of tailings dam construction and uplift, upstream and downstream. The design of the embankments for the TSF and CP was based on the criteria summarised in Table 20-4.

Table 20-4: Embankment Dam Raise Design criteria

Criteria	NAG TMF			PAG TMF	Clarification pond
	West Dam	North Dam	East Dam		
Construction Type	Downstream	Downstream	Upstream	Downstream	Downstream
Crest With	6	6	6	5	5
Downstream Slope	2.5 (H) : 1 (V)	2.5 (H) : 1 (V)	5.0 (H) : 1 (V)	3.0 (H) : 1 (V)	3.0 (H) : 1 (V)
Upstream Slope	2.5 (H) : 1 (V)	2.5 (H) : 1 (V)	2.0 (H) : 1 (V)	2.5 (H) : 1 (V)	2.0 (H) : 1 (V)
Crest Elevation	281 mASL*	282 mASL*	283 mASL	277 mASL	236
Water Freeboard	1.5 m	2.3 m	0.5 m	1 m	1 m
Tailings Freeboard				2 m	n/a**

*The difference between elevation of North Dam and West Dam comes from different final tailing elevations. Western Dam will have tailings elevation of + 279 m whereas for North Dam the final tailings elevation is expected to be +280 m.

**n/a – not applicable. There are no tailings in clarification pond.

When assessing options for tailings disposal facilities from a technical perspective there are four main factors that can be used to evaluate the suitability of options:

- Storage capacity, which is the total volume of tailings which can be stored and hence years of operation;
- Construction material requirement, which adds to the cost of the facility;
- Ratio between storage capacity and construction material, represents in effect the efficiency of a tailings facility. The higher the ratio, the more tailings can be placed for every unit of construction material required; and

Rate of rise of the facility, which informs on the viability of raising a facility as it may impact the stabilityThese parameters are directly related to the method of uplift used.

20.3.5.2 Embankment dams consequence classification

The current classification of the safety class (Länstyrelsen Gävleborg, 2019) for the SW, NW and E Dams is presented in Table 20-5.

Table 20-5: Classification of the safety class (Länstyrelsen Gävleborg, 2019)

Dam	Dam safety classification
Dam Uxatjärn (clarification pond)	C
TSF NW Dam	C
TSF SW Dam	U (no classification)
TSF E Dam	U (no classification)

Based on the final design, the classification of the dam based on an update dam breach analysis will have to be reviewed and updated should it be required

20.3.5.3 Flood event

The extreme flow corresponding to a 24-hour extreme precipitation with a return period of 100-years (flood event) is calculated for the TSF and CP (Golder, 2021a), see Table 20-6.

Table 20-6: The estimated flow corresponding to a 24-h extreme precipitation with a return of 100-years (Golder, 2021).

Description	Unit	TMF	Clarification pond
Average flow	m^3/s	0.94	1.23
Exceeded average flow	m^3/s	1.68	2.19

20.3.5.4 Mine Waste Production

The operation is estimated to produce two basic types of mine waste. Waste rock and tailings. Their production figures are summarized in Table 20-7 and Table 20-8.

20.3.5.4.1 Waste rock

Table 20-7: Waste Rock - LoM (From MPlan)

Description	Data	Unit
Lifetime of the mine (LoM)	16	year
Total tonnes mined (ore and waste)	14,450,000	t
RoM production	2,970,000	t
Total mass of waste rock	10,060,000	t
Total mass of overburden waste (moraine)	1,420,000	t
Waste rock for mine backfill	6,380,000	t
Waste rock to waste rock piles	5,100,000	t

t = metric tonne;

20.3.5.4.2 Tailings

The particle size of the enrichment sand varies between a sand (63–200 µm) and a silt (20–63 µm). The curves of the wet sieving are presented below in Figure 20-11. The bulk density of the tailings was determined to be 1.75 t/m^3 (Ekblom 2012), with an estimated dry density of 1.4 t/m^3.

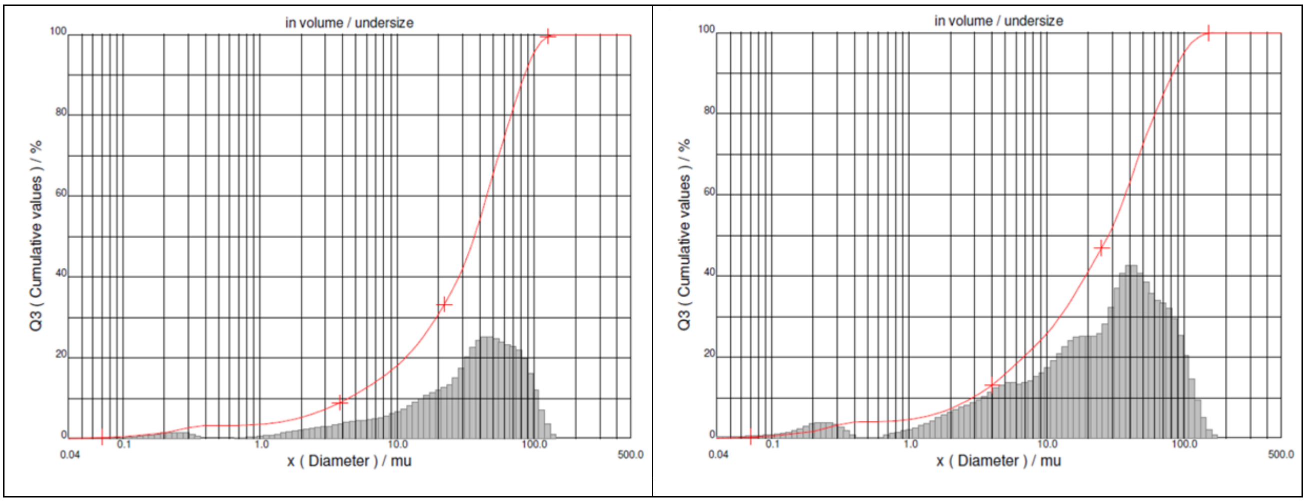


Figure 20-11: Particle Distribution Curves for the existing TMF, oxidized (Ekblom, 2012).

The tailings deposited in the TSF for the PEA Project design consists of the mineral quartz, feldspar, mica, calcite, and graphite. The predominant sulphide mineral is magnetic (FeS). There is also pyrite (FeS_2) and zinc aperture (ZnS). The sulfur concentration in the non-weathered tailings stored in current NAG is 3.88%.

The tailings have the potential to produce acidic leachate with possibly elevated metal levels. The existing tailings has been subjected to weathering due to dry conditions and the upper layer is relatively oxidised. Concentrations of barium, nickel, vanadium, and zinc exceed the target levels for less sensitive land use (AFRY, 2020).

This study has been undertaken on the basis that two tailings streams will be produced from the plant. One tailings stream is called the Non-acid Generating (NAG), while the other is the Potential Acid Generating (PAG). The NAG tailings will represent 90% of the tailings production and will be deposited on the existing tailings area, while the PAG tailings represent 10% of the total tailings production and will be deposited in a separate tailings are built purposely.

The planned tailings production is summarised in Table 20-8.

Table 20-8: Estimated tailings production (LEM)

Description		Unit	Value
Life of Project (LoP)		years	19
Annual Tailings production rate		t/year	160 000
Assumed in-situ Density		t/m^3	1.4
Annual Tailings Volume	NAG (90%)	m^3 /year	102,857
	PAG (10%)	m^3 /year	11,429
	Total	m^3 /year	114,286
Total Volume over LoP	NAG (90%)	m^3	1,954,286
	PAG (10%)	m^3	217,143
	Total	m^3	2,171,429

20.3.5.5 Coordinate and elevation system

The coordinate systems SWEREF99 TM (X, Y) and RH 2000 (Z) are used in this study.

20.3.6 TSF conceptual design

As mentioned in the previous section, two separate tailings management facilities are planned to be constructed to accommodate the proposed LoP tailings volume:

- the current TSF footprint will be used for the storage of the non-acid generating tailings (TSF NAG), extended to the North; and
- a smaller TSF for the potentially acid generating waste (TMF PAG), built in the eastern part of the TMF NAG.

The final proposed layout is shown in Figure 20-12.

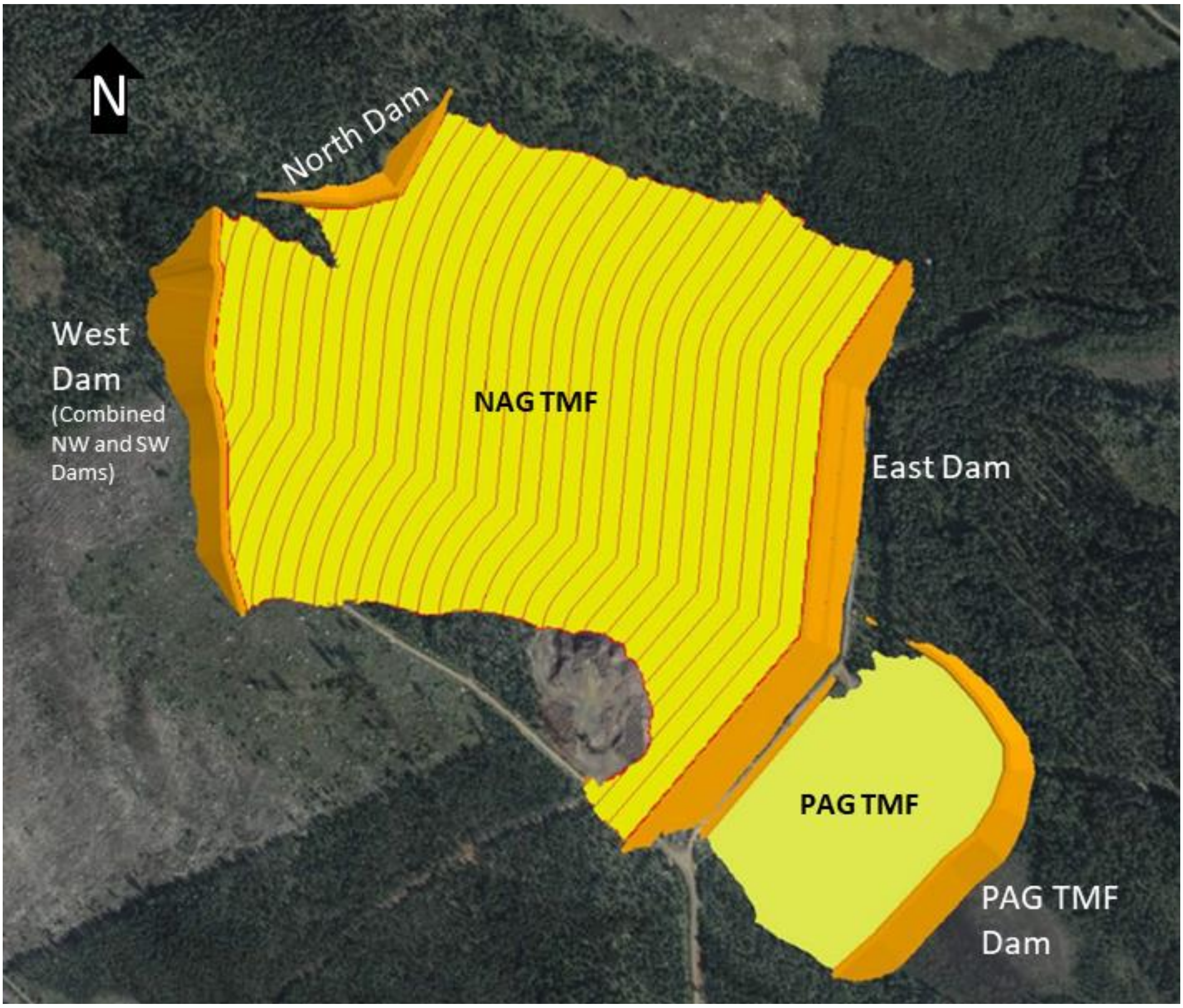


Figure 20-12: TMF Conceptual Raise Design.

Details of the NAG and PAG TMF design is discussed in further detail in the subsequent two Sections.

20.3.6.1 NAG TMF conceptual design

The NAG TMF Raise will consist of the following:

- West Dam – Downstream raising of the existing NW and SW Dam to elevation +281 m (Figure 20-13 and Figure 20-14).
- East Dam – Upstream raising of the existing East dam to elevation +283 m and lengthening by 380 m to the north (Figure 20-15).
- North Dam – Construction of a new rockfill dam (with upstream moraine wedge) to an elevation of +282 m in the north valley (Figure 20-16).

Further details of the dam raises are discussed in the sections below.

20.3.7 West Dam typical cross section

The typical cross section (starter dams and subsequent downstream raises of the west dams is presented in Figure 20-13 and Figure 20-14.

The current concerns of seepage and the potential for internal erosion through the existing dams will be mitigated thought initial excavation and strengthening work, as follows:

- The slope cleared from any vegetation and any unsuitable material will be excavated.
- A moraine wedge will be placed against the downstream slope to reduce the seepage.
- A filter layer will be placed downstream of this to prevent internal erosion and piping.
- A downstream rockfill shoulder will be added to improve stability.
- The average downstream slope will be flattened to 1 Vertical (V) : 2.5 Horizontal (H).

During the initial phase, the dam crest will remain at its current elevation. During the LoP, the embankment will be raised in three stages based on the deposition and construction schedule. The dam raise design consist of a rockfill shoulder with low permeability moraine layer 4-metres wide. A 2-metre-wide filter layer will be placed between the moraine and rockfill zones.

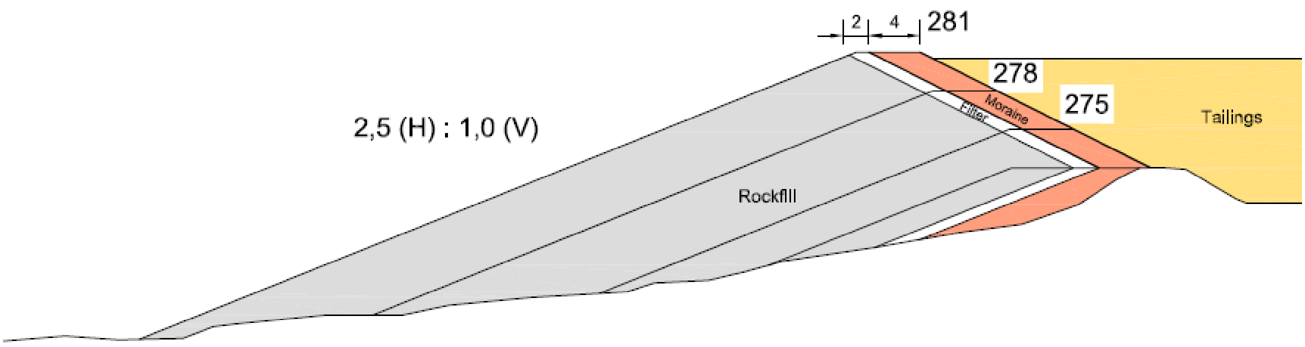


Figure 20-13: TSF NAG West Dam typical cross section (at existing Northwest dam location)

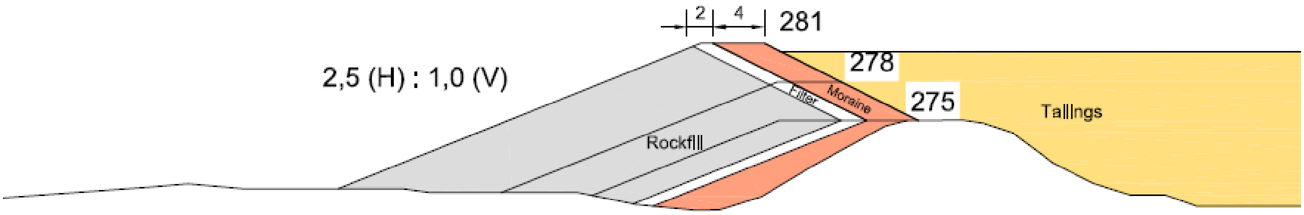


Figure 20-14: TSF NAG West Dam typical cross section (at existing Southwest dam location)

20.3.7.1.1 East Dam typical cross section

The typical cross section (starter dams and subsequent downstream raises of the western dams is presented in Figure 20-15.

The current concerns of seepage and the potential for internal erosion through the existing dams will be mitigated though initial excavation and strengthening work, as follows:

- The slope cleared from any vegetation and any unsuitable material will be excavated.
- A filter layer will be placed downstream of this to prevent internal erosion and piping.
- A downstream rockfill buttress will be added to improve stability with the average downstream slope will be flattened to 1(V):2.5(H).

The dam will then be raised upstream through the construction of two 3 m high berms, to form an average upstream slope of 1(V):5(H). Each berm will consist primarily of rockfill with a 2 m wide filter section and a 4 m wide upstream erosion protection wedge.

The East Dam will be required to be lengthened to the north, with a portion of the first upstream raise (crest El. +280 m) to be constructed on natural ground.

PAG tailings will eventually be deposited against the toe of the existing East Dam.

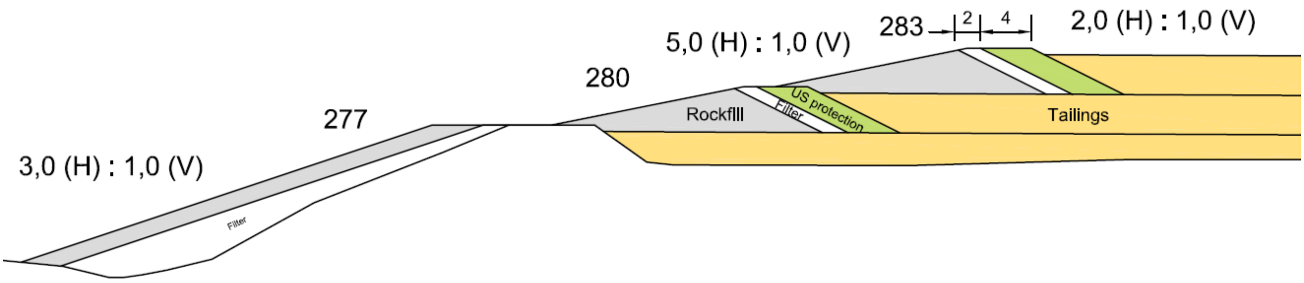


Figure 20-15: TSF NAG East Dam

20.3.7.1.2 North Dam Typical Cross Section

A new dam will be required within the northern valley to contain the tailings during the latter part of the LoP.

This North Dam will be a downstream raised rockfill dam similar to the proposed construction for the existing West embankments and will constructed in three stages at elevations 276, 279 and 282 mamsl. A 4 m wide moraine wedge will be placed against the upstream slope to reduce seepage, and filter layer will be placed between the moraine and rockfill.

The typical cross section is presented in Figure 20-16.

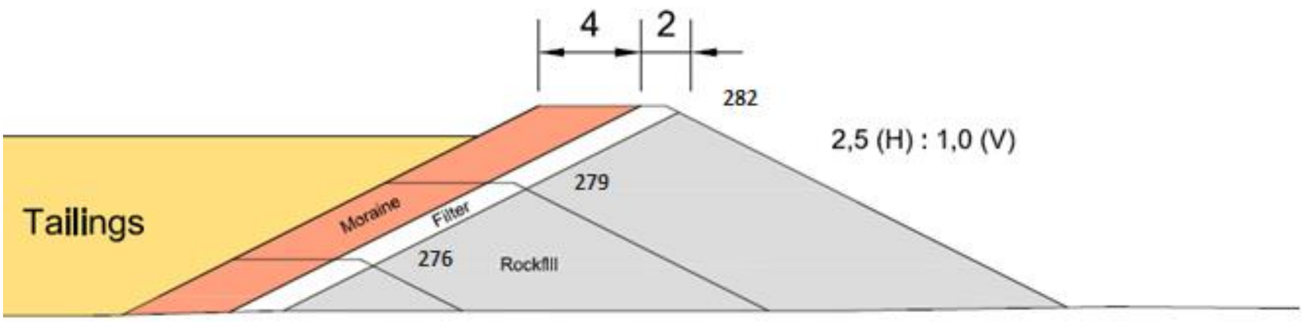


Figure 20-16: TSF NAG North dam

20.3.7.1.3 TSF NAG closure

The closure is discussed in section 20.4.7.

20.3.7.2 PAG TSF Conceptual design and section

The new PAG TSF is to be a lined facility to prevent seepage from the PAG tailings into the environment. The liner will be placed on the basin foundation and against the upstream face of the dam. The lining system proposed consists of a bituminous geomembrane which can be placed directly on the prepared subgrade. The exact detail of the lining system will, however, be determined during later design stages.

