NI 43-101 Preliminary Economic Assessment Technical Report on the Driftwood Creek Magnesite Deposit Project Brisco, British Columbia, Canada

PREPARED FOR:

MGX Minerals Inc.
Vancouver, BC, Canada

RESOURCE EFFECTIVE DATE:
December 31, 2016

REPORT DATE:
April 16, 2018

QUALIFIED PERSONS:

Allan Reeves, P.Geo.
Tuun Consulting Inc.

Matt R. Bender, P.E.
Samuel Engineering Inc.

Antonio Loschiavo, P.Eng.
AKF Mining Services Inc.

Table of Contents

TABLES

FIGURES

APPENDICES

1 SUMMARY

1.1 Introduction

This Resource Technical Report was prepared for MGX Minerals Inc. (MGX or the Company), providing a National Instrument (NI) 43-101 compliant Preliminary Economic Assessment (PEA) of the potential magnesium oxide (MgO) resources at the Driftwood Creek Magnesite Deposit (the Project or the Driftwood Creek Project), located in British Columbia (BC), Canada. Note that the resources have been updated (effective date: December 31, 2016), and these updated values are used in this report. MGX is listed on the Canadian Securities Exchange as symbol (XMG).

Magnesium oxide is classified as an industrial mineral, and this report has utilized the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practices Industrial Minerals Guidelines (November 2003) to supplement both the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) and the CIM Guidance on Commodity Pricing (2015). For an industrial mineral, the guideline states *"the QP should give priority to: (i) the value of the intended mineral product; (ii) market factors; and (iii) applicability of the market criteria to the mineral deposit being assessed."*

The only mineralization of economic interest on the property is magnesite. Magnesite is magnesium carbonate ($MgCO_3$), and has a theoretical magnesium oxide content of an average grade of 47.6% MgO. Magnesite products are obtained by calcining magnesium carbonate or magnesium hydroxide at different temperatures. Caustic-calcined magnesia (CCM) is a reactive oxide easily hydrated with water, and is prepared by burning off carbon dioxide at extremely high heat. MGX's initial plan is to produce dead-burned magnesia (DBM), the principal industrial mineral derived from magnesite. It is a refractory material primarily used to line furnaces in the steel industry.

Over 90% of magnesite resources are sedimentary-hosted, either sparry type (also called Mount Brussilof type), as defined in Simandl and Hancock (1998), or Kunvarrara type, as defined in Simandl and Schultes (2001). The Driftwood Creek magnesite is a sparry-type deposit, like the nearby Mount Brussilof Mine at Radium, BC.

Information on active North American magnesium-producing mines is difficult to obtain. In the United States (Bray, 2016; USGS, 2014 Annual Report), only US Magnesium LLC in Salt Lake City was recovering magnesium electrolytically, from the Great Salt Lake brines, for which the United States Geological Survey (USGS) withheld proprietary production data. The only other US project on record is the Nevada Clean Magnesium (Canada) Tami-Mosi (Wardrop PEA, 2011), which is proposed to test recovery of magnesium from dolomite.

In Canada, the situation is similar, with the British Columbia Ministry of Energy and Mines (BC MEM) withholding proprietary data on the Mt. Brussilof Magnesite Mine (Baymag Inc.), which transports its ore to production facilities in Exshaw, Alberta (AB). Two projects that have been proposed would entail recovery of magnesium from asbestos tailings, and mining of magnesium-rich dolomite.

Alliance Magnesium Inc. proposed in 2014 to electrolytically produce magnesium from asbestos tailings in Quebec. Gossan Resources Ltd. has the Inwood Magnesium Project in Manitoba (MB), which is a magnesium-rich dolomite deposit that would require a specialized, high-efficiency production process. In 2013, Gossan announced that they had failed to conclude a definitive agreement with the process developer.

The Driftwood Creek Magnesite Project is amenable to the production of DBM or CCM. According to the USGS (Bray, 2016), DBM consumption decreased by 9% in 2015, while CCM continued to increase for animal supplement, fertilizer, and environmental applications. Magnesium usage in automobile parts continues to increase.

The outlook is favourable; new capacity in China is expected to be limited, as older and smaller high-cost producers have shut down, with more production anticipated to be lost as the government enforces environmental regulations on energy-intensive industries.

1.2 Property Description

The Project property is located approximately 53 kilometres (km) southeast of Golden, BC, and approximately 210 km northwest of Cranbrook, BC. Access is by Forest Service Road (FSR) from either Brisco or Spillimacheen. Local infrastructure is the paved Highway 95, with a Canadian Pacific Railway (CPR) spur nearby. The property consists of seven contiguous mineral tenures, with a total area of 835.44 hectares (ha) (2,064.42 acres).

1.3 History

The Project was first described by J.W. McCammon in the 1964 BC Minister of Mines Annual Report. The 1978 BC Assessment Report, prepared by Kaiser Resources, stated that geological mapping indicated a large deposit of magnesite.

In 1987, Canadian Occidental Petroleum Ltd. (Canoxy) staked claims over the deposit, and followed up with line-cutting, geological mapping, and rock chip sampling in 1989. In 1990, four NQ-diameter diamond drill holes were completed on the eastern part of the deposit.