The PAG TSF will be formed by the construction of an eastern rockfill embankment dam (Figure 20-17). The dam will have a 4 m wide moraine wedge placed on the upstream face, and with a 2 m wide filter layer between the moraine and rockfill. The downstream slope will be constructed at a grade of 3(H):1(V).

The PAG tailings will be kept 2 m under water consistently to prevent oxygen ingress and potential oxidation and acid generation from the PAG tailings.

The dam will be constructed in two stages based on the deposition and construction schedule.

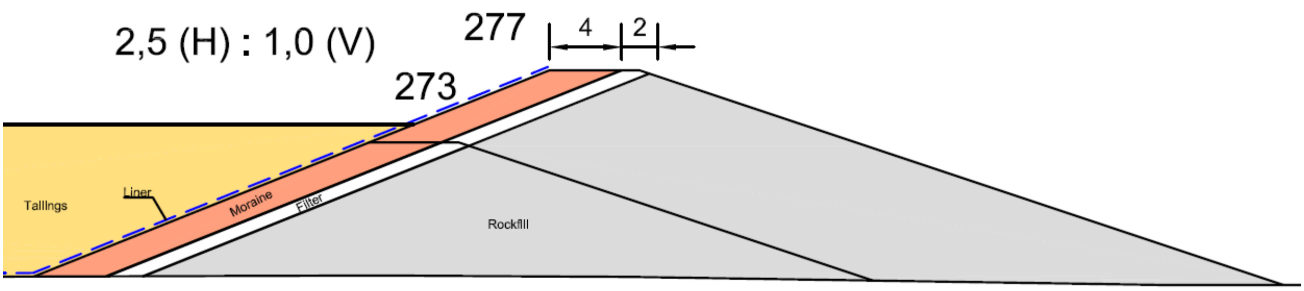


Figure 20-17: PAG TSF Embankment Dam

20.3.7.2.1 TSF PAG closure

The closure is discussed in section 20.4.7.

20.3.7.3 Tailings deposition modelling

The tailings deposition over the life of the facility has been modelled in Muck3D^{TM} software. and the results of the modelling shown in APPENDIX B.

The deposition within the NAG TSF will initially be from the intermediate berm in the centre of the facility to deposit tailings against the original NW and SW Dams. Deposition will then move back to the NAG East Dam, from where the tailings will be uniformly spigotted from over the life of the facility. Deposition within the PAG TSF will be from the dam on the eastern perimeter.

Tailings Storage curves for the NAG TSF (Figure 20-18) and PAG TSF (Figure 20-19) have been generated and show the elevation increase and approximate timeline for the tailings deposition and embankment raises.

The rate of rise (RoR) was estimated for both facilities (Figure 20-20). The NAG TSF RoR is initially approximately 0.3 m/year and decreases over the life to less than 0.1 m/year. This is well below the typical recommended maximum RoR of 2 m/year for an upstream raise facility. The RoR for the PAG SMF is initially high due to filling of the low point within the natural valley. This is not a concern as it is a downstream raised facility.

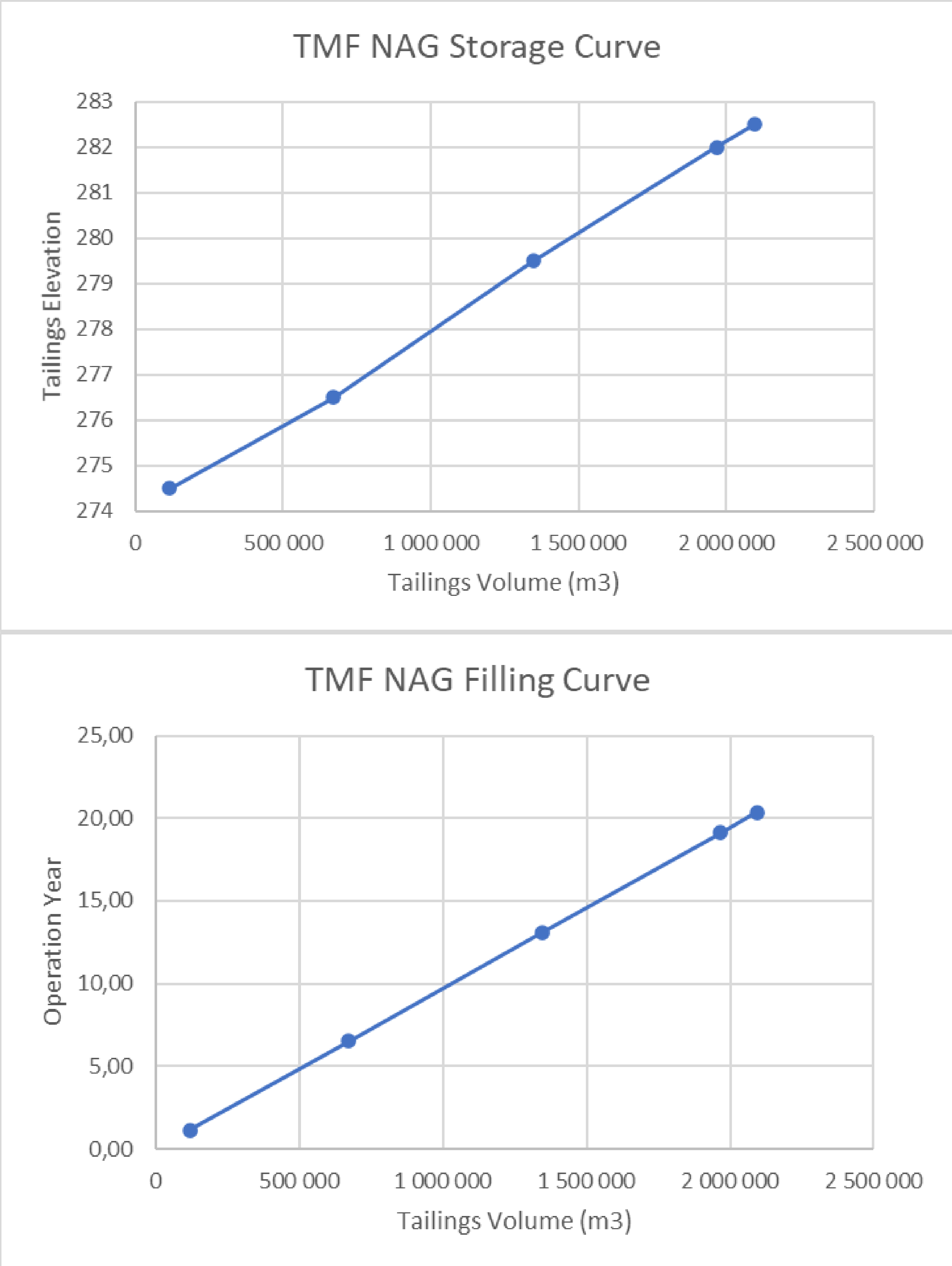
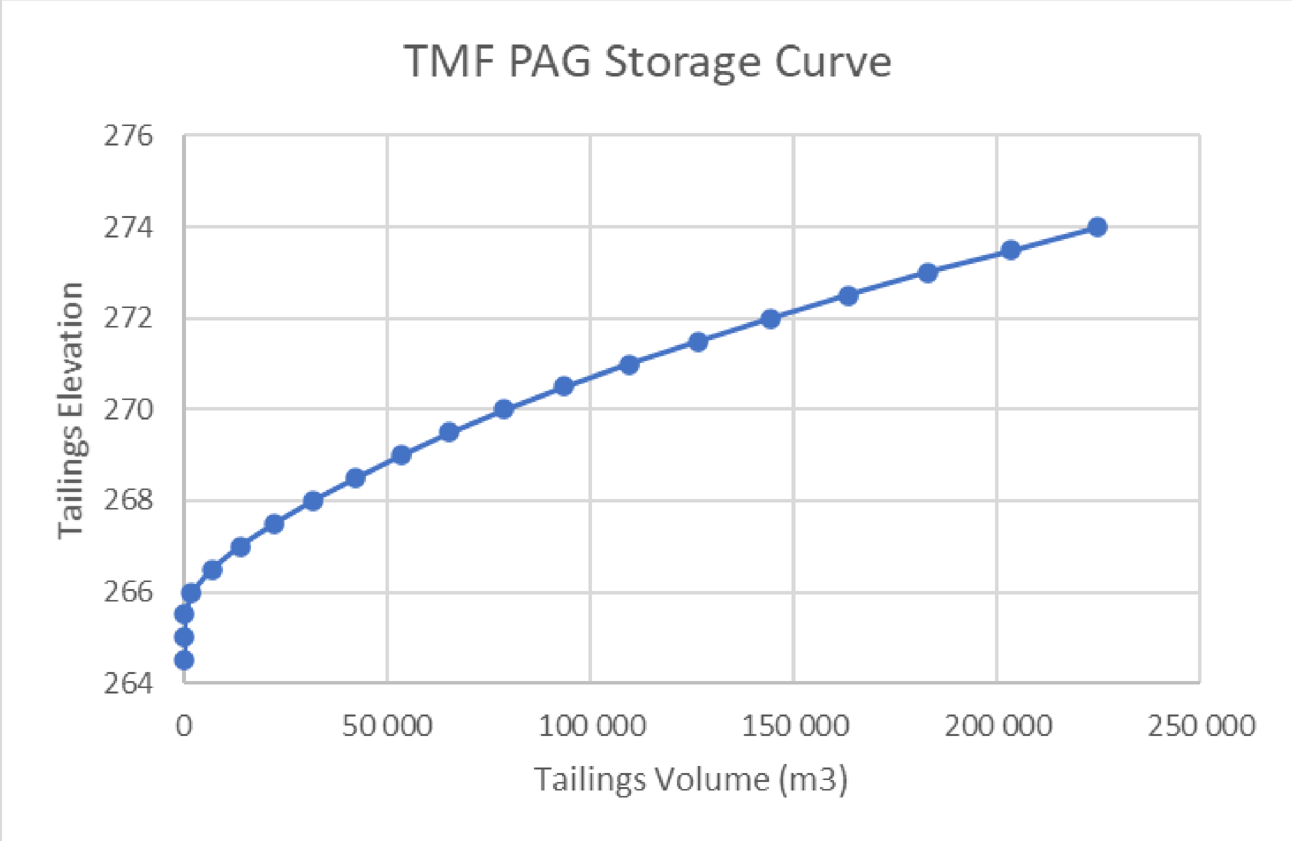


Figure 20-18: NAG TSF Tailings Storage Curves



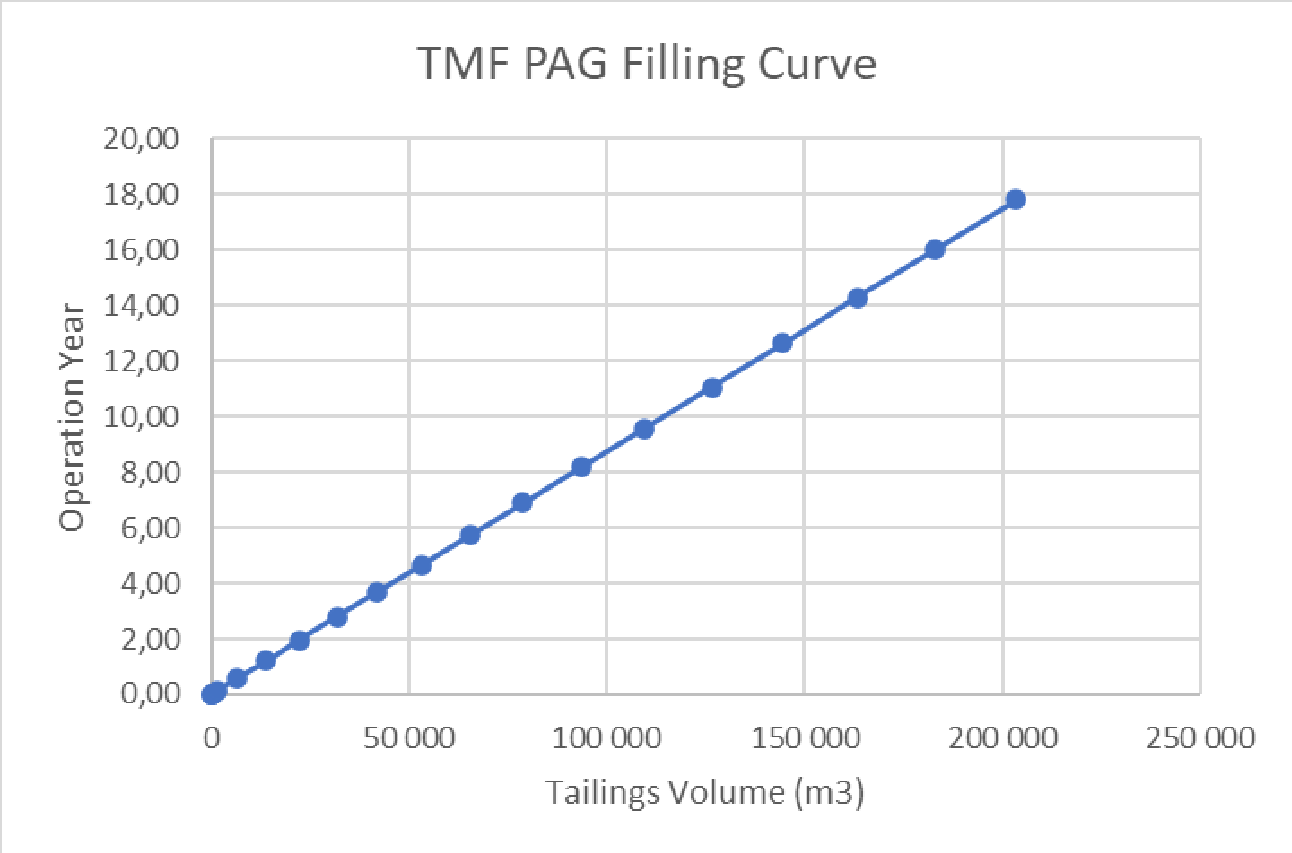
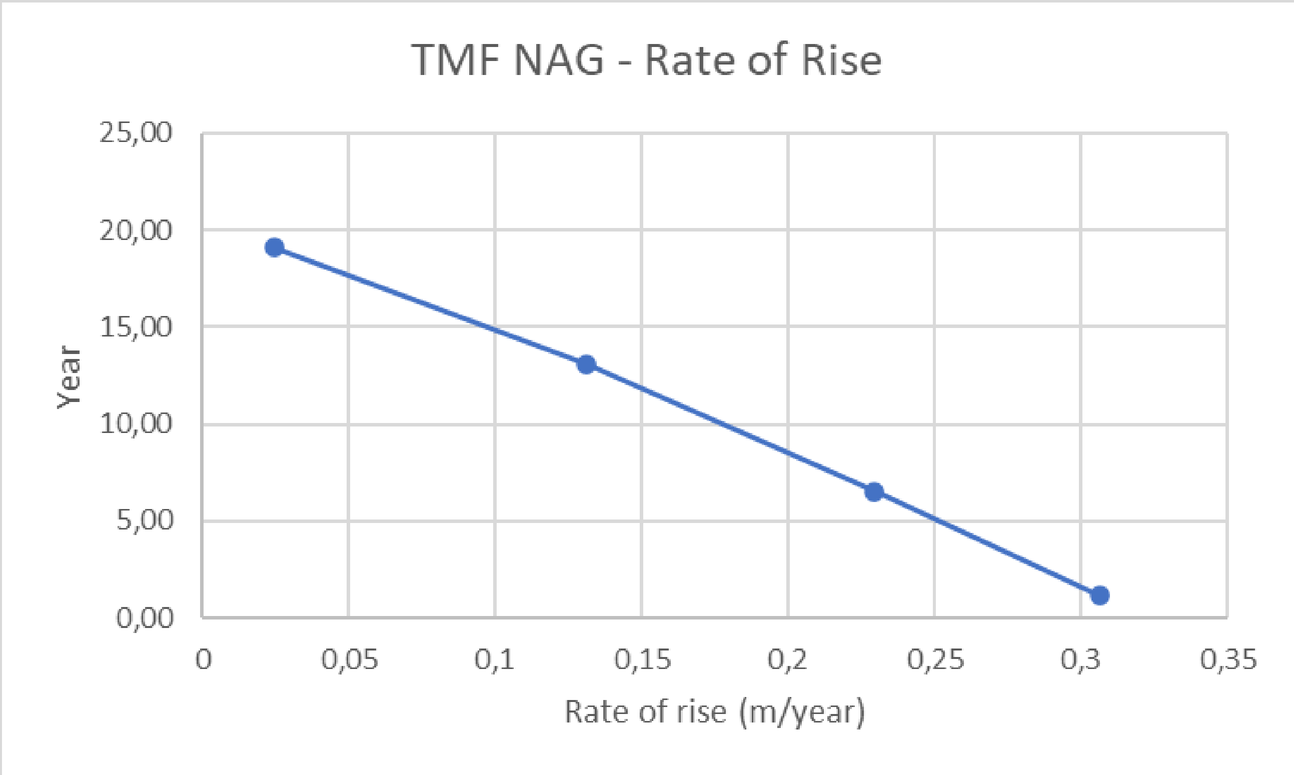


Figure 20-19: PAG TSF Tailings storage curves



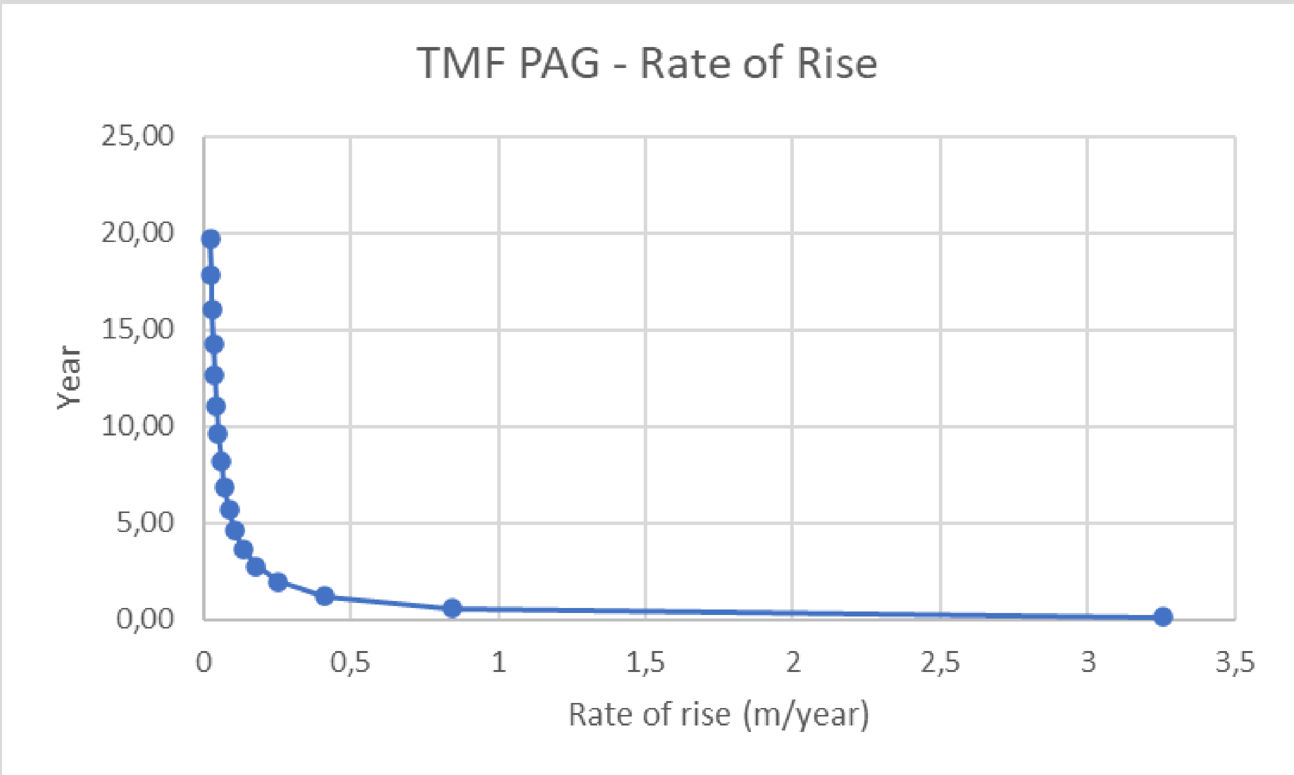


Figure 20-20: NAG and PAG TSF estimated Rate of Rise

20.3.7.4 Construction schedule

The schedule of TSF construction is based on the rate of filling of the facilities. It has been assumed that each raising of the embankments can be completed over a period of one year prior to that particular raise being brought into operation.

The construction programme was therefore divided into several staged raises taking place in years 0, 1, 6, 9 and 12. The construction in each year is detailed in the sections below. An overview of construction and deposition schedule is presented in Table 20-9.