Ownership of the property was shared by Klewchuk, Rodgers, and Kikauka by the time Tusk Exploration Ltd. (Tusk) conducted diamond drilling in 2008. Tusk drilled seven NQ holes.

In July 2014, the owners entered into an option agreement with MGX, and subsequently drilled eight BTW diamond drill holes in 2014, and fourteen more in 2015. That was followed by the collection of a 100-tonne bulk sample in July 2016. Metallurgical testwork is currently in progress. After the release of the maiden resource estimate in 2016, MGX embarked on an infill program that autumn, drilling sixteen diamond drill holes, totalling 1,211.5 m, with the purpose of improving confidence in readily accessible, near-surface resources. These near-surface resources would then form the bulk of the short-term mine plan for permitting purposes, in the event of a positive economic outcome of the PEA.

1.4 Geology and Mineralization

The rock units identified on the property, from oldest to youngest, are Hmn1A, Hmn1B, Hmn2, Hmn3, and Hmn4. All five units were placed within the Mount Nelson formation of Proterozoic (Helikian) age by J.E. Reesor in 1957.

The magnesite deposit is within unit Hmn1B, and has been described as white-buff to cream-coloured, very fine-grained to very coarse-grained (coarser grained near faults/conduits), and containing irregular concentrations of siliceous veinlets, laminae, or blebs of up to 2 cm thick.

The Project magnesite occurrence is classified as a sparry magnesite deposit (E09) by BC MEM (Simandl and Hancock, 1998). This deposit type is characterized by stratabound (and typically stratiform), lens-shaped zones of coarse-grained magnesite, mainly occurring in carbonates, but also observed in sandstones or other clastic sediments.

1.5 Exploration Status

In 2016, 25 percussion drill holes (PDH) were drilled and sampled for obtaining approximately 100 tonnes of magnesite as a bulk sample for detailed metallurgical testwork. Specific gravity (SG) testing was also undertaken.

As noted above, MGX conducted an infill diamond-drill program aligned with recommendations from the maiden resource report. The program included supplemental SG testing to confirm that spatial variability is minimal within the magnesite deposit.

1.6 Sample Preparation, Analyses, and Data Verification

Drill core from the 2008, 2014, 2015, and 2016 drill programs were split, with one half of the core bagged in 2 m or 3 m intervals for shipment to one of ALS Minerals (ALS) laboratories, and the other half retained in the core racks. Supervision and sample security at the Vine Creek core facility in Cranbrook meets industry best practices.

Whole rock analysis was conducted by ALS at one of their facilities, in either Kamloops or North Vancouver. Blank samples were inserted into the sample stream every 20 samples by Andris Kikauka, P.Geo. ALS maintains ISO/IEC 17025:2005 and ISO 9001:2015 certification.

Drill hole locations have been confirmed by Differential Global Positioning System (DGPS) surveys performed by WSP Group of Cranbrook, BC.

The 2016 infill drilling was sampled (primarily at 3 m intervals) and handled as above.

1.7 Metallurgical Testing

In the 2008 BC Assessment Report #30243, it was reported that SGS Lakefield Research (SGS) conducted preliminary beneficiation testing of two composite samples (West Zone and East Zone). A preliminary flowsheet and a reagent scheme were developed, with magnesite concentrate being recovered as silicate flotation tailings with an estimated grade of 93.4% MgO (East) and 86.3% MgO (West). Efforts to reduce iron content in the concentrate were unsuccessful.

The June 2016 East Zone bulk sample is still undergoing testing.

To date, there has been only one prior testwork program conducted on the deposit. In 2007/2008 SGS Lakefield conducted preliminary beneficiation testing of two composite samples from the West and East Zones. This consisted of four outcrop samples, each approximately 50 kg.

Samples were ground, split, and subjected to chemical analysis, X-ray diffraction analysis (XRD), screening and fractional analysis, heavy liquid separation, grinding tests, and flotation tests. Results showed that reverse flotation can produce high grades of magnesite concentrate.

From the testwork results obtained, these conclusions can be made:

- This mineralized material has a high magnesite grade estimated at 93.4% for the East Zone and 86.3% for the West Zone. It responded well to beneficiation by silicate flotation, with magnesite concentrate generated as silicate tailings.
- All the efforts to reduce iron content of the magnesite concentrate were unsuccessful. It is believed that this is due to the presence of iron in magnesite crystal structure as solid solution.
- Heavy media separation (HMS) can be considered as a potentially suitable process for primary upgrading, to reject a large portion of silicate minerals, at approximately 73% to 80%, and calcite at nearly 40% in a coarse fraction.
- Grinding and screening to different fractions failed to generate an acceptable magnesite concentrate.
- High intensity dry and wet magnetic separations to separate iron-containing minerals were attempted. These methods failed to perform a reasonable task in reducing the iron content of the magnesite concentrate.

1.8 Mineral Resources Estimate

The Mineral Resource update was conducted by Tuun Consulting Inc. (Tuun), and in Tuun's opinion, the existing sample data is considered adequate for estimating the Mineral Resource. Tuun considers that the primary focus of the Driftwood Creek Magnesite Deposit will be amenable to magnesite quarrying by a small excavator and truck fleet.

This Mineral Resource is based on drill data, BC Assessment Reports, and sections developed over many years. The information was reviewed, and all work is believed to have been executed in a professional manner that met the standards of care in place at the time.