20.3.7.4.1 Year 0

This includes all necessary work required to be undertaken prior to the restart of the mine, which is assumed to be at the beginning of year 1. The following work will allow for deposition capacity of approximately 1.14-years. This phase includes:

- Remediation and construction of shallower downstream slopes for the existing SW and NW dams (NAG TSF).
- Construction of a buttress for the East Dam (NAG TSF).
- Construction of the PAG TSF embankment dam to a crest elevation of +273 m.

The plan view is shown in Figure 20-21.

20.3.7.4.2 Year 1

This phase is undertaken during the year 1 of the LoP and will provide tailings storage of approx. 553,000 m^3 corresponding to 5.38 years of deposition. The construction includes:

- NAG TSF existing NW Dam raised to elevation of +275 m (existing SW Dam already at this elevation).
- North dam constructed to crest elevation of +276 m.

The plan view is shown in Figure 20-22.

The CP embankment dam is also to be raised to crest elevation +236 m (as discussed in Section 20.3.8).

20.3.7.4.3 Year 6

The following construction activities will be undertaken during the year 6.

- Raising of the NAG TSF West Dam to elevation +278 m.
- Raising of the NAG TSF North Dam to elevation +279 m.
- Raising of the NAG TSF East Dam to elevation +280 m.

This stage provides a deposition life of an additional 6.56 years with a tailings storage volume of approx. 675,000 m^3. The plan view is shown in Figure 20-23.

20.3.7.4.4 Year 9

During year 9 the construction of the second raise of TSF PAG Dam to elevation 277 m will be undertaken.

20.3.7.4.5 Year 12

The following construction activities will be undertaken during year 9.

- Raising of the NAG TSF West Dam to elevation +281 m.
- Raising of the NAG TSF North Dam to elevation +282 m.
- Raising of the NAG TSF East Dam to elevation +283 m.

This will provide for a deposition life of an additional 6.02-years with a tailing's storage volume of approximately 620,000 m^3.

The plan view is shown in Figure 20-24.

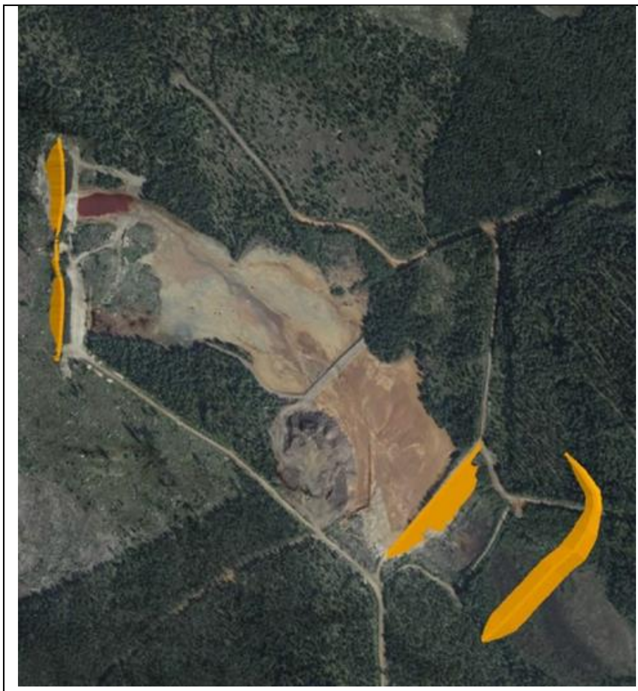


Figure 20-21: Year 0 Dam Construction

- PAG TSF 1st Stage Built (+ 273 m)
- Buttress of East Dam of the NAG TSF (+277 m)

Figure 20-22: Year 1 Dam Construction

- Raise NAG TSF Northwest Dam (+275 m)
- Construct NAG TSF North Dam (+276 m)

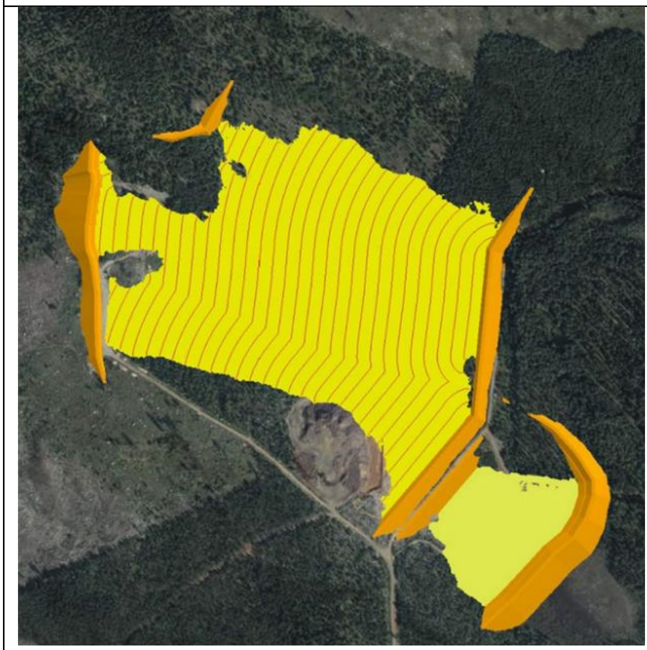


Figure 20-23: Year 6 and 9 Dam Construction

- Raise NAG TSF West Dam (+278 m)
- Raise NAG TSF East Dam (+280 m)
- Raise NAG TSF North Dam (+279 m)
- Raise PAG TSF (+277 m) in Year 9

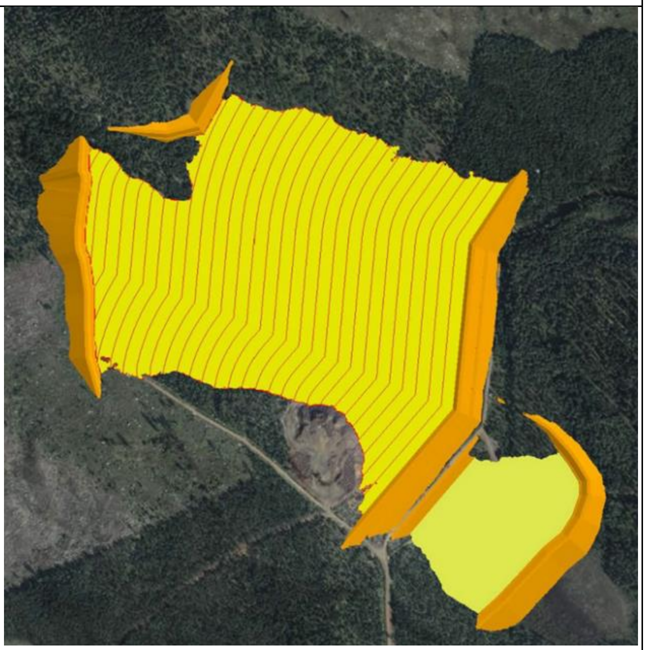


Figure 20-24: Year 12 Dam Construction

- Raise TSF NAG West Dam (+281 m)
- Raise NAG TSF East Dam (+283 m)
- Raise NAG TSF North Dam (+282 m)
- Raise PAG TSF (+277 m)

Table 20-9: Construction and deposition schedule with final elevations at the end of particular year.

Year	TSF NAG West Dam		TSF NAG Intermediate Dam	TSF NAG North Dam	TSF NAG EDam		TSF PAG		Clarification pond	Comment
	NW	SW								
0	DS slope	275	Deposition	Initial	Buttr.		273 m	Deposition		construction of DS slope of Western dam, buttress of Eastern dam and PAG, deposition from intermediate berm and PAG
1	275 m			276 m		Deposition		Deposition	236 m	Western dam built to 275 m in the full length, North dam built to 276 m, uplift of clarification pond to 236 m, deposition from intermediate Eastern dam and PAG
2–5						Deposition		Deposition		No construction, deposition from Eastern dam and PAG
6	278 m			279 m	280 m	Deposition		Deposition		Uplift of Western wall in the full length to 278 m, Uplift of Northern wall to 279 m and Eastern wall to 280 m. Deposition in PAG and from Eastern wall.
7–8						Deposition		Deposition		No construction, deposition from Eastern dam and PAG
9						Deposition	277 m	Deposition		Uplift of PAG to 277 m, deposition from Eastern dam and PAG
10–11						Deposition		Deposition		No construction, deposition from Eastern dam and PAG
12	281 m			282 m	283 m	Deposition		Deposition		Uplift of Western wall in the full length to 281 m, Uplift of Northern wall to 282 m and Eastern wall to 283 m. Deposition in PAG and from Eastern wall.
13–20						Deposition		Deposition		No construction, deposition from Eastern dam and PAG

20.3.7.5 Construction material requirements

For all construction works there are four basic materials that have been utilised in the design and reflected during the cost estimation phase. The cost estimations are covered in Section 21.

- Rockfill - works as a buttress material used at the downstream side of the dam structures.
- Filter – prevents internal erosion and piping of the low permeability moraine into the supporting rockfill. For this study, a single filter material has been assumed although it is likely that two different filters (fine filter and transition) may be required when the design will be detailed. The overall material volume will remain unchanged, however, as the thickness assumed can accommodate for this.
- Moraine – low permeability material installed on the upstream slope of the dam to reduce seepage.
- Liner – Installed along the base and on the upstream face of the TMF PAG to prevent seepage into the foundation.
- Foundation preparation- assumed to be one 1 m thick for every surface require improvement/preparation.

The estimated material quantity and foundation preparation required is summarised in Table 20-10 and is based on the construction schedule presented in Section 20.3.7.4. Figure 20-25 shows graphically the material quantities required over the proposed life of the mine.

The quantities shown include all materials required for the raising of the NAG TSF, construction of the PAG TSF, and raising of the Clarification Pond embankment dam.

Table 20-10: Schedule of required construction material quantities

Year	Rockfill (m^3)	Moraine (m^3)	Erosion protection (m^3)	Filter (m^3)	PAG TSF Liner (m^2)	Foundation preparation (m^3)
0	35,000	8,000		11,000	54,000	66,000
1	15,000	6,000		2,000	None	12,000
6	74,000	16,000	67,000	7,800	None	28,000
9	61,000	5,000		2,300	None	None
12	112,000	12,000	38,000	7,000	None	23,000

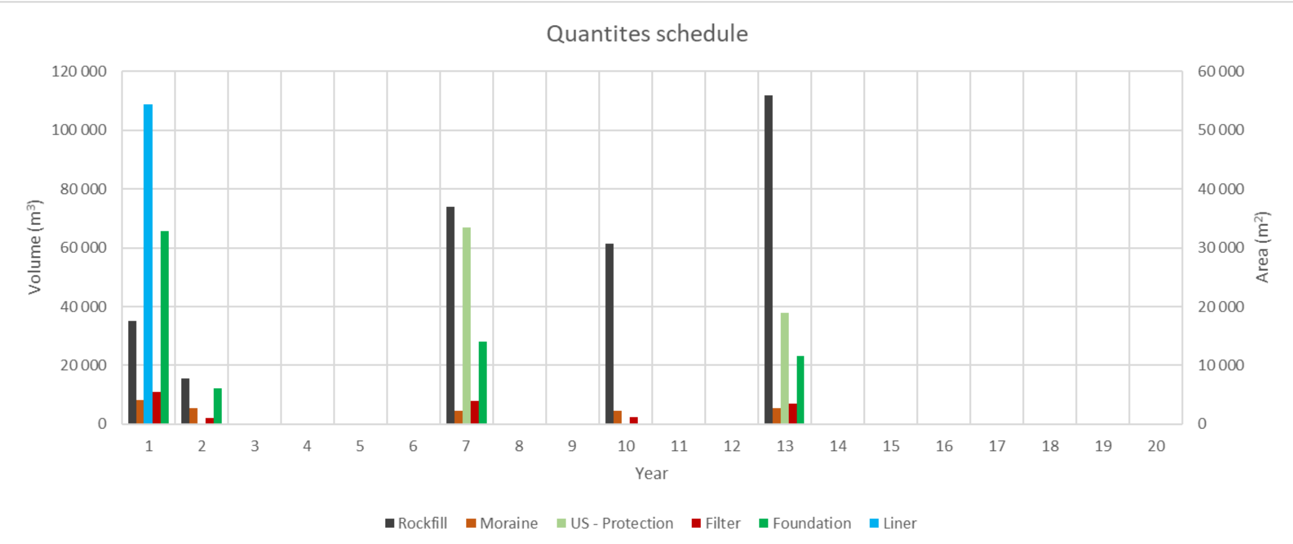


Figure 20-25: Material quantities schedule

20.3.7.6 Construction material sourcing

The requirement for construction material is relatively low and therefore it is assumed that all material will be borrowed and/or sourced locally. Investigation on potential borrow areas (moraine) and confirmation sourcing of the different material (rockfill, filters) will be required during the next stages of the studies. It has been assumed that the waste rock will not provide adequate construction material for rockfill due to its acid generation potential.

20.3.7.7 Operation, Maintenance and Surveillance

20.3.7.7.1 Operation

The operation of the TSFs require management of the tailings deposition to permanently store both streams of tailings, to manage water within TSF NAG, and to permanently store and manage water within TMF PAG.

The Tailings Storage Facilities are to be operated with the following philosophy:

- The tailings within the NAG TSF are to be deposited from the Intermediate berm during year 1 and from year 2 onwards around Eastern dam of TSF PAG, leaving no point significantly higher or lower than any other point such that the pond is maintained in the western portion of the facility.
- The minimum required beach length and beach freeboard is to be maintained in the NAG TSF (Intermediate berm and Eastern dam) through the sequencing of the deposition locations and scheduled embankment raise construction.
- The PAG tailings are to be deposited within TSF PAG:
- Leaving no point significantly higher such that the tailings remain permanently submerged; and
- Ensuring there is no excessive depth of water against the liner in the deepest part of the TSF PAG pond.
- The seepage from the facilities is to be minimised, with collection of seepage and run-off water downstream of the facility embankments.

20.3.7.7.2 Maintenance

Facility maintenance is necessary for the safe operation and the effective management of the two facilities.

Maintenance is divided into planned and un-planned maintenance:

- Planned preventative maintenance may be routine, predictive and/or from routine inspection observations (low risk/ consequence observations) where the maintenance is not required immediately and can be scheduled. Data collected during regular inspections of the facility should be reviewed when scheduling planned maintenance; and
- Un-planned (event driven) corrective maintenance shall generally derive from routine inspection observations (medium-high risk/consequence observations) and or extreme events (extreme meteorological events, seismic events, etc.) where the maintenance requirement is critical and is required immediately.

Maintenance and overall performance of the facilities involves the maintenance of the following components:

- embankments,
- perimeter Collection Channels,
- pipe and pumping Systems,
- monitoring Instrumentation, and
- site Access Roads.

20.3.7.7.3 Surveillance and Monitoring

A system of surveillance is to be established to monitor the performance of the facilities. This is to consist of routine and planned inspections and installation of geotechnical and environmental monitoring instrumentation.

The routine surveillance by mine staff will observe any changes in deposition, water elevation, water flow or any other operational change with direct impact on the performance of the facility. Technical review by experienced engineers as per the requirements in GruvRIDAS and GISTM are to be established.

Instrument installation may consist of the following:

- Settlement (and displacement) monitoring through the installation of survey points along the dam crest of all embankment dams at all stages of construction
- Displacement monitoring through the installation of inclinometer through the embankment crest. This can be installed at select points of each raise of the NAG TSF East Dam, and the final crest elevation of the remainder of the dams.
- Foundation pore pressure monitoring within the moraine foundation of all dams. This can be through the installation of standpipe or vibrating wire piezometers.
- Tailings pore pressure monitoring within the tailing's foundation of the upstream raised NAG TSF East Dam through the installation of standpipe or vibrating wire piezometers.
- Groundwater monitoring wells downstream of the facilities will be required to monitor for potential seepage.

Seepage monitoring points may also be required at the downstream toe, depending on the performance of the dams.
The exact detail and location of the monitoring instruments will be defined in later design stages.

20.3.8 WATER MANAGEMENT

In the water balance of the mine site, the processing plant, the open pit, the TSF and the CP were included as the active parts.

The water management is presented in Figure 20-26 and can be summarised as follows:

- The open pit will be dewatered to be able to start mining operations and will be pumped continuously during mining operations.
- The water pumped from the open pit will go to the concentrator plant prior to use for the process (*Flow L1*). The water will be cleaned, e.g. sedimentation (*Flow OP1*) and liming (*Flow S1*) before it reaches the concentrator plant. Potentially, the water can be pumped from the open pit directly into the TSF (*Flow OP2*).
- From the processing plant, water will be pumped into the TSF NAG (*Flow P1*) and TSF PAG (*Flow P2*).
- Two separate TSFs are designed: the existing facility for tailings originating from non-acid forming rock (TSF NAG) and a smaller tailings facility for potentially acid-generating tailings (TMF PAG). The intermediate berm divides the existing TSF NAG into an eastern and western part. In the water balance, the TSFs are considered as one unit. However, under mining operation, the water from TSF PAG will be pumped into TSF NAG (*Flow TMF1*) to control the water level in the TSF PAG and the seepage flow into the wetland Stormyran.
- From the western part of the TSF NAG, the water will be pumped into the clarification pond (*Flow TMF2*). The water in the tailings facilities consists of the precipitation, run-off water from the surroundings and the water pumped form the process plant.
- From the CP (Uxatjärn), the water will be pumped through the liming station (*Flow CP*) before it will be pumped either into the processing plant and used for the process (*Flow L1*) or discharged into the Älmån river (*Flow L2*).

Evaporation and seepage are included in the water balance in the form of water losses from the TSF and the CP.

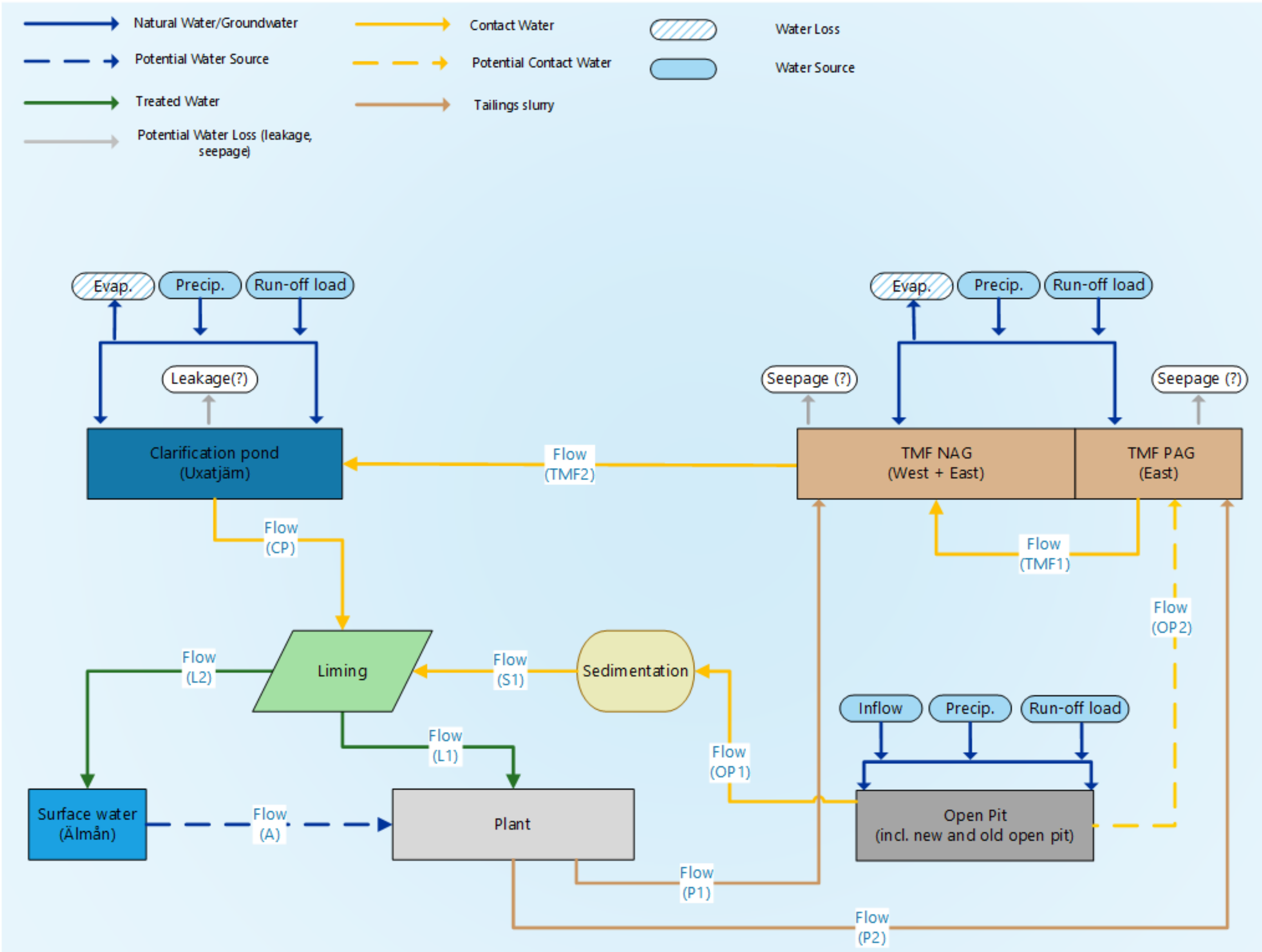


Figure 20-26: Conceptual water balance model with mining operation (Golder, 2021).

20.3.8.1 Clarification pond

The use of the small Lake Uxatjärn, located close to the TSF, as a CP for the mine will continue. There is an existing small embankment dam for the CP. The CP will be used in its current state during the first year of operation before the embankment be rehabilitated and raised for the remaining LoP, to ensure it is brought to required technical standard as well as to provide greater operational flexibility.

The embankment dam needs to be rehabilitated and raised, and the spillway will be redesigned to accommodate appropriate design floods. The embankment dam will be constructed similar to the original dams of the TMF. The dam will have a low permeability moraine core with supporting rockfill shell on the upstream and downstream slopes. with the dam will have a 1(V):2(H) upstream slope and a 1(V):3(H) downstream slope. The crest of the dam embankment will be +236 m. The proposed configuration is shown in Figure 20-27, and a typical cross-section is shown in Figure 20-28.

Water from the clarification pond will be reused into the processing plant. If excess water is required to be discharged, it will undergo treatment before being released to the Älmån river.

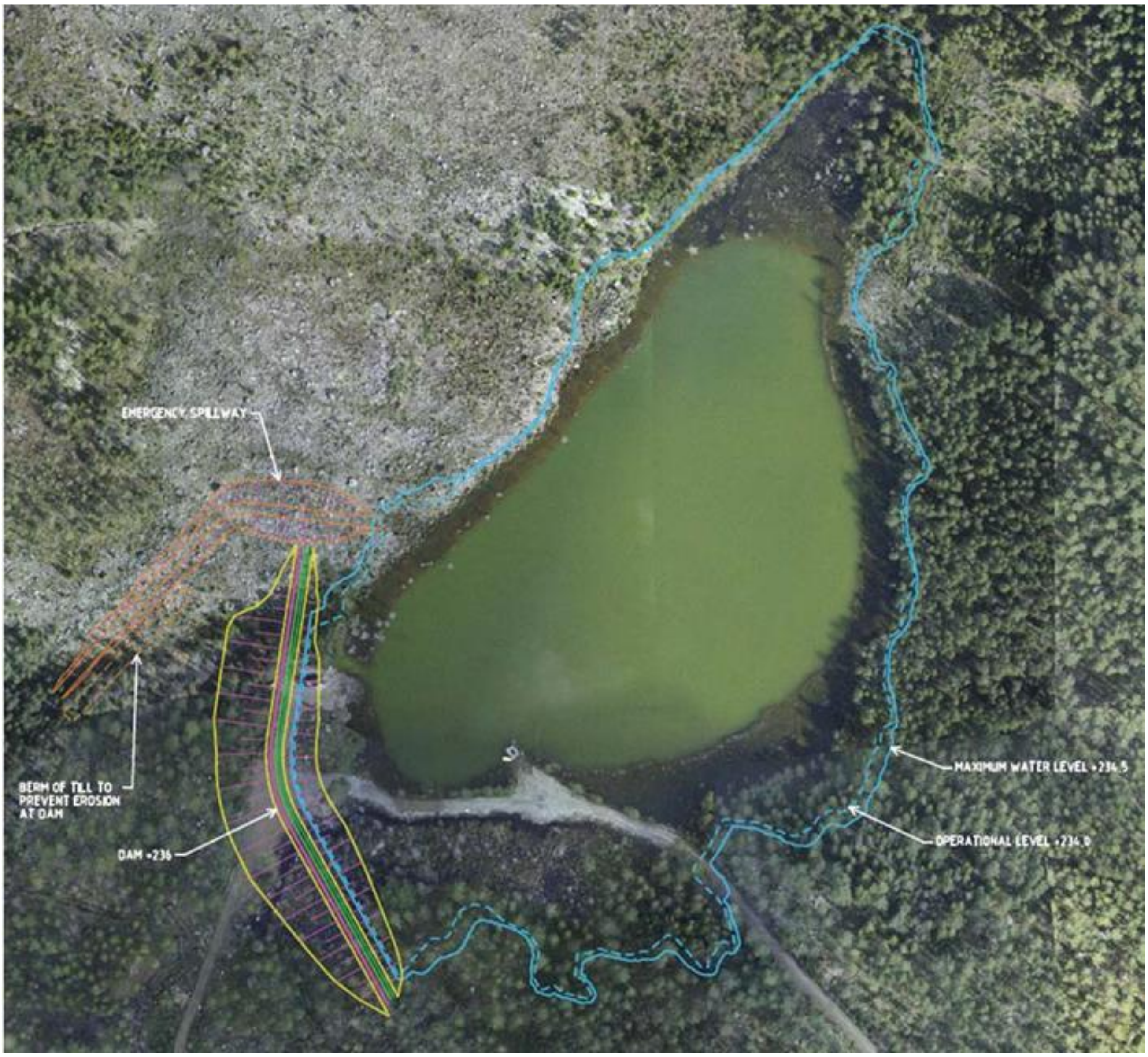


Figure 20-27: Clarification Pond Proposed Expansion.

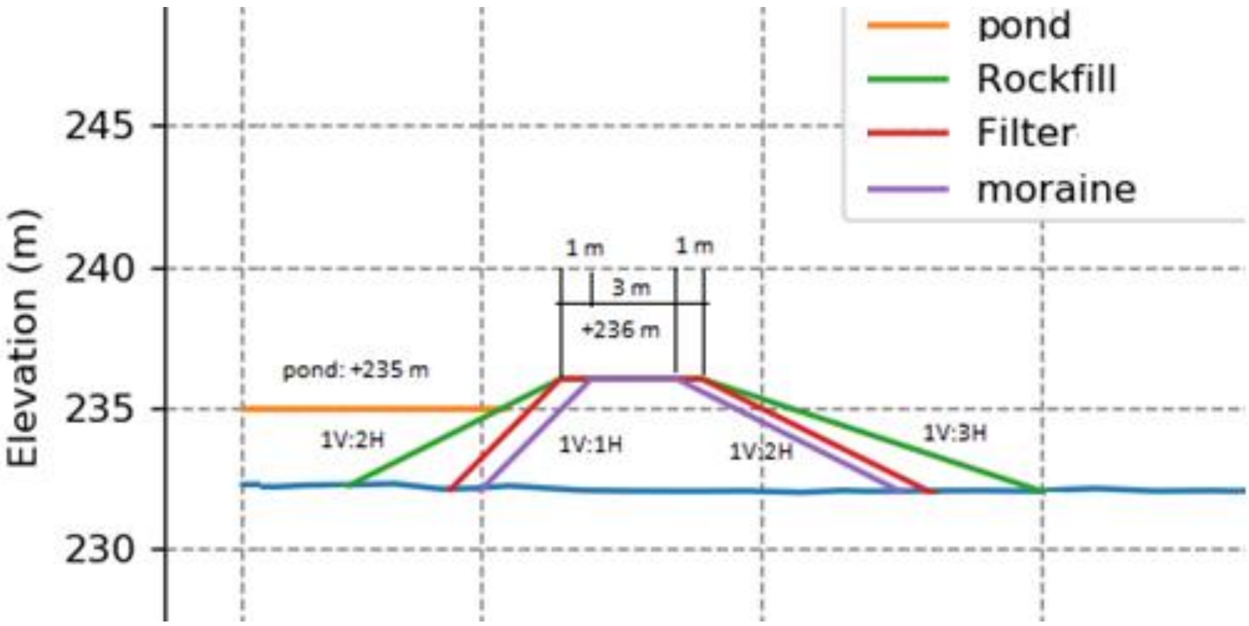


Figure 20-28: Simplified cross section through proposed solution for CP (uplift to 236 mamsl).

20.3.8.2 Spillways

In the current design, two spillways are planned:

- Spillway from the NW dam of the NAG TSF to the clarification pond
- Spillway at the CP

Due to the operating philosophy and the configuration of the PAG pond (tailings kept underwater, no external catchment area, reclaim water barge, no spillway has currently been incorporated in the design for the PAG TSF.

20.3.8.2.1 Dam spillway

Since cessation of activities on site a spillway has been constructed at the North-West boundary of the TMF for discharge of water to the clarification pond, see Figure 20-29.

Figure 20-29: The existing spillway at the TSF NW dam.

The choice of spillway for emergency discharge will require more detailed studies. Two design options would be adequate, either an open channel cut into the embankment (similar to the one currently built), which would have to be rebuild for every raises, or a central intake tower with a pipe leading water through to the dam wall. From a dam safety and regulatory perspective, the open channel spillway is a preferred option and has been assumed for this study.

20.3.8.2.2 Clarification pond spillway

In the CP, the water level is intended to be lowered so the highest water level at a high flow (100 years flow, see Table 20-6) is below the level of the natural ground of the downstream dam, e.g. +232.0 m. In this case, the dam at Uxatjärn will not be loaded neither at normal flow nor in the case of an extreme flow.

The regulation of the water level is intended to take place through:

- Operation spillway: Pumping through the treatment plant (to plant or recipient)
- Emergency spillway: Wide threshold (5 meters) in position for previous spillway.

20.4 Rehabilitation and closure

This PEA conceptual closure plan briefly describes the suggested future closure activities at the Project following the ending of the operations.

Mine closure and rehabilitation intends to restore the land areas used during the mine's operation to resemble the natural conditions in the area. Any release of environmentally harmful substances should be managed and reduced to acceptable levels.

The time perspective for the closure of mining areas is in Sweden "philosophically" set to the next ice age. The international practice used today implies that post-treatment of mining areas must be stable for a period of 1,000-years or more. The time perspective of 1,000-years is the starting point for the post-processing of the activities described in this closure plan. Hence, monitoring would be needed for at least several years.

At the time of the closure this must be done with the best available technique (BAT). The closure plan therefore needs to be updated continuously during the LoP in order to include additional items and changes in the operation.

20.4.1 Methodology

Golder presents several closure options for the Project, but the closure costs have been calculated for only one main option. The chosen option has been deemed, based on current knowledge of the planned operation, to be the most suitable option based on existing knowledge about Swedish closure strategies and practice.

The overall closure objective is to provide a "walk away solution". If the closure objectives are not met additional work must be performed during the monitoring period which is expected to be up to 30-years according to current regulatory practise.

20.4.2 Limitations

The current conceptual closure plan includes only the planned operation. It does not include the further planned refining of graphite in Edsbyn, as that operation is not directly connected to the mining operation and hence does not need a closure plan.

20.4.3 General description of the planned operations

Based on current knowledge the operation comprises the following parts:

- Mining of RoM in an extended open pit;
- Plant area with concentrator, workshops, storage buildings, facilities for employees, etc.
- Disposal of tailings in two separate compartments in the TSF – Desulfurised tailings disposed on top and using the current TSF (tailings management facility) and sulfur rich tailings in a separate lined compartment (PAG TSF).
- CP (Uxatjärn) and a nearby water treatment plant.
- Waste-rock dump located north to northwest of the industrial area.

20.4.4 General description of closure objectives

The closure objectives are based on the expected future land use and possible effect on receptors e.g., humans, plants, and wildlife, both in the short-term and long-term. The objectives are as follows:

- The area around the former mine site is with time to be returned to natural/forest land.
- TSF is to, with time, to become a "natural" meadow/heath and/or forest.
- The open pit and CP pond will with time become natural water bodies i.e. lakes.
- Tailings and waste-rock should not to produce acid mine drainage, thereby not affecting the surroundings including surface waters post closure.

Waste rock and tailings as well as the open pit and CP should pose no human or environmental risks.

20.4.5 Industrial area and infrastructure closure

Buildings, open spaces, and parking spaces will be decommissioned. Hard surfaces will be ripped, and a soil layer applied (0.5 m) and re-vegetated. Potential contamination and waste found through investigations will be excavated and disposed of in accordance with current legislation at the time of closure. Roads will be removed when they are no longer needed for monitoring and control purposes.

Buildings located in the industrial area will be dismantled. Equipment such as crushers, mills, etc. will be sold/disposed of. Inert material such as concrete slabs may be left after fracturing to prevent water build-up. Other underground components such as pipes may be plugged and left in place subject to check that they do not contain hazardous substances, or after decontamination.

20.4.6 Open pit closure

The mined-out parts of the pit will be used during operation to store waste-rock. During closure the open pit will be emptied of equipment, then naturally filled with groundwater and surface runoff when the pumps are turned off. Waste-rock stored in the pit will be water covered in the process, thereby limiting oxidation and environmental impact. Fencing will be carried out gradually during the operation and some re-sloping of pit walls might be done during closure in order to ensure safe access to the pit water surface.

20.4.7 TSFs and waste rock dump closure

Even though no test work has been carried out it is assumed that the de-sulfurised tailings would be classified as inert or non-acid producing and only require rehabilitation through vegetation by placing a 0.5 m layer of moraine.

No test work has been done using sulfur-rich tailings, but it is highly likely that they would be classified as hazardous waste and highly acid producing. The sulfur-rich tailings are planned to be disposed in a separate lined cell under a water cover. During closure, a high-water surface ("groundwater table") would be kept, thereby reducing infiltration of oxygen. The surface of the separate TMF compartment would then be capped using a low permeable cover system consisting of e.g., a barrier soil layer of 0.3 m of clayey moraine or bentonite amended moraine followed by 1.5 m of unsorted till to prevent the barrier soil from being affected by freezing. It is expected that natural precipitation together with the kept liner system will continue to keep the groundwater table high.

The waste rock is assumed to be acid producing, since it currently is assumed not to be feasible during operation to separate, acid producing from non-acid producing waste rock. Any surplus waste rock not placed and water covered, in the mined out open pit, will be re-sloped and covered with a low permeable cover system (0.3 m of clayey moraine or bentonite amended moraine followed by 1.5 m of unsorted till) limiting oxygen penetration and oxidation.

20.4.8 Clarification pond and water treatment system closure

The closure phase is likely to continue for a number of years after the end of operation. The old TMF has also left some residual contaminants in the groundwater underlying the TMF. It is unknown, but not unlikely, that a restart of operation would benefit the clean-up of the groundwater due to both an increased flush out as well as the ceasing of oxidation of old tailings due to the tailings being covered by fresh de-sulfurised tailings being more inert. However, due to the lack of data and investigations, it is expected that the clarification pond will still receive acidic waters for a number of years after closure and hence there is a need of keeping the water treatment plant in order to avoid the discharge of acidic and contaminated water into the small river of Älman. The produced water treatment sludge would be disposed of in the deeper parts of the mined out open pit in order to reduce transportation and disposal costs.

Adding to this the dam wall and outlet (spillway) would be rebuilt ensuring long-term stability.

Monitoring and control of discharged water would be needed for up to 30-years.

The overall target would be that the CP in time would start functioning as a natural lake, Lake Uxatjärn.

20.4.9 Monitoring, maintenance, and control

Post-closure monitoring of the mine site might be required for a time up to 30-years according to current Swedish practice after closure and rehabilitation.

A monitoring programme also has to be agreed with the appointed authority well in advance of the planned closure. Data from the monitoring program need to be evaluated continuously in order be able to identify and possibly take actions in case the closure does not fulfil the set-up requirements and target.

21 CAPITAL AND OPERATING COSTS

21.1 Cost estimate basis

The capital cost estimate (CAPEX) has been prepared using a constant money model which assumes the purchasing power does not change with time. This means the cost of CAPEX and operating costs (OPEX) are constant through time in a like-for-like manner.

The design life of project is 19-years, with the mine operating for 15-years, and the plant fed by stockpiled material in the final years.

The Cost Estimate has been compiled from supporting engineering documents and cost information derived from the following sources:

- Quotations from equipment suppliers and local service providers.
- Historical cost information sourced from in-house and commercial databases.
- Woxna Graphite derived data from existing operations.

21.1.1 Methodology

The cost estimate is based on the current level of engineering design and has been generated from supporting engineering quantities and cost information. Cost information has been derived from the following sources:

- budget quotations for the crushing plant;
- budget quotations for other plant equipment from suppliers on preliminary process data;
- historical cost information sourced from in-house and commercial databases;
- actual expenditures and budgeted figures for site refurbishment works;
- data from current and historical operations;
- factors for ancillary works have been applied to the Mechanical Equipment;
- factors for indirect costs have been applied to the total direct costs; and.
- factors are derived from the in-house database and from estimating publications.

21.1.2 Estimate classification

This Project has been deemed to be an Association for the Advancement of Cost Engineering (AACE) Class 5 estimate. As such the methodology outlined in this document shall be applied.

21.1.3 Assumptions

21.1.3.1 Processing

The following assumptions have been made during the preparation of this estimate:

- New equipment will be purchased where additional equipment is required.
- Existing equipment will be refurbished where required.
- Equipment costs are based on selected suppliers that may not necessarily be the final equipment supplier for the project.
- Equipment costs are based on information and testwork available at the time of design.
- All required earthworks materials such as fill, sand, gravel, crushed rock, etc. can be sourced within 2 km of the plant site.

21.1.3.2 Waste management

The costing has been established based on a set of unit rates established for the major cost contributors and provides accuracy to AACE Level 5 at cost +100% / -50%. The financial analysis for the construction, maintenance, monitoring and surveillance of the mine waste management facilities is based on the following assumptions:

- All costs used are in USD. The conversion rate used for financial estimate was 8.3984 SEK : 1 USD.
- Capex is based on unit rates obtained by Golder on similar projects undertaken in the recent past in the Nordic countries. The unit rates are summarised in Table 21-1.
- The costs are based on modelled and placed quantities for the following items:
  - liner;
  - rockfill;
  - moraine; and
  - filter.
- For the costs associated with foundation preparation, the following has been assumed:
  - For the NAG TSF, the foundation preparation costs are based on the footprint of the dam structure.
  - For the PAG TSF, the footprint of the dam and the tailings areas have both been considered. It is assumed that foundation preparation reaches in average 1 m of depth.
- Some items have not been designed as part of this study and their corresponding costs have been accounted for as “unmeasured items”. These are as follows:
  - spillway construction;
  - access Road Construction;
  - tailings and water pipe delivery system;
  - reclaim water system, including barge and pumps; and
  - installation of monitoring instrumentation.
- The unmeasured items have been estimated by Golder to represent 30% of the total construction costs for the Project and have been included for every cost presented.

Table 21-1: Unit rates

Item	Unit rate	Unit	Unit rate	Unit
Foundation preparation	5.95	USD/m^2	50	SEK/m^2
Liner	9.53	USD/m^2	80	SEK/m^2
Rockfill	9.53	USD/m^3	80	SEK/m^3
Filter	23.81	USD/m^3	200	SEK/m^3
Moraine	11.91	USD/m^3	100	SEK/m^3

21.1.4 Currency and exchange rates

Capital and operating cost estimates are prepared in mixed currencies and reported in United States dollars (USD). The exchange rates used in calculations are shown in Table 21-2.

Table 21-2: Currency and exchange rates[5]

Currency	Code	Rate
United States dollar	USD	1.0000
Pound sterling	GBP	0.7317
Euro	EUR	0.8309

[5] Exchange rates are for a base date of 3rd February 2021.

Currency	Code	Rate
Swedish Krona	SEK	8.3984
Australian dollar	AUD	1.3119
Canadian dollar	CAD	1.2775
South African rand	ZAR	14.9294

21.1.5 Base date and reporting currency

This cost estimate has an effective date of the 9th June 2021.

The estimate is reported in USD.

21.1.6 Exclusions

No allowances have been made for:

- cost escalation;
- currency fluctuations;
- container demurrage costs;
- product transportation or transportation security;
- required permits, licences or legal and administrative costs associated with government mining and environmental regulations. This includes reporting requirements during operation and related administrative costs;
- monitoring during operation and post closure;
- insurance;
- reconciliation of the CAPEX or OPEX estimates; and
- management reserve.

21.1.7 Contingency

Contingency is an amount added to an estimate to allow for items, conditions, or events for which the state, occurrence, or effect is uncertain and that experience shows will likely result, in aggregate, in additional costs. Typically estimated using statistical analysis or judgment based on past asset or project experience.