Table 1-1: Driftwood Creek MgO % Resource Estimate

Class	Tonnes ('000s)	MgO (%)	Al_2O_3 (%)	CaO (%)	Fe_2O_3 (%)	SiO_2 (%)	LOI (%)
Measured	4,703	43.31	1.01	0.95	1.29	5.06	47.83
Indicated	3,145	43.22	0.99	1.05	1.34	4.66	47.99
M&I	**7,848**	**43.27**	**1.00**	**0.99**	**1.34**	**4.90**	**47.89**
Inferred	56	42.95	0.93	0.66	1.43	6.07	47.46

Notes: *__1.__ Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves. __2.__ The Lerchs-Grossman (LG) constrained shell economics used a mining cost of US$8.82/t, processing+ g&a costs of US$106/t, and a commodity price of US$600.00/t 95%MgO DBM. __3.__ Mineral resources are reported within the constrained shell, using a cutoff grade of 42.5% MgO (based on a 20 year LOM) to determine "reasonable prospects for eventual economic extraction." __4.__ Mineral Resources are reported as undiluted. __5.__ Mineral Resources were developed in accordance with CIM (2014) guidelines. __6.__ Tonnages are reported to the nearest kilotonne (kt), and grades are rounded to the nearest two decimal places. __7.__ Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade, and contained metal.*
M&I = Measured and Indicated; t = tonnes% = percent; LOI = loss on ignition.

Mineral Resources were estimated in conformity with CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines. Mineral resources are not Mineral Reserves and have no demonstrated economic viability. This PEA does not support an estimate of Mineral Reserves, since a prefeasibility study (PFS) or feasibility study (FS) is required for reporting of Mineral Reserve estimates. This report is based on mine plan tonnage (mine plan tonnes and/or plant feed).

Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that all or any part of the Mineral Resources or mine plan tonnes would be converted into Mineral Reserves.

1.9 Mineral Reserve Estimate

AKF Mining Services Inc. (AKF) has not developed a Mineral Reserve estimate for the Project as part of this PEA. Significant additional data collection and technical work is required to elevate the technical confidence of the Project to a level consistent with Mineral Reserve estimation, in accordance with the CIM *Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended,* NI 43-101, May 10, 2014.

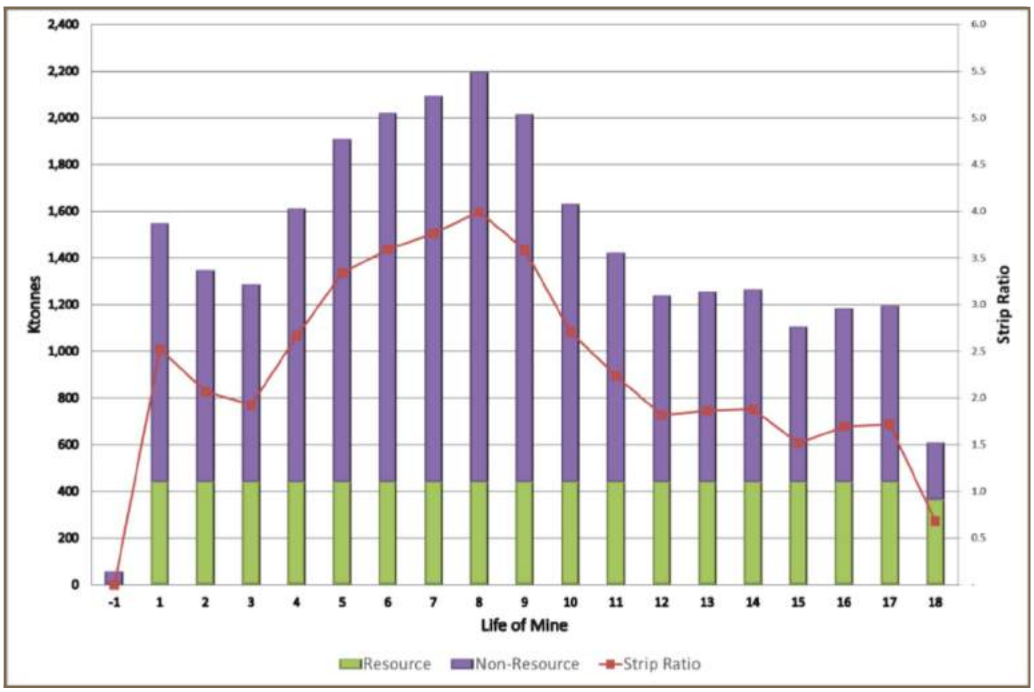
Mineral resources that are not Mineral Reserves do not have demonstrated economic viability. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

AKF is not aware of any previous Mineral Reserve estimates on the Driftwood Creek deposit that have been completed in accordance with any international reporting code.