Contingency usually includes: as yet undefined items needed to complete the current project scope; quantity variations; estimating errors and omissions; budget-pricing variances for materials and equipment; variability in market conditions, wage rates, labour head count, labour productivity, construction schedules, and project execution parameters.

As contingency is based purely on the specific project scope, and by definition it cannot cover out-of-scope items or changes.

The contingency estimate for the Project is broken down into the risk drivers identified as follows:

- project definition;
- estimation methods and estimation data;
- engineering design aspects;
- supplier quotations or database costs; and.
- site data and testwork.

Contingency has been estimated using a factored method which is suitable for an AACE Class 5 cost estimate. As a result of the estimating techniques employed, a 20% contingency has been applied against direct costs.

21.2 Capital cost estimate

21.2.1 Capital cost summary

The initial and LoP capital expenditure is summarised in Table 21-3.

Table 21-3: CAPEX Estimate – Initial and LoP

Cost Centre	Initial CAPEX [USD]	LoP CAPEX [USD]
Mining	2,029,500	2,029,500
Buildings	1,689,478	1,689,478
Earthworks	2,337,400	7,007,000
Civil	-	-
Concrete	334,952	334,952
Steelwork	328,417	328,417
Mechanical	69,354,246	69,354,246
Mechanical Installation	7,880,600	7,880,600
Mobile Equipment	133,390	133,390
Electrical	3,773,693	3,773,693
Control and Instrumentation	1,022,662	1,022,662
Piping	2,110,985	2,110,985
Platework	581,168	581,168
Insurance Spares	-	-
DIRECT TOTAL (D)	**89,546,992**	**96,246,092**
Engineering, Procurement & Construction Management (EPCM)	5,000,000	5,000,000
Construction Facilities	696,188	696,188
Commissioning	882,143	882,143
Field Indirect	-	-
Owners Cost	654,890	7,832,924
Contingency	17,441,918	17,441,918
INDIRECT TOTAL (I)	**24,675,140**	**31,853,173**
FIXED CAPITAL TOTAL (F = D + I)	**114,222,131**	**128,099,265**
Cash Reserves	-	-
Inventory	4,858,455	4,457,145
WORKING CAPITAL TOTAL (W)	**4,858,455**	**4,858,455**
TOTAL CAPITAL INVESTMENT (F + W)	**121,110,086**	**132,957,720**

The initial capital expenditure is summarised by area in Table 21-4.

Table 21-4: Initial CAPEX – Area Breakdown

Description	Initial CAPEX [USD]
Direct Costs	
100 Mine	2,029,500
200 Concentrator	133,390
210 Crushing	1,749,241
220 Rod mill circuit (with flash float)	728,333
230 Rougher flotation circuit (with regrind)	525,835
235 Cleaner flotation circuits (with regrind)	4,135,849
240 Filtration	647,554
245 Drying	784,196
270 Air	141,089
280 Water	913,422
300 Upgrade Plant	1,560,000
320 Spheronizing	25,939,572
330 Jet Milling	3,093,430
340 Thermal treatment	28,064,816
350 Air	1,039,432
370 Spheroid Coating	17,590,755
420 Final Product Packaging	162,676
500 Waste Management / Tailings	2,337,400
Direct Total (D)	**91,576,492**
Indirect Costs	
Commissioning	882,143
Construction Facilities	696,188
Contingency	17,441,918
EPCM	5,000,000
Owners Cost	654,890
Indirect Total (I)	**24,675,140**
Working	
Cash Reserve	-
Management Reserve	-
Critical and Insurance Spares	2,723,215
Operating Spares and Consumables	2,135,239
Working Total (W)	**4,858,455**
Grand Total (D + I + W)	**121,110,086**

21.2.2 Sustaining capital

21.2.2.1 Mining

Sustaining capital for mining has not been estimated as it is included in contract mining OPEX.

21.2.2.2 Processing

There is no sustaining capital scheduled for the process plant. All required works are included in OPEX maintenance costs.

21.2.2.3 Tailings

Sustaining capital for the tailings management facility are presented in Table 21-5 and Figure 21-1 below. These costs are primarily related to the earthworks to increase the dam walls.

Table 21-5: TSF sustaining capital (USD)

Year	West Dam	North Dam	East Dam	PAG	Clarification pond	Total (USD)
1	273,000	26,000	-	-	130,000	**429,000**
6	676,000	91,000	858,000	-	-	**1,625,000**
9	-	-	-	910,000	-	**910,000**
12	936,000	182,000	845,000	-	-	**1,963,000**

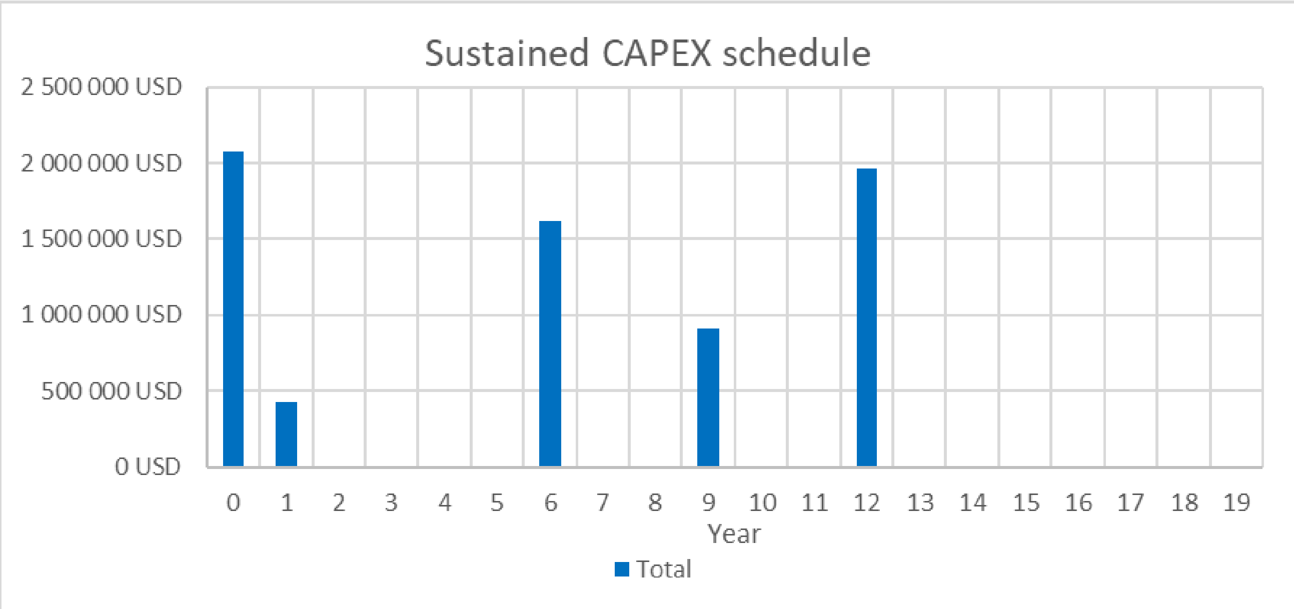


Figure 21-1: TSF Sustaining CAPEX schedule

The total estimated CAPEX for the TSF over the LoP is therefore approximately USD 6.9 million over the LoP.

21.2.2.4 Mine closure

The mine closure CAPEX is estimated to be USD 6.0 million. Although the cost for closure is realised after year 19, the sustaining capital costs have been spread over each operating year of the mine in the form of closure fund contributions. These contributions are USD 377,800 per annum for the 19 years of mine operations (see Section 21.2.8).

21.2.3 Indirect Costs

The costs factors in Table 21-6 below, have been applied to the direct costs to generate the indirect costs.

Table 21-6: Indirect costs and factors

Cost Centre	Factor	Comments
EPCM	-	Estimated – an hours allowance for high level design and supervision assistance is included
Owner's costs	-	Estimated by Woxna. Permitting and closure costs estimated by Golder.
Construction Facilities	1%	Considers brownfield site
Critical and Insurance Spares	5%	Percentages of mechanical equipment costs
Commissioning spares	1%	Percentages of mechanical equipment costs
Operating Spares and Consumables	-	Estimated, 3-months operating spares.
Cash reserves	-	Not estimated
Management reserves	-	Not estimated
Insurance	0%	Assumed
Contingency	20%	Section 21.1.10

21.2.3.1 Freight forwarding

Freight forwarding costs has been estimated based on available information and/or factoring of direct costs as appropriate.

21.2.3.2 Insurance and operating spares

An allowance for spares is included within the cost estimate.

Critical and Insurance spares are major items and parts kept on hand to ensure the uninterrupted operation of production equipment if there is an unexpected breakdown or equipment failure. They do not include items that are generally consumed or replaced during the regular maintenance cycle.

Operational (maintenance) spares are the spares required for use during the routine maintenance schedule of the process plant. The first 3-months spares are included in the capital cost estimate, as they are required to initially stock the inventory. The OPEX cost for maintenance going forward has been used to replace these as they are consumed.

21.2.3.3 Engineering procurement and construction management (EPCM)

EPCM will be estimated as a percentage of direct costs to cover the costs of basic and detailed engineering, procurement, construction management and commissioning contractor services.

21.2.3.4 Construction field indirect costs

All field indirect costs are associated but not directly related to the capital build of the Project and cannot tangibly be identified or directly attributed to an item of the work breakdown structure in the Project. Items that may be considered as construction field indirect costs include:

- Supervision
- field engineering;
- construction supplies;

- construction equipment;
- temporary access to site during construction; and
- temporary buildings.

21.2.4 Working capital

Working capital is the funds, in addition to fixed capital, which is required to get the Project started and meet subsequent obligations as they come due. Working capital includes inventories, cash, and accounts receivable minus accounts payable. Characteristically, these funds can be converted readily into cash. Working capital is normally assumed recovered at the end of the Project.

Only the working capital for inventories have been estimated for this PEA.

21.2.5 Capital cost estimate – Mining

As the mining is to be contracted, the contractor supplied mobile and fixed plant are addressed through the mine operating cost estimates. It is assumed that the existing mining infrastructure is adequate to execute the mine plan. As such no capital costs for mining have been estimated for this PEA.

21.2.6 Capital cost estimate – Processing

Direct costs for each processing component have been determined and included in the CAPEX. The direct costs for the concentrator and upgrade plant include:

- all labour required for Project construction and management activities (excluding EPCM cost);
- all material and equipment required for construction;
- mechanical, electrical, control, instrumentation, civil works, earthworks, and piping installation services;
- transport and freighting services;
- insurance and capital spares; and
- construction and installation.

21.2.6.1 Earthworks

Bulk earthworks are assumed to be minimal for the process plant areas of the Project. The upgrade works to both the main flotation concentrator plant, and the thermal upgrade plant, do not require earthworks.

21.2.6.2 Buildings

Refurbishment of plant buildings of the thermal upgrade facility has been estimated using per-square-metre rates. These rates include refurbishment of concrete floors; cladding and insulation; lighting and electricals; fixtures; additional ventilation, and removal of internal walls as required for the new plant.

21.2.6.3 Mechanical equipment

The primary equipment packages have been estimated based on vendor budgetary quotations.

The secondary and tertiary equipment have been factored from the primary equipment costs.

Mechanical installation costs have been based on a factoring of the mechanical equipment costs, freighted. The factor used have been benchmarked against similar projects.

21.2.6.4 Mobile equipment

Mobile equipment has been estimated using Zenito’s in-house cost database.

21.2.6.5 Discipline engineering

The following engineering works have been factored into the estimate from the mechanical equipment costs:

- concrete works;
- steelworks;
- electrical;
- control and instrumentation;
- piping; and
- platework.

An enhanced factoring method have been applied, whereby the factors used for each area have been benchmarked against similar projects and areas in Zenito's in-house database. These factors will cover the supply of all necessary materials, equipment, and labour for the completion of the works.

21.2.7 Capital cost estimate – Waste management

The initial capital expenditure for the TSF is presented in Table 21-7.

Table 21-7: Initial CAPEX (USD)

Year	West Dam	North Dam	East Dam	PAG	Clarification pond	Total
0	156,000	-	312,000	1,612,000	-	**2,080,000**

21.2.8 Capital cost estimate – Environmental permitting of Mine Site and VAP

The initial capital expenditure for the environmental permitting of the Mine Site and VAP facility in Edsbyn is presented in Table 21-9.

Table 21-8: CAPEX - Environmental Permit Mine Site and VAP facility

EIA related material	Estimated cost (SEK)	Comment
Site localisation study	100,000	
Waste rock and tailings characterisation –	500,000	Update
Hydrogeological investigation	1,500,000	Incl installations
Surface water chemistry	400,000	More elements and filtered samples
Air and dust	250,000	Refinery in Edsbyn
Noise and vibration assessment	300,000	Includes transports in Edsbyn
Status report	250,000	
Waste management plan	200,000	Update
Closure plan	200,000	Final conceptual plan incl. bond calculation
Impact classification of dams ("GruvRidas")	200,000	Update if needed
Consultation document	150,000	To be used during the consultation work
Consultation (meetings and correspondence)	150,000	
IM/GIS for illustrations and assessment	200,000	
EIA report (main document)	650,000	
Additional information/data upon request from authorities/stakeholders	300,000	
Court hearings (5 days)	150,000	Three environmental consultants
TOTAL	**5,500,000**	

21.2.9 Capital cost estimate – Closure

Financial assurance is a key component for a mine closure. It provides a level of certainty to regulatory authorities and communities that the financial resources exist when needed i.e., in case of a bankruptcy. The closure estimations in this conceptual plan rely on several assumptions as well as local experience and local contractor unit estimates. It has further been assumed that natural available moraine exists close to the mine.

The closure costs (CAPEX) have been estimated to a total of USD 6.0 m, including a 10% contingency.

The closure costs are incurred at the end of mine life, and as such are not included in the initial CAPEX estimate. The closure costs have been spread over the life of mine for the financial economic analysis.

Table 21-9: Mine closure CAPEX costs

Breakdown	Description	Cost (USD)
Infrastructure		**888,071**
	Crushing of concrete slabs	148,654
	De-installation of equipment (other)	-
	De-installation of equipment (processing plant)	41,675
	Demolishing of buildings	125,024
	Excavation, transport of contaminated surface soil (E.g. oil spill)	54,365
	Landfilling cost	114,676
	Rehabilitation of surfaces	316,609
	Removal of roads	-
	Vegetation/Hydroseeding	87,068
Open Pit		**66,823**
	Fencing	66,823
Main Tailings Impoundment		**1,568,229**
	Breaching of wall/drain	-
	Cover layer	1,217,609
	De-installation of pumps	11,907
	De-installation of tailings and water pipes	3,870
	Seeding of grass	334,843
PAG Tailings Impoundment		**958,030**
	Breaching of wall/drain	-
	Cover layer	888,240
	De-installation of pumps	-
	De-installation of tailings and water pipes	-
	Seeding of grass	69,790
Waste rock dump		**1,618,946**
	Cover layer	1,501,010
	Seeding of grass	117,936
Clarification Pond		**-**
	Clarification Pond	-
Permitting and Approval of Mine Closure		**274,339**
	Approval and Permitting (EIA, Technical Description and supporting documents)	68,585
	Final Mine Closure Plan - Detailed Design	68,585
	Supporting studies (geohydrology, environment, detailed design)	137,170
Project and Construction Management		**68,585**
	Project and Construction Management	68,585
Contingency		**544,302**
	Contingency	544,302

Breakdown	Description	Cost (USD)
Grand Total		**5,987,325**

21.3 Operating cost estimate

The following sections summarise and describe the development of the various operating cost components. The overall OPEX is provided in Table 21-10.

Table 21-10: Overall Operating Expense (OPEX) Estimate

Description	USD/a	USD/t ROM	USD/t Graphite Product
General & Administration	325,725	2.04	21.50
Mining	3,630,426	22.69	239.63
Processing	20,816,104	130.10	1,374.00
Waste	312,534	1.95	20.63
Grand Total	**25,084,788**	**156.78**	**1,655.76**

The overall Project product is a combination of both CSPG and micronized graphite.

The overall OPEX figures are presented assuming a nameplate production year. This means assuming a plant feed of 160 ktpa, an average strip ratio of 3.7 for the first 15-years of mining (thereby avoiding start up and shutdown irregularities), and that annual payments through the LoP to accumulate the closure fund are incorporated.

21.3.1 General and administration

The annual general administration costs including the process plant, mine and ancillaries were based on predominantly Woxna Graphite input, which are heavily influenced by current and historical operational costs.

The general administration (GA) cost is applied at a fixed rate of USD 325,725 pa, regardless of the feed tonnage of the process plant, or amount of product produced.

21.3.2 Mining

Mining OPEX costs presented in Table 21-11 below have been estimated and supplied by MPlan. These costs have been benchmarked against similar contract mining operations in Sweden.

Table 21-11: OPEX costs – Mining

Cost section	Rate	Unit
Ore mining cost	4.51	USD/t RoM
Waste mining cost	4.51	USD/t RoM
Overhead mining cost	122,322	USD/a

21.3.3 Processing

Table 21-12 below contains a summary of all the processing operating costs for a typical full production year.

Table 21-12: OPEX summary – Processing

Cost section	USD/a	USD/t ROM	USD/t Product
Consumables	6,848,246	42.80	452.03

Cost section	USD/a	USD/t ROM	USD/t Product
Labour	4,874,249	30.46	321.73
Maintenance & Operating Spares	4,357,956	27.24	287.65
Mobile Equipment	144,394	0.90	9.53
Reagents	1,454,122	9.09	95.98
Utilities	3,137,137	19.61	207.07
Total	**20,816,104**	**130.10**	**1,374.00**

Table 21-13 below contains a summary of the operating costs, for the concentrator facility, for a typical full production year.

Table 21-13: OPEX summary – Concentrator facility

Cost section	USD/a	USD/t ROM	USD/t Product
Consumables	187,513	1.17	12.38
Labour	2,827,670	17.67	186.64
Maintenance & Operating Spares	918,429	5.74	60.62
Mobile Equipment	108,615	0.68	7.17
Reagents	87,784	0.55	5.79
Utilities	829,648	5.19	54.76
Total	**4,959,659**	**31.00**	**327.37**

Table 21-14 below contains a summary of the operating costs, for the VAP- thermal upgrade facility, for a typical full production year.

Table 21-14: OPEX summary – VAP-thermal upgrade facility

Cost section	USD/a	USD/t ROM	USD/t Product
Consumables	6,660,733	41.63	439.65
Labour	2,046,579	12.79	135.09
Maintenance & Operating Spares	3,439,527	21.50	227.03
Mobile Equipment	35,779	0.22	2.36
Utilities	1,366,338	8.54	90.19
Total	**15,856,445**	**99.10**	**1,046.63**

21.3.3.1 Operating spares and maintenance

Operating and maintenance spares have been calculated based on the estimated capital cost for existing and new mechanical equipment. The calculated costs consider the type of equipment and their operating duties.

Table 21-16 below contains a summary of the operating spares and maintenance costs for a typical full production year.

Table 21-15: Operating costs - Spares and maintenance

Area	Cost USD/annum
200 Concentrator	446,516
210 Crushing	165,069
220 Rod mill circuit (with flash float)	39,449
230 Rougher flotation circuit (with regrind)	47,282

Area	Cost USD/annum
235 Cleaner flotation circuits (with regrind)	151,787
240 Filtration	15,847
245 Drying	28,878
270 Air	2,664
280 Water	20,938
320 Spheronizing	2,304,034
330 Jet Milling	89,071
340 Thermal treatment	517,578
350 Air	34,403
370 Spheroid Coating	492,102
420 Final Product Packaging	2,339
Total	**4,357,956**

21.3.3.2 Mobile equipment

Mobile equipment operating costs have been calculated for each set of equipment and the estimated annual operating hours. The costs have allowance for: fuel consumption; wear parts; lubricants; tyres; as well and general and overhead costs.

Table 21-16 below contains a summary of the mobile equipment running costs for a typical full production year.

Table 21-16: Operating costs - Mobile equipment

Area	Cost USD/annum
200 Concentrator	11,599
210 Crushing	97,016
300 Anode Plant	35,779
Total	**119,791**

21.3.3.3 Reagents

Reagent consumption rates for the process plant are taken from the mass balance, which has been calculated using historical operating data, available test work or industry norms for a plant of this type and size. The annual reagent consumption costs have been determined using calculated consumption rates and prices per tonne of reagent based on database prices.

The primary reagents are diesel and MIBC for the flotation, and flocculant for the water supply clarifier.

Table 21-17 below contains a summary of the reagent costs for a typical full production year.