1.10 Mining Methods

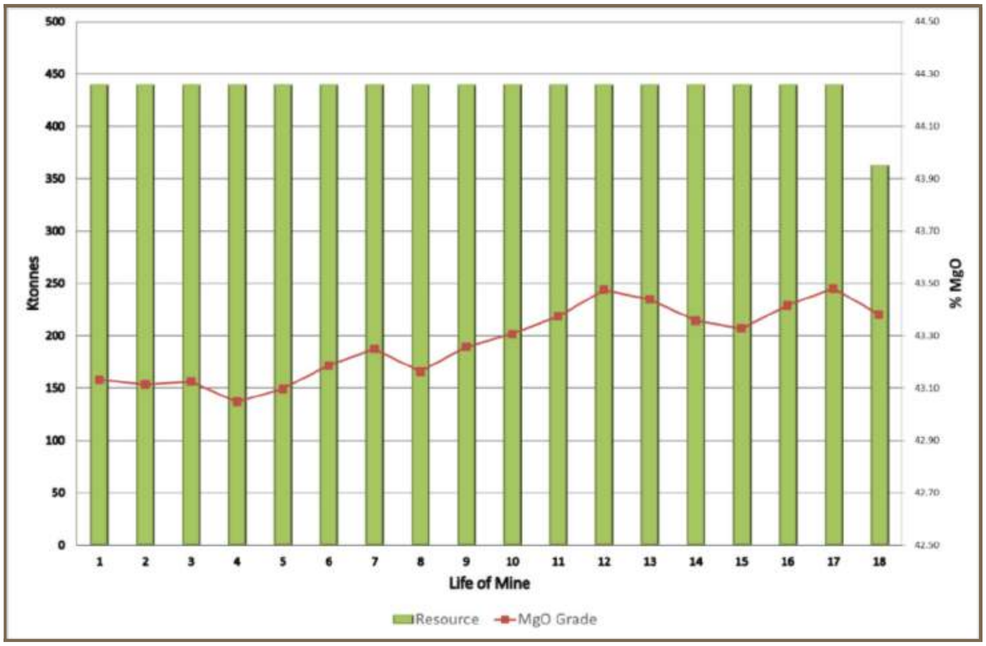
The Project will be mined by conventional, quarry pit, truck-and-excavator operation. Pit optimization and mine planning were carried out on the basis of supporting a plant capacity of approximately 1,200 tonnes per day (t/d), using Measured, Indicated and Inferred resources provided by Tuun. The Project will have an operating life of 19 years, with one year of mine rock pre-stripping followed by 18-years of production operations, as shown in the mine production schedule and strip ratio graph provided in Figure 1-1. The plant production schedule and mill feed grades are shown in Figure 1-2.

All mining equipment will be supplied and operations performed by contractors. Total material movement peaks at approximately 2.2 million tonnes per annum (Mt/a), which requires a modest production fleet of up to six conventional 40-tonne haul trucks and two-2.4 m^3 excavators. Drilling can be completed with two crawler-mounted Ranger drills capable of drilling up to 127 mm diameter holes, in combination with packaged emulsion for blasting.



Source: AKF (2018)

Figure 1-1: Mine Production Schedule and Strip Ratio



Source: AKF (2018)

Figure 1-2: Plant Production Schedule and Feed Grade

All resource material to the plant will be mined and hauled downhill in 40-tonne mine haul trucks to the ready pile located at the bottom of the hillside as discussed in Section 18. The resource material will then be loaded onto 40-tonne highway trucks and transported to the plant facility in Cranbrook, BC.

1.11 Recovery Methods

Mineralized material will be mined on site and then transported 210 km via truck to the plant located in Cranbrook, BC. Here the mineralized material will undergo crushing, grinding, flotation upgrading, calcination, and sintering to produce a saleable DBM product. The plant will also have the ability to produce CCM as a separate product.

The conceptual flowsheet is partially based on the development work completed by SGS Lakefield and a proposal from Industrial Furnace Company Inc. for the calcination and sintering operations. The process operations can be divided into three stages, where the mineralized material will initially be sized and screened, then upgraded, and finally calcined into CCM and then sintered into DBM.

Plant throughput is designed at 1,200 t/d. The plant is expected to achieve an average recovery of 90% with a magnesium oxide (MgO) purity of 94.6%. The DBM product will be bagged and transported to market for sale as a powder.

Dewatered tailings will be trucked back to the mine site for storage in the dry-stack tailings management facility (DS-TMF).

1.12 Project Infrastructure

1.12.1 Mine Access

Mine Access will by the FSR from Brisco along Highway 95. Local infrastructure is the paved Highway 95, with a CPR spur located at Brisco.

1.12.2 Mine Site Infrastructure

Proposed infrastructure includes:

- Portable ATCO trailers;
- 40 kW trailer-mounted genset;
- No water supply will be required on site;
- Portable contractor maintenance shop;
- 20,000 L capacity fuel tank farm with dispensing system; and
- Mine water containment facility.

1.12.3 Rock Management Facility

Over the life-of-mine (LOM), the open pit will produce approximately 19.174 Mt of non-resource material rock, which will be stored in the rock management facility (RMF). The RMF will have a maximum design elevation of 1,430 masl, and a footprint area of approximately 0.32 km^2, as shown in Section 16, Figure 16-6.

1.12.4 Dry-Stack Tailings Management Facilty (DS-TMF)

The DS-TMF will be located at the mine site in close proximity to the run-of-mine (ROM) mineral loading ready-pile area. This will allow for simple access for the return of the mine-haul trucks to end dump then proceed to the ROM mineralized material loading, reducing truck turnaround time.

The facility area is approximately 0.11 km^2, which will contain approximately 1.4 Mt of dry-stack tailings, as shown in Section 18, Figure 18-2. The tailings will also contain between approximately 8% to 12% moisture. All water runoff will be captured and managed by the mine water containment facility.