Table 21-17: Operating costs - Reagents

Area	Cost USD/annum
230 Rougher flotation circuit (with regrind)	86,884
280 Water	900
370 Product Coating	1,366,338
Total	**1,454,122**

21.3.3.4 Labour

The required personnel have been estimated by Woxna Graphite in conjunction with Zenito. The personnel plan employs staff for most roles, supplemented by contractors for electrical, instrumentation and mechanical maintenance. The required personnel for the concentrator is based upon the historical personnel requirements when the plant was in operation.

The rates for staff and contractors are based on historical figures when the processing plant was in operation, with allowance for inflation.

Table 21-18 below contains a summary of the labour costs for a typical full production year.

Table 21-18: Operating costs - Labour

Area	Cost USD/annum
000 General	1,511,057
200 Concentrator	1,316,613
300 Anode Plant	2,046,579
Total	**4,874,249**

21.3.3.5 Consumables

Consumption of consumables have been calculated based upon available testwork, supplier estimates, and historical operations of the existing plant. Rates for consumables have been estimated by suppliers or by using historical rates where available for the existing plant.

The primary consumables are:

- grinding media for the rod and regrind mills;
- argon and nitrogen shielding gas for the thermal treatment furnaces; and
- packaging for the final products.

Annual grinding media consumption is estimated using the rate of consumption in kilograms per tonne, and the cost per tonne of grinding media. The rate of rod consumption was provided by Aminpro (metallurgical test work provider) and compared with benchmarked rates.

Ceramic regrind media consumption and costs were supplied by BGRMM.

Furnace crucible consumption and rates were supplied by the furnace supplier.

Annual costs of the liners for the mills have been calculated using the wear rates for each mill liner in grams per tonne. These rates have been sourced from database figures. This rate is applied to the plant feed tonnage and multiplied by the cost of liners.

Table 21-19 below contains a summary of the consumable costs for a typical full production year.

Table 21-19: Operating costs - Consumables

Area	Cost USD/annum
220 Rod mill circuit	30,400
235 Cleaner flotation circuits (with regrind)	157,113
340 Thermal treatment	3,143,575
370 Product Coating	3,177,109
400 Product Handling	51,422
420 Final Product Packaging	288,628
Total	**6,848,246**

21.3.3.6 Electrical power

For practical reasons, the entire site electrical power consumption is based on the process plant load that represents typically 95% of the energy consumption. The electricity consumption is based on the connected and running power (kW) for the entire site as per the loads derived from the mechanical equipment list.

Electricity costs in Sweden include a grid connection fee of 614.5 SEK/kW/a, which is charged against the connected load. The consumption price for 2021 has been quoted at 0.173 SEK/kWh.

The average electrical demand is 3.3 MW for the concentrator, and 9.6 MW for the thermal upgrade plant.

The annual electrical power costs are presented in Table 21-20 below.

Table 21-20: Operating costs – Electrical power

Area	Cost USD/annum
200 Concentrator	577,810
245 Drying	251,838
320 Spheronizing	308,128
330 Jet Milling	52,357
340 Thermal treatment	1,315,181
350 Air	205,419
370 Product Coating	426,405
Total	**3,137,137**

21.3.4 Waste management

Operating expenditure for the tailings management facility has been estimated for the duration of the LoP and include the following items:

- owners costs (project management);
- infrastructure mechanical and civil maintenance (roads, pipes, pumps etc), including consumables, relocation, etc;
- power costs; and
- design, construction supervision and monitoring activities.

Based on comparable projects in the Nordic regions, operating costs can be estimated per tonne of tailings stored. This range is from SEK2.5 /t to SEK7/t, i.e. USD 0.3 /t and USD 0.8 /t. The variation in operating costs is mostly due to the following:

- site location;
- access to infrastructures and labour;
- size of the operation; and
- complexity of design and operation.

Based on this, it is estimated that the Project will be towards the high range of the estimation, with tailings requiring two separate deposition facilities, deposition, and reclaim water systems, and being a relatively small operation. Operating costs have therefore been estimated to USD 0.7/t, which represents approximately USD 110,000 pa over the 19-years of LoP.

21.3.5 Mine closure

Table 21-21: Closure OPEX costs

Breakdown	Description	Cost USD/annum
Year 1 after closure		**203,456**
	Annual Reporting	5,715
	Approval of Monitoring Programme	5,715
	Assays	4,768
	Contingency 15%	18,772
	Installation of groundwater monitoring wells	20,837
	Installation of monitoring equipment (weirs etc.)	59,535
	Maintenance of Equipment	-
	Maintenance of infrastructure	-
	Monitoring Programme - Set-up	5,715
	Sampling/Field Work	22,862
	Water Treatment OPEX	59,535
Year 2-30 after closure		**125,679**
	Annual Reporting	5,715
	Assays	4,768
	Contingency 15%	8,627
	Maintenance of Equipment	5,954
	Maintenance of infrastructure	18,218
	Sampling/Field Work	22,862
	Water treatment OPEX	59,535

22 ECONOMIC ANALYSIS

This report contains certain "forward-looking statements" and "forward-looking information" as defined under applicable Canadian and U.S. securities laws. Forward-looking statements are based on forecasts of future results, estimates of amounts not yet determinable and assumptions that, while believed by management and the consultant to be reasonable, are inherently subject to significant business, economic and competitive uncertainties, and contingencies.

The base case financial model prepared is a constant money model which assumes the purchasing power does not change with time. This means the cost of capital, operating costs and revenue are constant through time in a like-for-like manner. A scenario analysis is undertaken to assess the impact of time varying changes cost and revenue drivers.

22.1 Analysis criteria

Table 22-1: Depreciation and loss parameters

Parameter	Method	
Asset depreciation method	straight-line	
Depreciation carry-forward	infinite	
Operating loss carry-forward	Infinite	
Parameter	**Value**	**Unit**
Fixed Plant, Equipment & Infrastructure	20	% per annum
Asset End-of-Life Salvage Value	0	USD

Table 22-2: Base case financial inputs

Description	Value	Unit	Reference
Corporation tax rate	20.6	%	Client
Discount rate	8	%	Client
Finance rate	0	%	A
Royalty rate	0.2	%	Client
Base currency code	USD	-	Client
Price Spheronized Coated Graphite[1]	10,000	USD/t	Benchmark Mineral Intelligence
Price Micronized Graphite[1]	1,200	USD/t	Client

1. Effective date 9th June 2021

Zenito believes the depreciation and taxation methods that are applied to this PEA and economic assessment are appropriate. Zenito however is not a professional accounting practice, and the post-tax economic performance should be taken as a guide.

22.1.1 Carried forward depreciation and losses

As Woxna Graphite have existing infrastructure and have operated historically, some depreciation carries forward and historical losses have been incorporated into the economic model as summarised in Table 22-3 below.

Table 22-3: Historical book assets

Item	Value	Unit
Operating Loss	8.10	M USD
Total	**8.10**	**M USD**

22.2 Economic performance

Table 22-4 below presents the results of economic modelling for the production schedule under the CAPEX and OPEX regimes presented herein, and the economic parameters as described in the relevant sections.

The preliminary economic assessment is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Table 22-4: Economic summary

Parameter	Value	Unit
Life of Project	19	years production
Initial Capital Investment	121,110,086	USD
Life of Mine Capital Investment	132,957,720	USD
Price CSPG	10,000	USD/t Product
Price Micronized Graphite	1,200	USD/t Product
Revenue	480	USD/t ROM
	5,326	USD/t Product
Cost of goods sold	165	USD/t ROM
	1,828	USD/t Product
Gross Margin	316	USD/t ROM
	3,499	USD/t Product
	65.7%	
Accumulated Project Revenues	1,425	million USD
Accumulated Project EBITDA	936	million USD
Average Annual Revenue	75	million USD
Average Annual EBITDA	49	million USD
Pre-Tax		
Pre-Tax Internal Rate of Return (IRR)	42.9%	
NPV@8%DR	317,031,929	USD
Total Payback Period	4.24	Years (from initial investment)
Payback after first production	2.24	Years (from first production)
Post-Tax		
Post-Tax IRR	37.4%	
NPV @8%DR	248,167,633	USD
Total Payback Period	4.51	Years (from initial investment)
Payback after first production	2.51	Years (from first production)

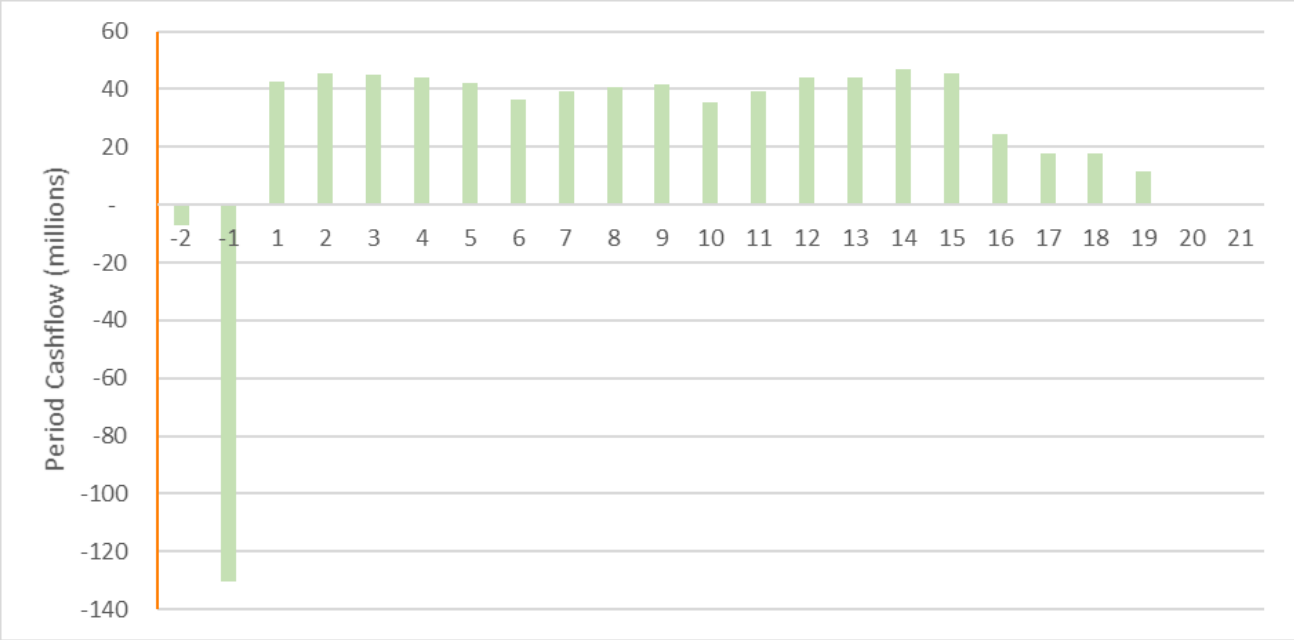


Figure 22-1: Post-tax annual pre-finance cash flows

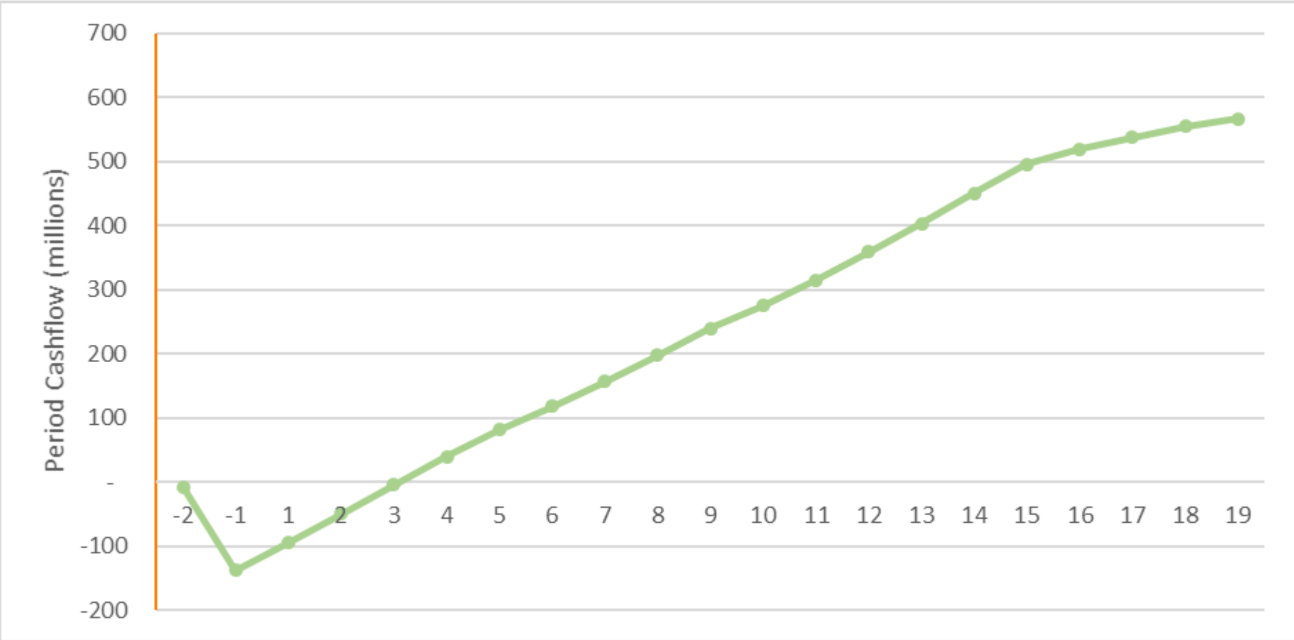


Figure 22-2: Post-tax cumulative pre-finance cash flows

22.3 Financial model

The Table 22-5 below presents the results of the financial modelling for the production schedule under the capital/operating cost and, economic parameters as described in the relevant sections.

Table 22-5: Discounted cash flow analysis (M USD)

Item	LoP Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
CAPEX	132.96	7.53	113.58	0.55	0.38	0.38	0.38	1.35	1.03	0.38	0.92	0.74	0.38	1.56	1.16	0.38	0.38	0.38	0.38	0.38	0.38	0.38
OPEX	486.55	-	-	30.54	29.01	27.51	28.05	27.79	27.68	28.26	29.08	29.22	27.34	28.28	29.41	27.95	28.04	27.11	18.40	15.83	15.83	11.23
Revenue	1,254.70	-	-	74.62	74.37	71.82	71.23	69.88	69.65	72.06	75.84	77.05	67.02	73.60	80.13	77.93	81.27	78.75	46.65	36.45	36.45	19.94
Sales	1,425.24	-	-	84.76	84.48	81.58	80.91	79.38	79.12	81.85	86.15	87.52	76.12	83.60	91.02	88.52	92.31	89.45	52.99	41.41	41.41	22.65
Cost of Sales	489.06	-	-	30.69	29.16	27.66	28.19	27.93	27.82	28.40	29.23	29.37	27.47	28.43	29.57	28.11	28.21	27.26	18.49	15.90	15.90	11.27
Annual Operating Income	936.18	-	-	54.07	55.32	53.92	52.72	51.45	51.30	53.45	56.92	58.15	48.65	55.17	61.45	60.42	64.11	62.19	34.50	25.50	25.50	11.37
Annual Depreciation Claimed	125.65	-	-	22.83	22.91	22.99	23.02	23.10	0.65	0.69	0.69	0.71	0.73	0.73	0.79	0.83	0.83	0.83	0.83	0.83	0.83	0.83
Annual depreciation allowance	-	-	-	22.83	22.91	22.99	23.02	23.10	0.65	0.69	0.69	0.71	0.73	0.73	0.79	0.83	0.83	0.83	0.83	0.83	0.83	0.83
Depreciation carry forward	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Working capital take out	-4.86	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-4.86
Taxable Operating Income	815.38	-	-	31.24	32.40	30.93	29.70	28.36	50.65	52.76	56.24	57.43	47.92	54.44	60.66	59.59	63.28	61.36	33.67	24.67	24.67	15.40
Taxes	166.30	-	-	4.77	6.68	6.37	6.12	5.84	10.43	10.87	11.58	11.83	9.87	11.21	12.50	12.27	13.04	12.64	6.94	5.08	5.08	3.17
Corporations Tax	166.30	-	-	4.77	6.68	6.37	6.12	5.84	10.43	10.87	11.58	11.83	9.87	11.21	12.50	12.27	13.04	12.64	6.94	5.08	5.08	3.17
Loss Carry Forward	-8.10	-8.10	-8.10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Loss Carry Forward Claimed	-	-	-	8.10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Annual Post Tax Income	649.08	-	-	26.48	25.73	24.56	23.58	22.51	40.22	41.89	44.65	45.60	38.05	43.23	48.16	47.31	50.24	48.72	26.73	19.59	19.59	12.23
Annual Cash Income	774.73	-	-	49.31	48.64	47.55	46.60	45.61	40.87	42.58	45.34	46.32	38.78	43.96	48.95	48.14	51.07	49.55	27.56	20.42	20.42	13.06
Annual Cash Flow	641.78	-7.53	-113.58	48.76	48.27	47.17	46.22	44.26	39.84	42.20	44.41	45.57	38.40	42.40	47.79	47.76	50.70	49.17	27.18	20.04	20.04	12.68
Cumulative Cash Flow	-	-7.53	-121.11	-72.35	-24.09	23.08	69.31	113.57	153.41	195.61	240.02	285.60	324.00	366.40	414.19	461.96	512.65	561.82	589.01	609.05	629.09	641.78

22.4 Sensitivity analyses

A sensitivity analysis has been carried out, with the project base case as defined as a starting point, to assess the impact of changes in total pre-production CAPEX costs, OPEX costs, and the product price, on the Project's NPV @ 8.0% and IRR. Each variable was examined individually. An interval of ±30% with increments of 10% was used.

Table 22-6: Pre-tax sensitivities

CSPG price per tonne	USD 7,000	USD 8,500	USD 10,000	USD 11,500	USD 13,000
NPV(8%)	M USD 146	M USD 231	M USD 317	M USD 403	M USD 489
IRR	25.1%	34.1%	42.9%	51.5%	60.0%

Table 22-7: Post-tax sensitivities

CSPG price per tonne	USD 7,000	USD 8,500	USD 10,000	USD 11,500	USD 13,000
NPV(8%)	M USD 112	M USD 180	M USD 248	M USD 257	M USD 384
IRR	22.5%	30.1%	37.4%	44.5%	51.5%

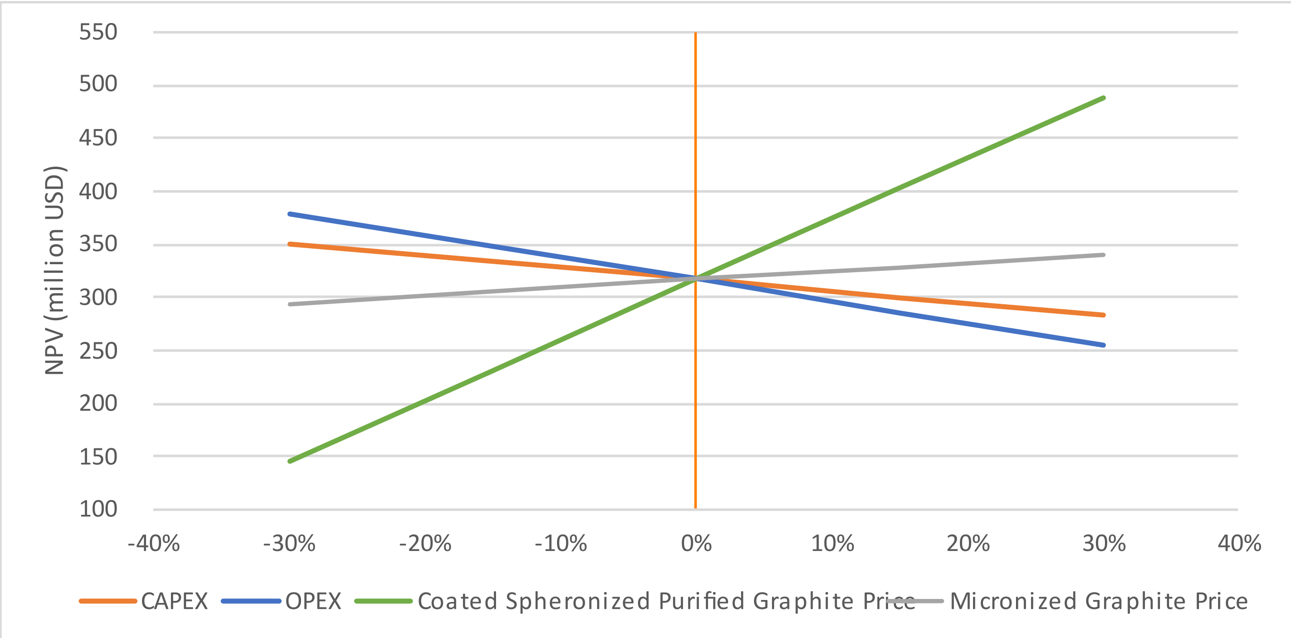


Figure 22-3: Pre-tax NPV sensitivity (@8% DR)

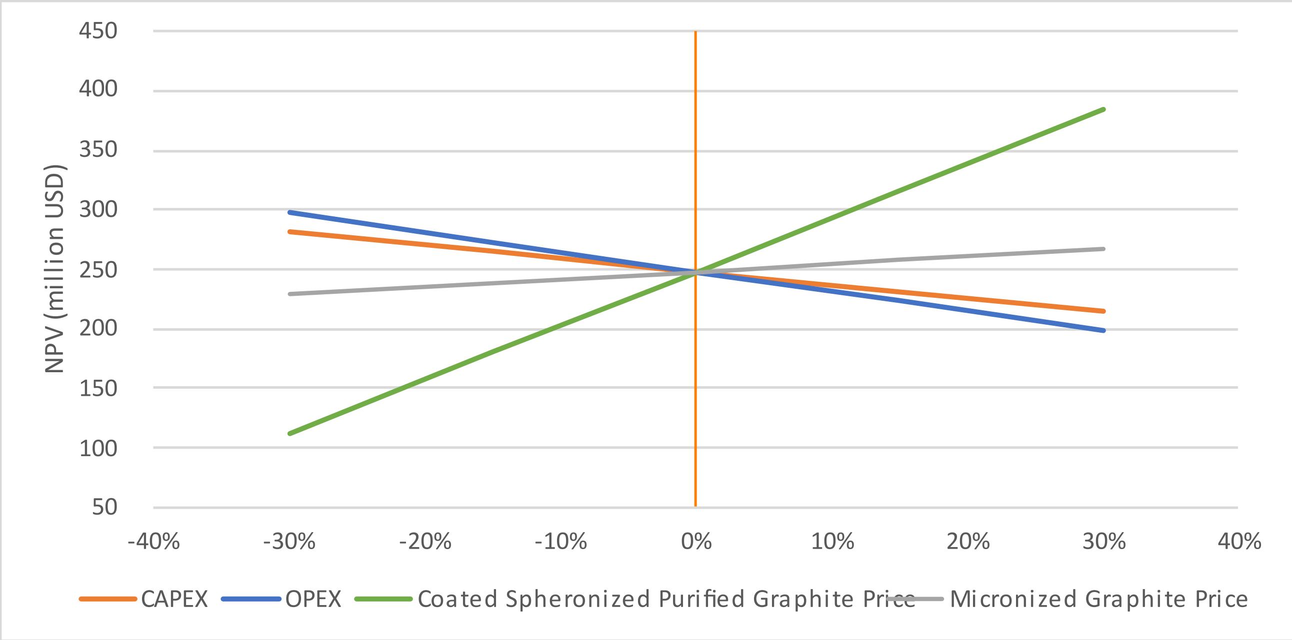


Figure 22-4: Post-tax NPV sensitivity (@8% DR)

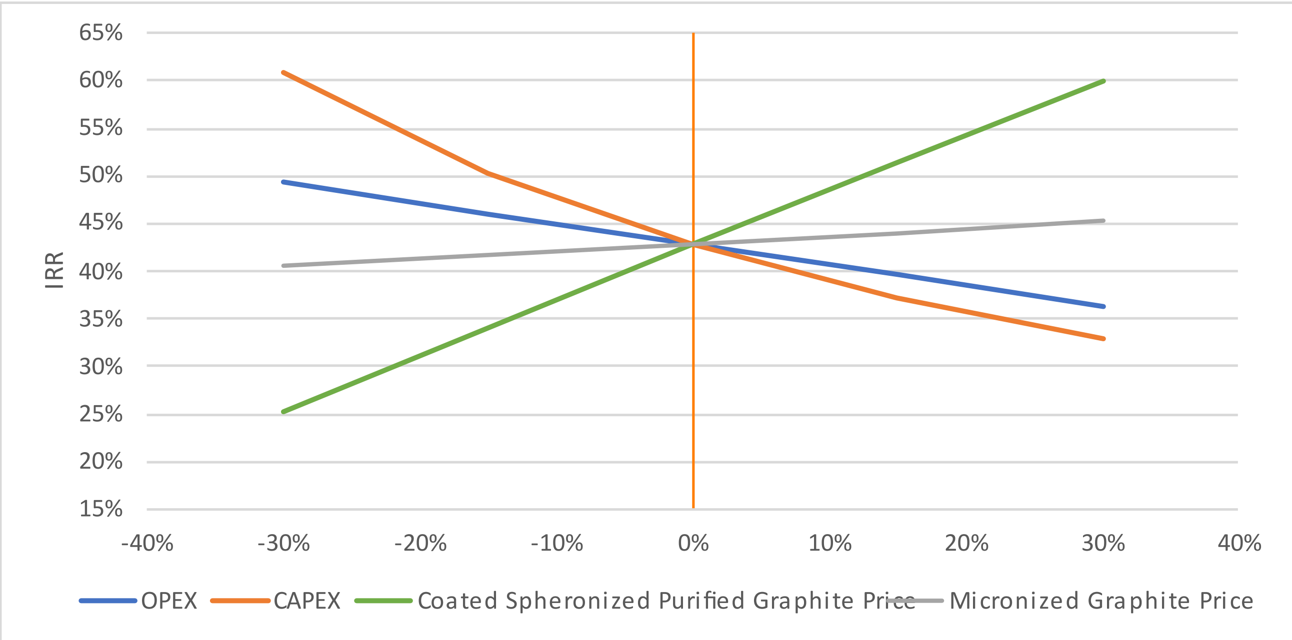


Figure 22-5: Pre-tax IRR sensitivity

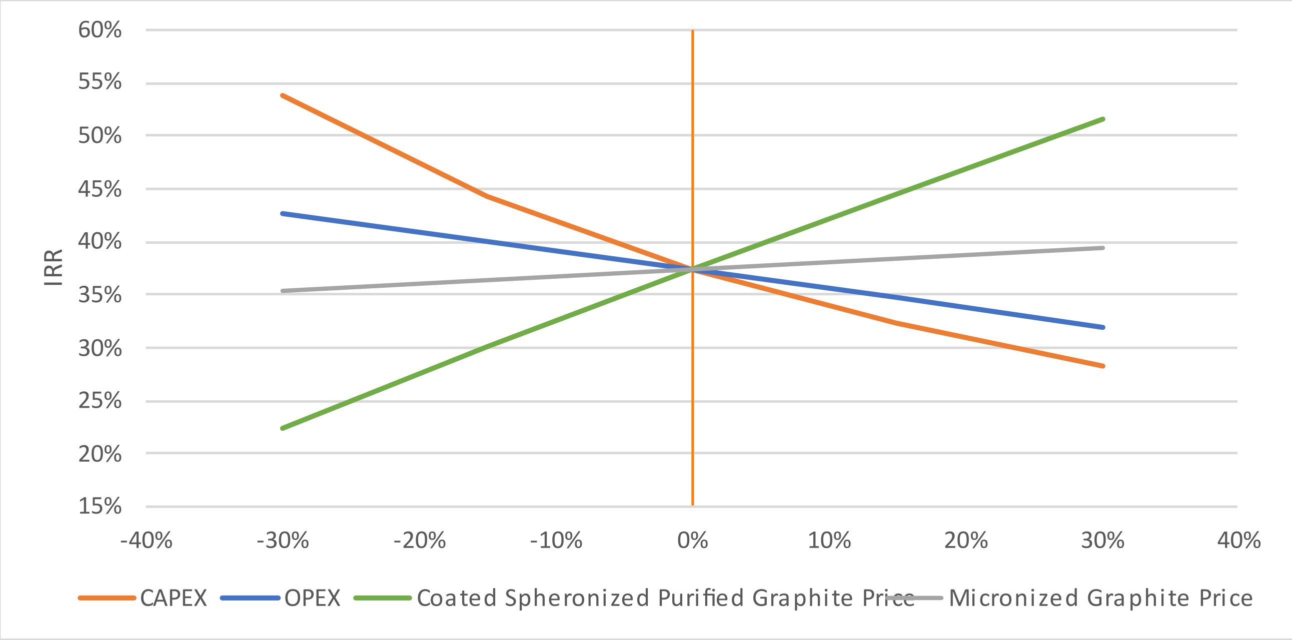


Figure 22-6: Post-tax IRR sensitivity

23 ADJACENT PROPERTIES

There are no known operators of any relevant activities on directly adjacent properties or locally adjacent properties.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Pit shell constraints for Grobapo and Mattysmyra Mineral Resource updates

MPlan prepared constraining pit shells for ReedLeyton to be used for the Mineral Resource update of the Grobapo and Mattysmyra deposits using optimised pit shells generated using Datamine™ NPVS software.

The input assumptions were based upon input assumptions used for the Kringel deposit. Although there was no engineering or geotechnical work performed explicitly to support these assumptions, the Kringel deposit input parameters were deemed to be a reasonable benchmark for these neighbouring deposits.

The key assumptions used in the generation of the resource constraining pit shells for both Grobapo and Mattysmyra resources were:

- Overall Slope Angle for Resource Pit Shell: 55 degrees
- Mill Cut-off Grade = 4.00%
- Break Even Cut-off Grade = 4.21%
- The cut-off grades assumed:
- Assumed Graphite Price of USD 2,320
- Recovered value of 2,103 USD/t after applying costs, taxes, mining, and process recovery factors.
- Mining Cost: 4.51 USD/t rock mined
- Process Cost: 84.18 USD/t mill feed
- Dilution 2.5%
- Mining Recovery 97.5%
- Process Recovery 93.7%

No other relevant data and information is for presentation at this stage.

25 INTERPRETATION AND CONCLUSIONS

25.1 Summary interpretation and conclusions

The Project currently comprises an existing open pit graphite mine, graphite processing facility, tailings storage dam, and related infrastructure. These existing facilities are to be expanded and upgraded as described.

The Mineral Resources estimated and disclosed herein supersede the Historic and Previous Mineral Resources and apply current practices and assumptions.

The following Mineral Resource Estimate have been prepared in accordance with the CIM Definition Standards of May 2014 (Published). The classification of the resource at the appropriate levels of confidence are considered appropriate on the basis of drill hole spacing, sample interval, geological interpretation and all currently available assay data.

- Kringel is a mineral resource with of Measured 1.0 million tonnes and Indicated 1.6 million tonnes at average grades of 9.1%C.
- Mattsmyra is a mineral resource with Indicated 5.8 million tonnes at average grades of 7.1%C and Inferred 1.5 million tonnes at average grades of 8.0%C.
- Gropabo is a mineral resource with Indicated 2.3 million tonnes at average grades of 7.7%C and Inferred 0.6 million tonnes at average grades of 8.0%C.

At Kringel, the existing open pit mine will be dewatered and further developed in accordance with the proposed mining plan. A conventional open pit mining operation is planned for the Kringle Mine and is very close to the Woxna Facility.

Conventional drill and blast combined with trucks and shovel mining method have been selected as the optimum method for Kringel based on the following advantages:

- Well adapted to rock material characteristics (variable waste thickness, high rock strength material, faults between the four main graphite mineralisation domains);
- High flexibility and reactivity to change in planned production rates, mine plan, hauling destinations, economic parameters;
- High selectivity, with a large range of bucket sizes;
- Low initial CAPEX investment and ease to find a local subcontractor or supplier;
- Low risk for a well-known mining method that has been previously used at Kringel.

The mining operations will be conducted by a specialist mining contractor, to maximise the operational efficiency, provide flexibility, and lower investment risk to the project.

The mine plan begins in the east pit and is designed to provide approximately 160,000 tonnes of high-grade 'Type A' process plant feed every year until it runs out in year 16, after which the lower grade 'Type B' material that has been stockpiled until then is now fed to the plant. The production plan for the process plant continues on until year 19 when the 'Type B' stockpile is exhausted.

The Kringel mine depth is limited to 70-metres, (applied due permitting), and by the mining concession limits at its geographic boundary. The proposed mine plan fully exploits the available resource within these limits and is therefore relatively insensitive to other factors that might otherwise expand the pit, such as an increase in product pricing.

The project aims to produce high purity graphite anode materials suitable for the lithium-ion battery industry. To achieve this, the existing concentrator facilities will be upgraded, and a new value-add production (VAP) facility will be developed. The concentrator upgrade includes a new crushing circuit, and additional flotation and milling. The VAP facility includes dry product sizing, spheronizing, high temperature thermal treatment, and graphite coating plant.

The concentrator is designed to process 160,000 tpa of mineralised material with a grade of approximately 9.65% C to produce an average of approximately 15,692 tpa of graphite concentrate, grading 93% C.

The concentrate is fed to the VAP Upgrade plant which will produce an average of approximately 6,604 tpa 99.95% C Coated Spheronized Purified Graphite (CSPG) and an average of approximately 7,749 tpa 93% C jet milled graphite.

The existing Woxna tailings facility will be expanded, and an additional separate tailings management facility is planned to be constructed to combined accommodate the proposed LoP tailings volume:

- The current TMF footprint will be used for the storage of the non-acid generating tailings (TMF NAG), extended to the North to store 1.954 million cubic meters of tailings; and
- A smaller TMF for the potentially acid generating waste (TMF PAG), built in connection to the eastern part of the TMF NAG to store 0.217 million cubic meters of tailings.

These tailings facilities holding a total of 2.171 million cubic meters of material will be developed in up to five stages, and ultimately will be sufficient to hold the planned life of project tailings.

The use of the small Lake Uxatjärn, located close to the TMF, as a clarification pond for the mine will continue. The existing small embankment dam will be rehabilitated and raised to ensure it is sufficient.

Water from the clarification pond will be reused into the processing plant. If excess water is required to be discharged, it will undergo treatment before being released to the Älmån river.

The project has existing permits in place to enable the restart of operations, however these are limited to only 100,000 tpa plant feed, and do not cover the upgrade facility. Therefore a re-permitting exercise is necessary. Re-permitting is expected to be straight forward.

The Project has an initial capital investment of USD 121.1 million and a life of mine capital of USD 133.0 million.

Operating costs are estimated at USD 25.1 million per annum.

With Spheronized Coated Graphite sell price of USD 10,000 per tonne, and USD 1,200 per tonne for micronized graphite, and applying discount rate of 8% the project pre-tax NPV is USD 317.0 million with IRR 42.9%, and post-tax NPV is USD 248.2 million with IRR 37.4%. The post-tax payback period is 4.51-years from initial investment, or 2.51-years from initial production.

The Project utilises low-cost low carbon footprint hydroelectric power from the grid, leading to favourable economics, and lower Project carbon footprint. With a clean energy supply, and no hydrometallurgy in the process, the products are sustainably produced.

The project provides a local security of supply for Sweden and Europe.

Future opportunities exist to extend the project duration, and/or increase the project throughput via development of the other deposits.

The project economics show that VAP facility is responsible for substantial economic value add, and this fact may be leveraged via the sourcing of alternate suitable feed materials.

25.2 Opportunities

25.2.1 Mining

Mining fleet is proposed to be via conventionally fuelled diesel-powered vehicles. Future work could investigate potential for bio-diesel fuelling of this conventional fleet. In future, as contractors gain experience with electric vehicles (including maintenance skills for example) and build up their electric vehicle fleets, it may be possible to switch over to an electric powered mining fleet. This would represent an opportunity to lower the direct (onsite) usage of fossil fuels by the project, however it should be considered and analysed on a whole life basis, and for the moment the best option is the currently recommended conventional powered fleet.

25.2.2 Process

The following are potential opportunities regarding improving the processing of the mined material:

- An improved flotation concentrate grade and recovery by:
  - Improving and optimising the flotation circuit, for example number of cleaner/regrind stages.
  - Optimise flotation cells type, for example mechanical or pneumatic.
  - Optimise regrind mill type, for example tower mill type or possibly attrition scrubbing.
- Hydrometallurgical processes for reducing the impurities in the flotation concentrate
- improve spheronization yield for processing the whole flotation concentrate
- assess different type of furnace, for example vertical or vacuum
- optimise thermal conditions, for example temperature and duration at the temperature, as well as the heating rate.
- optimise density of feed to furnace
- assess optimum off-gas cleaning
- optimise heat recovery
- the use for nitrogen or argon as the inert gas in the thermal process
- maximising the value of the fines produced by spheronizing, for example thermally upgrading.
- The upgrade facility is a substantial source of the value add and project revenues, and therefore this opens the potential of utilising the upgrade facility independent to the Woxna mine for treatment of externally sourced graphite feedstocks.
- The opportunity for producing alternate / additional products remain a possibility, for example a large flake feedstock for expandable graphite could be produced via a relatively straight forward screening upgrade in the concentrator flotation circuit. At this stage this and other similar options have been ruled out since they add complexity to the project without providing an associated increased economic benefit, however, should markets change, so could this conclusion and the current approach.

25.2.3 Waste management

The key opportunities for the next phases of the design will be:

- Optimize design and cross section to minimize potential amount of construction material required
- Opportunity to accommodate increased or decreased production with the current design depending on potential increase in mined tonnages.
- The old TMF likely to benefit from a restart of operations, as it should help clean-up the facility by ceasing the oxidation of the old tailings as they are covered by fresh and more inert de-sulfurized tailings.
- Investigate the potential to use waste rock as a construction material, after confirmation it is not acid generating.
- Investigate the potential for dry stacked tailings.

25.3 Risks

25.3.1 Mining

Mining risks include:

- the accuracy of the block model used;
- unforeseen geotechnical issues whereas full geotechnical characterization of the pit slopes has not yet been performed;
- drilling and blasting uncertainty, whereas drill and blast-ability of the rock mass has not yet been characterised;
- seasonal surface water, groundwater, and weather related modifying factors, which have not yet been characterized;

- difficulty in identifying the Type-A and Type-B graphite for RoM or stockpile feed.

25.3.2 Process

Process risks include:

- Flotation concentrate grade is too low for efficient thermal upgrading and the high impurity level could result in difficulties regarding treating the off gases and efficiencies regarding heat recovery.
- The yield of the spheronizer could be low when treating all the flotation concentrate, producing high levels of fines with little value
- The packed density of the feed to the furnace could have an adverse effect on the thermal upgrading or the capex of the thermal units.
- potential adverse environmental concerns regarding the thermal off gases.
- the cost and application of coating technology and the value of the product could be lower than predicted.

25.3.3 Waste management

The key risks associated with the current design and proposed operation are as follows:

- Tailings characterization (geochemistry, density, production)
- Uncertainties on site conditions (existing dams, and foundation characterization)
- Availability and suitability of construction material.

26 RECOMMENDATIONS

26.1 Future project development

Based on the favourable economic results of the PEA it is recommended that a PFS be completed to give an indication of the project's feasibility. It may also be appropriate to develop a study solely focussed on the VAP facility, since it this facility that requires most of the additional definition work, limiting the scope in this way would increase the efficiency of this work, and furthermore it could make a lot of sense to consider the VAP as a stand-alone project.

Further definition of the project is required to complete a PFS and therefore it is recommended that various further studies and testwork be undertaken.

26.2 Mining

The mining study of the Kringel deposit may undertake the following detailed studies at the Pre-feasibility level of investigation to seek improvement upon the economic viability of the project and to help de-risk the project where uncertainty exists:

- Blasting /ripping study to determine what drill and blast Savings may be possible.
- Haulage study to determine the possibilities for optimal use of equipment and minimization of congestion.
- Investigation how many pushbacks and of what size and shape will help to improve the schedule and the NPV of the project.
- A hydrology / groundwater study to help determine more accurate dewatering requirements.
- In situ bulk density measurements on a representative size and number of samples to precisely evaluate the dry density and the in situ natural moisture content, taking into account the lithologies variability.[6]
- Mitigation measure: carry out additional measurements, measure dry density (after a stay in oven at 110 °C) and moisture content.

26.3 Environmental

Refer to Section 20 and specifically to Table 20-1 for a list of the environmental studies that will be needed as the project develops, and Table 21-8 which estimates the associated costs. These required studies are listed as follows:

- Site localisation study for waste rock dumps and TSF
- Waste rock and tailings characterization
- Hydrogeological investigation
- Surface water chemistry
- Nature value inventory
- Bottom fauna investigation, sediment sampling and fish inventory.
- Investigation of ground vibrations and noise
- Process and mine pit water treatment

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[6] Density measurements: Bulk densities used in this mining study were measured as part of the geotechnical tests on samples left at ambient temperature and still holding some moisture, therefore densities are considered as in situ wet/natural densities, not dry densities. Moreover, a large number of samples have been tested at this stage but not all samples have been regrouped by lithologies with the exception of **hg** and **grf** domains. The geotechnical test results suggest that the lithologies have variable densities depending on the domains. This has an influence on the planned mine production and needs to be estimated more precisely. The in situ natural moisture content needs to be evaluated. Bulk sampling for density and in-situ moisture evaluations is recommended following an international standard such as ISO or ASTM

- Mine Closure

26.4 Waste management

This report presents the concept for the Woxna Kringel Mine for the proposed 19 years of operation. Due to the nature of the study, some of items have been assumed and will need to be studied further during the next level of engineering.