1.13 Environmental Considerations

To date, no formal baseline characterization studies have been conducted for the Project. Baseline studies will be required prior to submittal of environmental permit documents. No known environmental condition exists that would preclude development of the Project.

1.14 Capital and Operating Costs

1.14.1 Capital Cost Estimate

The capital cost estimate, summarized in Table 1-2, includes the costs required to develop, sustain, and close the operation for a planned 19-year mine life. The construction schedule is based on an approximate 2-year build period. The intended accuracy of this estimate is ±25%, which is suitable for Project evaluation, but not for Engineering, Procurement, and Construction Management (EPCM), or financing.

Table 1-2: Capital Costs Estimate

Description	Pre-Production ($M)	Sustaining/Closure ($M)	LOM ($M)
EA, Permitting, Basic Engineering	6.8	0	6.8
Capitalized Stripping – Rock	0.5	0	0.5
Capitalized Stripping – Organics	0.3	0	0.3
Mine Site and Development	1.5	0	1.5
Plant Site (Timbec Site in Cranbrook, BC)	3.8	0	3.8
Process Plant	37.7	0.4	38.1
MgO Calcination	108.7	0.4	109.1
EPCM	14.4	0	14.4
Indirects	15.9	0	15.9
Reclamation/Closure	0.0	2.5	2.5
Owner's Costs	7.1	0	7.1
Subtotal	196.6	*3.3*	199.9
Contingency (20%)	39.3	0.7	40.0
Total Capital Costs	235.9	3.9	239.8

Notes: *LOM = life-of-mine; $M = million dollars; EPCM = Engineering, Procurement, and Construction Management; % = percent*

1.14.2 Operating Cost Estimate

The unit costs summarized in Table 1-3 are based on an annual production rate of 1,200 t/d, 365 days of operations. These unit costs include mining by a contractor, transportation of magnesite mineralized material from the mine to the plant in Cranbrook, processing, and general and administrative (G&A).

Table 1-3: Operating Costs Summary

	Processed ($/t)	LOM ($M)	Annual ($M/a)
Mining	30.30	237.7	13.2
Transport from Mine to Plant	43.95	344.7	19.2
Processing + G&A	62.06	486.8	27.0
Total	**136.31**	**1,069.1**	**59.4**

Notes: *Mining cost is based on $8.82/t mined.*
G&A = general and administrative; $/t = dollars per tonne; LOM = life-of-mine; $M/a = million dollars per annum

1.15 Economic Analysis

An economic evaluation of the Project was carried out incorporating all the relevant capital, operating, and sustaining costs, and federal, provincial, and mineral taxes. There are no royalties for the Project.

This PEA is preliminary in nature, and 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 Reserve, and there is no certainty that the PEA will be realized.

Table 1-4 summarizes the results of the economic analysis for the 19-year Project, with both pre- and post-tax results shown.

Table 1-4: Summary of Pre- and Post-Tax Results

Description	Value	Unit
MgO DBM Price	600	US$/t
Resources to the Plant	7,843	Kt
MgO Grade	43.27	%
MgO Produced	3,055	Kt
Non-Resource Material Mined	19,174	Kt
Strip Ratio	2.44	nr:r
Net Operating Income (EBITDA)	1,311,163	**($ x '000)**
Cash Costs (AISC)	351	**$/MgO t**
Capital Costs		
Initial Capital Incl. Contingency	235,885	($ x '000)
Sustaining Capital Incl. Contingency	3,935	($ x '000)
Total Capital Costs	**239,820**	**($ x '000)**
Working Capital	20,101	($ x '000)
Net Pre-Tax Cash Flow	1,051,242	($ x '000)

Description	Value	Unit
Pre-Tax		
NPV at 5%	$529,810	($ x '000)
IRR	24.5%	%
Payback	3.47	Years
Post-Tax		
NPV at 5%	$316,750	($ x '000)
IRR	19.3%	%
Payback	3.95	Years

Notes: *EBITDA = earnings before interest, tax, depreciation; NPV = net present value; % = percent; Kt = thousand tonnes; nr:r = non-resource:resource*

The principal assumptions used are shown in Table 1-5. The MgO DBM metal price scenario was used to prepare the economic analysis.

Table 1-5: Principal Assumptions

Parameters	Value	Unit
MgO Price[1]	600	US$/t
Exchange Rate[2]	0.77	US$:C$
MgO Recovery	90.0	%
Mining Costs[3]	30.30	$/t processed
Transportation: Mine to Plant	43.95	$/t processed
Processing + G&A	62.06	$/t processed
Discount Rate	5	%

Source: *AKF (2018)*

Notes: ***1.*** *MgO DBM price is FOB Cranbrook, BC.* ***2.*** *Exchange rate based on three -year trailing average from the Bank of Canada, as of January 2018.* ***3.*** *Mining Cost is based on $8.82/t mined.*
US$/t = United States dollars per tonne; G&A = general and administrative; $/t dollars per tonne

Figure 1-3 illustrates the post-tax undiscounted annual cash flow and tax schedule.

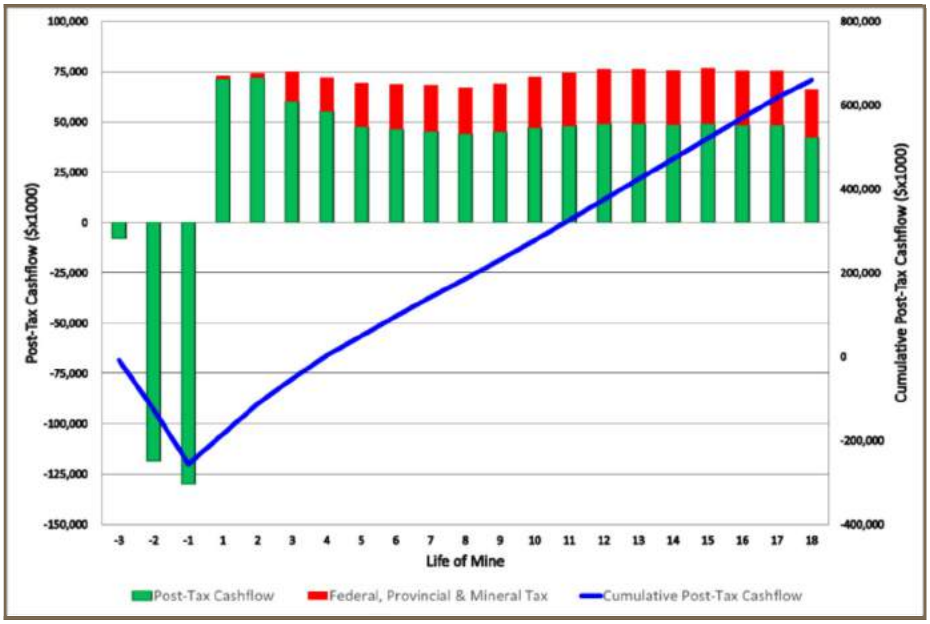


Figure 1-3: Post-Tax Undiscounted Cash Flow and Tax Scheduled

Sensitivity analysis for net present value (NPV), internal rate of return (IRR), and discount rates were carried out on the following parameters, as shown in Figure 1-4, Figure 1-5, and Table 1-6.