- Investigation
  - Geotechnical investigation (Dams and Foundation)
  - Hydrogeological investigation (Existing dams, foundation, and inundation area)
  - Laboratory analyses (tailings characterization, foundation, and construction material)
  - Borrow material (location, quantities, quality)
- Prefeasibility study - including options comparison
  - Confirmation of design criteria (particularly tailings density)
  - Check on Stability and seepage analysis.
  - Confirmation of water balance, design flood and design of water management system
- New Environmental Permit application.
- Feasibility study
- Detailed design for construction.
  - Construction Document including Detailed design drawings, technical specification.
  - Health and Safety and Risk Assessment
  - CQA and instrumentation plan
  - Dam breach Analysis and Consequence classification
  - OMS manual

26.5 Metallurgical Testwork

Zenito recommends that tests are performed to confirm the results for the upgrading of the flotation concentrate. A significant quantity of graphite concentrate will be required, and this would be best produced by restarting the existing flotation plant and producing an intermediate grade concentrate. This concentrate would then either be recycled through the existing flotation and regrind plant on a batch basis until the required graphite grade is produced or the intermediate concentrate could be upgraded in a laboratory which has large flotation and regrind units. Upgrade tests would then be carried out on the concentrate. These would include:

- Micronizing and spheronizing to produce the required samples for thermal upgrading.
- Thermal upgrading where the conditions for producing the high-grade graphite would be examined as well as assessing the off-gases and the required treatment of them.
- The upgraded spheronized product would then be processed through the coating process to produce a final product which could be used for marketing purposes.

These tests would be at a relatively large scale, for example pilot plant scale. It is possible that equipment suppliers might be able to perform some of the tests, for example spheronizing.

The thermal tests would include tests to assess the options for using different furnace types, for example vertical, vacuum or pusher furnaces. The tests would produce data for the specifying the thermal equipment and as well as producing the data for treating the off gases.

Based on external or internal pilot scale tests a demonstration plant should be built to further optimize and validate the integrated production of coated spherical purified graphite and produce meaningful quantities of qualification material that can be marketed towards potential customers in Europe.

Opportunities exist for improving and optimising the processes, including:

- Flotation tests, including regrind, to optimise number of cleaner flotation stages. Particle size of the final graphite flotation concentrate will not be critical due to jet milling at the VAP.
- Hydrometallurgical tests with the aim of reducing the level of impurities in the flotation concentrate so that the thermal treatment might be improved or optimised.
- Flotation cell type, for example mechanical or column cleaner cells.
- Spheronization tests to optimise and confirm conditions.
- Jet milling of fines to optimise and confirm conditions.
- Production of samples for marketing.
- These production samples are in support of further discussions with potential customers with the aim of agreeing off-take contracts to confirm product pricing.

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APPENDIX A. TAILINGS MANAGEMENT FACILITY, SITE PHOTOS

Figure 1: View from East Dam Crest looking East

Figure 2: East Dam downstream Slope

Figure 3: East Dam Downtream Slope

Figure 4: East Dam Downstream Slope

Figure 5: Intermediate berm

Figure 6: View of the Tailings Deposition area from the existing waster rock dumps

Figure 7: North West Dam spillway

Figure 8: North West Dam Northern Downstream Slope

Figure 9: North West Dam Crest

Figure 10: Supernatant Pond towards North West Dam

Figure 11: Clarification Pond

Figure 12: Clarification Pond Outlet

APPENDIX B. TMF DEPOSITION SCHEDULE

Figure 1: Year 0 - Deposition from the Intermediate Berm of the TMF NAG.

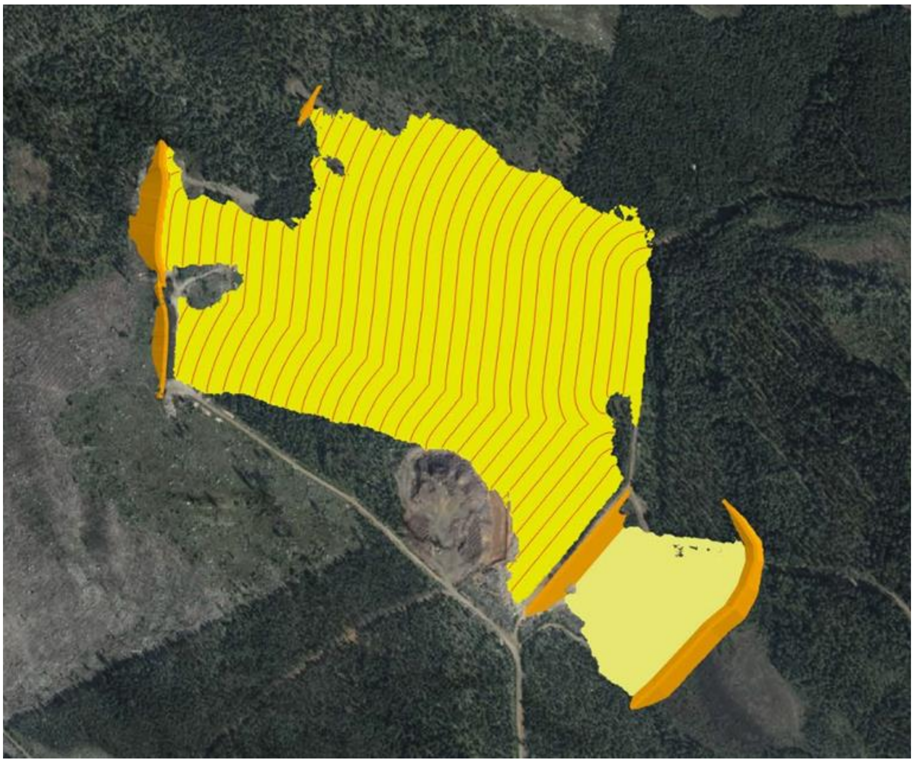

Figure 2: Year 1 - Deposition from East Dam of the NAG TMF (+276.5 m.).

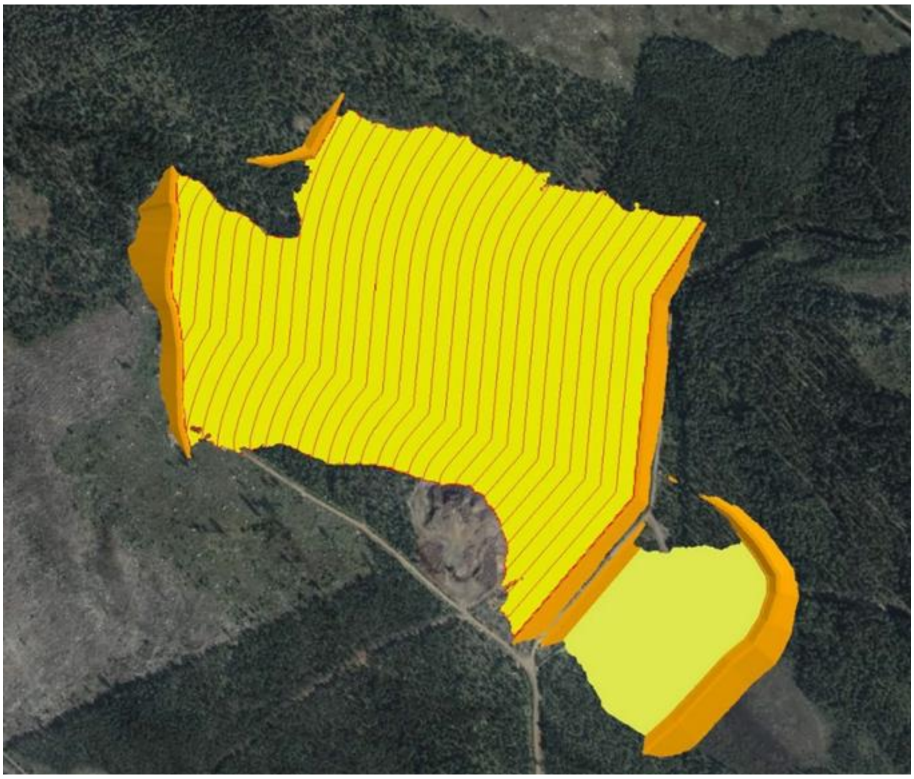

Figure 3: Year 7 - Deposition from East Dam of the TMF NAG (+ 279.5 m).

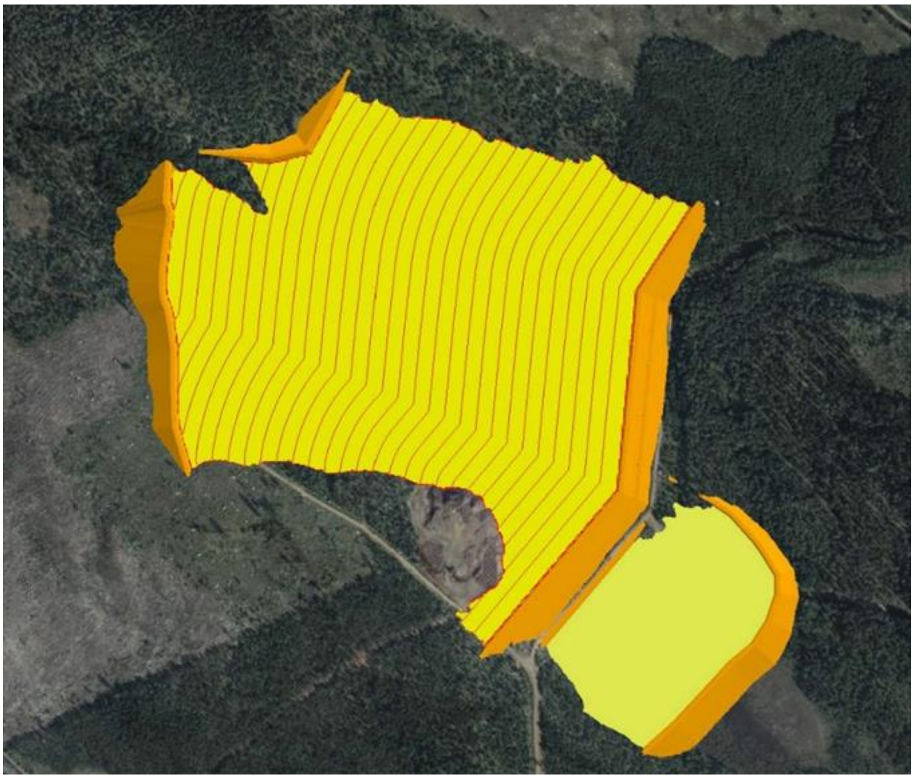

Figure 4: Year 13 - Deposition from East Dam of the NAG TMF (+ 282 m).

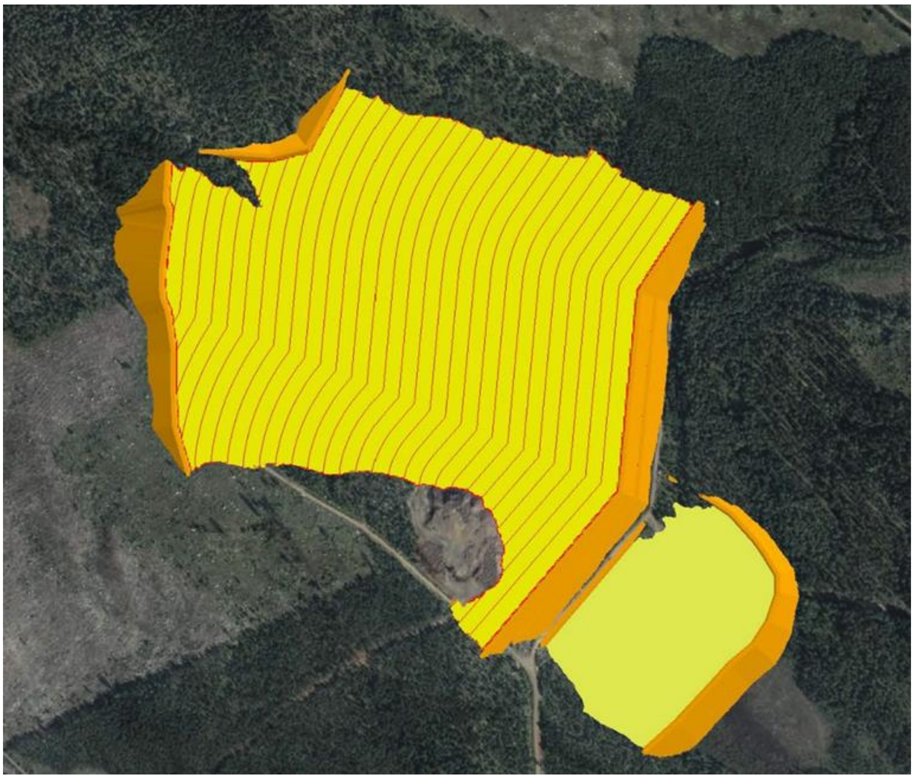

Figure 5: Year 20 - Deposition from East Dam of the NAG TMF (+282.5 m).

CERTIFICATE OF QUALIFIED PERSON

Christopher Stinton

As the individual who has co-authored or supervised the preparation of sections of the Technical Report prepared for Leading Edge Materials Corp. (Issuer) entitled "Woxna Graphite Project, QP Certificate - Christopher Stinton" dated effective 09 June 2021 (Technical Report), I hereby certify that:

1. I am a Principal Process Engineer with Zenito Limited (Zenito) of 27 Old Gloucester Street, London, WC1N 3AX, United Kingdom.
2. I graduated with an Honours of Bachelor of Science in Minerals Engineering from the Birmingham University, United Kingdom in 1979. I am a Chartered Engineer (CEng) registered with the Engineering Council UK and a Member of the Institute of Materials, Minerals and Mining (MIMMM). I have been working in minerals engineering since 1979 and have had experience in process design and engineering, plant operations and management.
3. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 (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 fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I have personally visited the Woxna Graphite Project site in October 2013.
5. I have authored or supervised the work carried out by other Zenito professionals for Zenito's contribution to the Technical Report and take responsibility for the Sections outlined in Table 2-1.
6. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
7. I have had prior involvement with the project on behalf of the Issuer as described in the following reports:
   - Technical Report entitled "Woxna Graphite Restart Project, Preliminary Economic Analysis" dated effective 11 October 2013, No. 0482-RPT-001.
8. I have read NI 43-101, and the sections of the Technical Report for which I am responsible, as stated above, have been prepared in compliance with NI 43-101 and Form 43-101F1.
9. To the best of my knowledge, information, and belief, as of the effective date, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23rd July 2021, London, United Kingdom.

Christopher Stinton, BSc (Hons), CEng MIMMM

Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Geoffrey Reed

As the individual who has co-authored or supervised the preparation of sections of the Technical Report prepared for Leading Edge Materials Corp. (Issuer) entitled "Woxna Graphite Project, QP Certificate - Geoffrey Reed" dated effective 09 June 2021 (Technical Report), I hereby certify that:

1. I am the Principal of ReedLeyton Consulting ("ReedLeyton") of PO Box 6071, Dural, NSW, 2158, Australia.
2. I graduated with a degree in Geology with a Bachelor of Applied Science from the University of technology, Sydney, NSW, Australia, awarded in 1997.
3. I am a Member of the Australasian Institute of Mining and Metallurgy since 1998. My relevant experience for the purpose of the Technical Report is drawn from working as a geologist for a total of over 24-years since my graduation from university.
4. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 (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 fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
5. I have personally visited the Woxna Graphite Project site in June 2012 and in June 2014.
6. I have completed the work in contribution to the Technical Report and take responsibility for and take responsibility for the Sections outlined in Table 2-1. My input is based in large part on examination of the material presented to me by Leading Edge Materials Corp during June 12, 2012 to October 20, 2012. First hand impressions about the style of mineralisation are based on examinations of drill core from representative drill holes during June 12 to 13, 2012 and June 17 to 18 2014.
7. I have authored or supervised the work carried out by other Zenito professionals for Zenito's contribution to the Technical Report and take responsibility for the Sections outlined in Table 2-1.
8. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
9. I have had prior involvement with the project on behalf of the Issuer as described in the following reports:
    - Technical Report entitled "Technical Report for the Woxna Graphite Project, Central Sweden" dated March 24, 2015.
    - Technical Report entitled "Woxna Graphite Restart Project, Preliminary Economic Analysis" dated effective 11 October 2013, No. 0482-RPT-001.
    - Technical Report for the Kringel Graphite Deposit, Part of the Woxna Graphite Project, Central Sweden" dated November 2, 2012.
10. To the best of my knowledge, information, and belief, as of the effective date, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23rd July 2021

Geoffrey Reed, B App Sc, MAusIMM (CP), MAIG

Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Henning Holmström

As the individual who has co-authored or supervised the preparation of sections of the Technical Report prepared for Leading Edge Materials Corp. (Issuer) entitled "Woxna Graphite Project, QP Certificate - Henning Holmström" dated effective 09 June 2021 (Technical Report), I hereby certify that:

1. I am a Principal Geologist with Golder Associates Corporation (Golder) of Östgötagatan 12, 116 25 Stockholm, Sweden.
2. I graduated with a degree in M.Sc. in Geotechnology from Luleå University of Technology, Sweden awarded in 1996 and a Ph.D in Applied Geology from Luleå University of Technology, Sweden awarded in 2000.
3. I am a Member of Australiasian Institute of Mining and Metallurgy (MAuIMM no 308735) as well as the Australian Institute of Geoscientists (MAIG no 5628). My relevant experience for the purpose of the Technical Report is drawn from working as a engineer and environmental scientist (several roles and positions) for a total of over 25-years since my graduation from University.
4. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 (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 fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
5. I have visited the Woxna Graphite Project Site regularly between 2011-2013, and most recently in November 2020.
6. I have supervised the work carried out by Golders for my contribution to the Technical Report, and take responsibility for the Sections outlined in Table 2-1.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had prior involvement with the project, as I am the former Director of Woxna Graphite, and on behalf of the Issuer as described in the following reports:
    - Technical Report entitled "Woxna Graphite Restart Project, Preliminary Economic Analysis" dated effective 11 October 2013, No. 0482-RPT-001.
9. I have read NI 43-101, and the sections of the Technical Report for which I am responsible, as stated above, have been prepared in compliance with NI 43-101 and Form 43-101F1.
10. To the best of my knowledge, information, and belief, as of the effective date, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23rd July 2021, London, United Kingdom.

Henning Holmström

Henning Holmström, M.Sc., Ph.D, MAusIMM, MAIG

Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Mathieu Gosselin

As the individual who has co-authored or supervised the preparation of sections 15 and 16 of the Technical Report prepared for Leading Edge Materials Corp. (Issuer) entitled "Woxna Graphite Project, NI 43-101 Technical Report - Woxna Graphite" dated effective 09 June 2021 (Technical Report), I hereby certify that:

1. I am CEO, President and Industry Expert-Mining with Gosselin Mining AB with an office situated at Industrivägen 23, Solna, Sweden 171 48.
2. I graduated with a degree in Bachelor of Engineering, Mining from McGill University, Montréal in 2004.
3. I am a member of Ordre des ingénieurs du Québec.
4. I have worked as a mining engineer continuously for a total of 17 years since my graduation from university in 2004. I have relevant experience in the evaluation and extraction of industrial minerals, phosphate, coal and graphite deposits. I have sufficient experience in the modifying factors, mining methods, mine life and production rates, mineral reserve and mining costs estimating techniques that are relevant to the deposit under consideration. I also have appreciation of extraction and processing techniques applicable to that deposit type.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("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 fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 15 and 16 and part of Section 1 of the technical report titled "Woxna Graphite Project, NI 43-101 Technical Report - Woxna Graphite" dated June 9th 2021 (the "Technical Report") relating to the Woxna Graphite Kringel property. I visited the Woxna Graphite Kringel property on 05/07/12 for 2 days and on 02/14/13 for 1 day.
7. I have had prior involvement with the property that are the subject of the Technical Report. The nature of my prior involvement is as qualified person in the superseded previous Preliminary Economic Assessment dated October 29th, 2013:
    - Technical Report entitled "Woxna Graphite Restart Project, Preliminary Economic Assessment" dated effective 11 October 2013, No. 0482-RPT-001 Rev 1.
8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43 101.
10. I have read NI 43-101 and Forms 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filling of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 26th Day of July 2021, Sölvesborg, Sweden.

Mathieu Gosselin, Eng.

Qualified Person