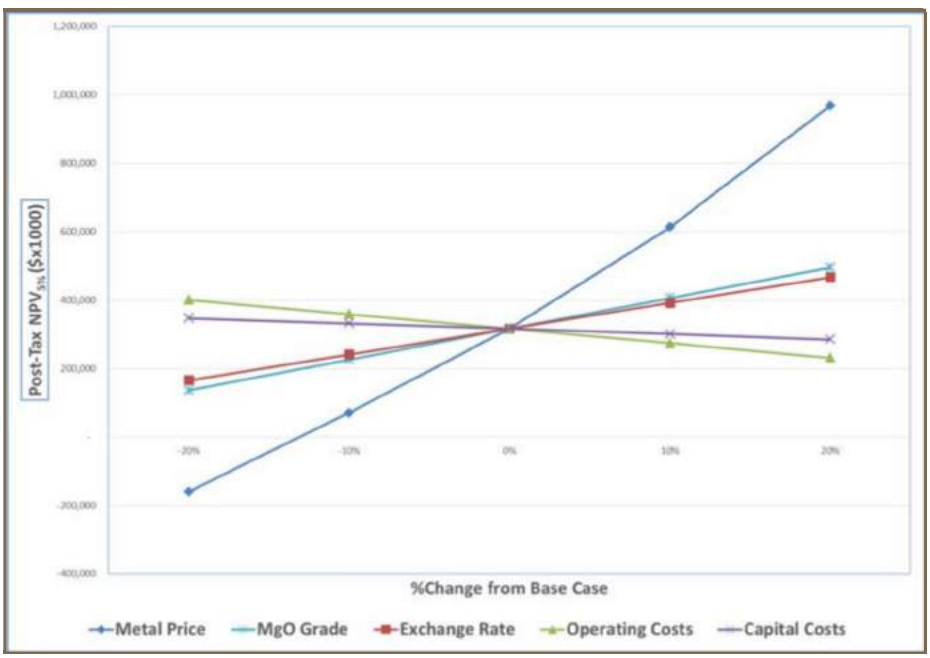


Figure 1-4: Post-Tax NPV at 5% Sensitivity Analysis

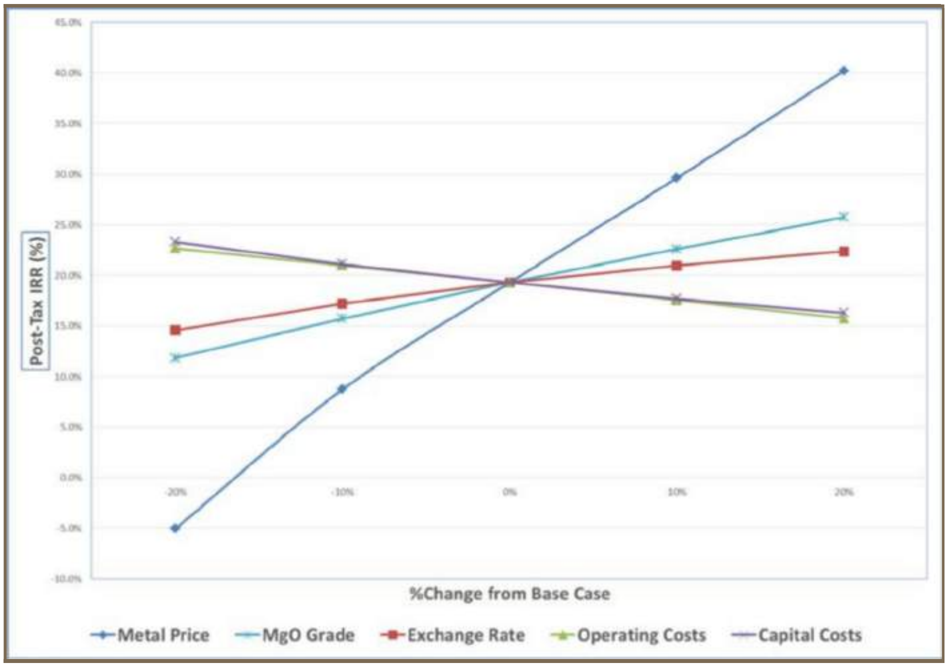


Figure 1-5: Post-Tax IRR Sensitivity Analysis

Table 1-6: Discount Rate Post-Tax Sensitivity

Discount Rate	Post-Tax NPV $M
0%	659,426
5%	316,750
8%	199,512
10%	142,854
12%	98,387

Notes: *NPV = net present value; $M = million dollars; % = percent*

1.16 Conclusions and Recommendations

In conclusion, the Project is economically viable with a positive post-tax $NPV_{5\%}$ of $316.7 million, an IRR of 19.3%, and payback of 4.0 years.

The Project has a planned 19-year LOM, and contains a 7.843 Mt resource to the plant, grading at 43.27% MgO, using a 42.5% MgO cutoff grade. The non-resource material will be 19.174 Mt, with a strip ratio of 2.4:1. This Project can be mined by conventional quarry methods, and recovered using processing methods consisting of crushing, grinding, flotation upgrading, calcination, and sintering, to produce saleable DBM and CCM products.

It is recommended the Project proceed to the next level of evaluation, either a prefeasibility or feasibility study stage, and that environmental baseline studies and a socioeconomic study program be initiated as soon as practical.

Estimated costs for a PFS-level study for this Project total $8.68 million.

2 INTRODUCTION

2.1 Terms of Reference

The resource estimate (Section 14), effective December 31, 2016, was prepared by Tuun Consulting Inc. (Tuun) to supplement this NI 43-101 compliant Preliminary Economic Assessment (PEA) Technical Report of the magnesite resources at MGX Minerals Inc.'s (MGX, or the Company) Driftwood Creek Magnesite Deposit Project (the Project or the Driftwood Creek Project) near Brisco, BC, Canada.

Mr. Allan Reeves, P.Geo., of Tuun is responsible for the resource estimate, which was prepared in compliance with the Canadian Securities Administrators' NI 43-101 and Form 43-101F and for Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, and 14 (except for sub-section 14.13).

Mr. Antonio Loschiavo, P.Eng., of AKF Mining Services Inc. (AKF), is responsible for the preparation of the following sections of this PEA: 1, 2, 3, 14.13, 15, 16, 18, 19, 20, 21 (except for 21.3, but include sub-sections 21.3.2 and 21.3.3), 22, 23, 24, 25, and 26.

Mr. Matt Bender, P.E., of Samuel Engineering (Samuel) is responsible for Section 13, 17, 21.3 (except for sub-sections 21.3.2 and 21.3.3), and 21.4.3.

2.2 Scope of Work

The updated Mineral Resource estimate and PEA presented in this report was commissioned by MGX.

2.3 Statement of Independence

The authors of this report have no beneficial interest in the outcome of the technical assessment. The fee for completing this report is based on normal professional rates plus reimbursement of incidental expenses. The payment of the professional fees is not contingent on the outcome of the report.

2.4 Site Visits

Mr. Reeves visited the Project site and Vine Creek core storage facility from June 9 to June 11, 2016.

Mr. Loschiavo visited the Project site on multiple occasions; his last site visit was conducted on June 9 to June 11, 2016.

2.5 Units and Currency

Unless otherwise stated, all units used in this report are metric. Assay values are reported in percentages. The currency, unless otherwise stated, is Canadian dollars.

2.6 Sources of Information

This report is based in part on internal Company documents, published government reports, and public information, as listed in Section 27, References.

The authors have not conducted detailed land status evaluations, and have relied upon previous qualified reports, public documents, and statements by the Company regarding Property status and legal title to the Project.

2.7 Units of Measure, Calculations, and Abbreviations

A list of units of measure, abbreviations, and acronyms used throughout this report is presented in Table 2-1 and Table 2-2.

Table 2-1: Units of Measure

Units of Measure	Description
A	ampere
cm	centimetre
m^3	cubic metre
°C	degree Celsius
dmt/h	dry metric tonne per hour
g/cm^3	gram per cubic centimetre
ha	hectare
hp	horsepower
h	hour
kg	kilogram
km	kilometre
KPa	kilopascal
kt	kilotonne (thousand tonnes)
kW	kilowatt
KWh	kilowatt hour
L	litre
MPa	megapascal
m	metre
µm	micron (micrometre)
mph	miles per hour
mm	millimetre
M	million
Mt	million tonnes

Units of Measure	Description
Mt/a	million tonnes per annum
min	minute
ppb	parts per billion
ppm	parts per million
lb	pound
s	second
km^2	square kilometer
m^2	square metre
t	tonne
t/d	tonnes per day
t/h	tonnes per hour
V	volt
W	watt
wmt	wet metric tonne

Table 2-2: List of Abbreviations and Acronyms

Abbreviations and Acronyms	Description
amsl	above mean sea level (metres)
ABA	acid base accounting
ARD/ML	acid rock drainage / metal leaching
AKF	AFK Mining Services Inc.
AGAT	AGAT Laboratories Ltd.
AB	Alberta
AISC	all-in sustaining costs
ALS	ALS Minerals Laboratory
As	arsenic
ARIS	assessment report indexing system
AACE	Association for the Advancement of Cost Engineering (currently AACE International)
APEG BC	Association of Professional Engineering Geologists of British Columbia
AAS	atomic absorption spectrometer/spectrometry
ADA	azimuth dip azimuth
BC EAA	BC Environmental Assessment Act
BC MEM	BC Ministry of Energy and Mines
BC MEMPR	BC Ministry of Energy, Mines, and Petroleum Resources

Abbreviations and Acronyms	Description
CEAA	Canadian Environmental Act
the Agency	Canadian Environmental Assessment Agency
CIM	Canadian Institute of Mining
Canoxy	Canadian Occidental Petroleum Ltd.
CPR	Canadian Pacific Railway
XMG	Canadian Securities Exchange
CAPEX	capital expenditure
X,Y,Z	Cartesian Coordinates, also Easting, Northing, and Elevation
CCM	caustic calcined magnesia
CRM	certified reference materials
Chem	ChemCognition LLC
CAGR	compound annual growth rate
DBM	dead burned magnesia
DMS	dense medium separator
DDH	diamond drill hole
DGPS	Differential Global Positioning System
DTH	down-the-hole
the Project / Driftwood Creek Project	Driftwood Creek Magnesite Deposit Project
DH	drill hole
DS-TMF	dry-stack tailings management facility
EBITDA	earnings before interest, tax, depreciation and amortization
EBITDA	earnings before interest, tax, depreciation and amortization
E	east
El. asl	elevation above sea level
EPCM	engineering, procurement, and construction management
EAO	Environmental Assessment Office
EMA	Environmental Management Act
EMP	environmental management plan
EU	European Union
FS	feasibility study
FSR	forest service road
FEL	front-end-loader
Whittle	Gemcom Whittle – Strategic Mine Planning™
G&A	general and administrative
GPS	Global Positioning System

Abbreviations and Acronyms	Description
HMS	heavy media separation
ha	hectare
H:V	Horizontal to vertical
IP	Induced Polarization
ICP-AES	inductively coupled plasma atomic emission spectroscopy
IRR	internal rate of return
IRR	internal rate of return
IDS or ID2	inverse distance square
kt	kilotonne
lat.	latitude
LG	Lerchs-Grossman
LOM	life-of-mine
LiDAR	Light imaging, Detection, and Ranging
long.	longitude
LOI	loss on ignition
$MgCO_3$	magnesium carbonate
MgO	magnesium oxide
MgO%	magnesium oxide percent
MMPO	Major Mine Permitting Office
MN	Manitoba
Manto	Manto Gold Corp.
MoU	memorandum of understanding
MGX	MGX Minerals Inc.
Ma	million years ago
MDRC	Mine Development Review Committee
MRC	mine review committee
MTO	mineral titles online
FLNRORD	Ministry of Forests, Lands, Natural Resource Operations, & Rural Development
MHF	multi-hearth furnace
NI 43-101	National Instrument 43-101
NN	nearest neighbour
NPV	net present value
NPV	net present value
NAG	non-potential acid generating
N	north
NOW	notice of work

Abbreviations and Acronyms	Description
OK	ordinary kriging
%	percent
PDH	percussion drill holes
pers.comm.	personal communication
pop.	population
PAX	potassium amyl qanthate
PFS	prefeasibility study
PA	preliminary assessment
PEA	preliminary economic assessment
PwC	PricewaterhouseCoopers
PFD	process flow diagram
QP	qualified person
QA/QC	quality assurance / quality control
QMS	quality management system
Q-Q	quantile-quantile
RPR	reviewable projects regulation
RMF	rock management facility
ROM	run-of-mine
SGS	SGS Lakefield Research
S	south
SG	Specific Gravity
St. Dev.	standard deviation
TFFE	targets for future exploration
Torch River	Torch River Resources
Tusk	Tusk Exploration Ltd.
Tuun	Tuun Consulting Inc.
US	United States
USGS	United States Geological Survey
UTM	Universal Transverse Mercator
US$	US dollars
VSK	vertical shaft kiln
VWP	vibrating wire piezometer
W	west
WRA	whole rock analysis
XRF	X-ray refractory

3 RELIANCE ON OTHER EXPERTS

The authors have relied on information contained in publicly available assessment reports obtained from the BC Ministry of Energy and Mines (BC MEM) Assessment Report Indexing System (ARIS). The work conducted for those reports was under the supervision of registered Association of Professional Engineering Geologists of British Columbia (APEG BC) professionals following industry best practices applicable at the time.

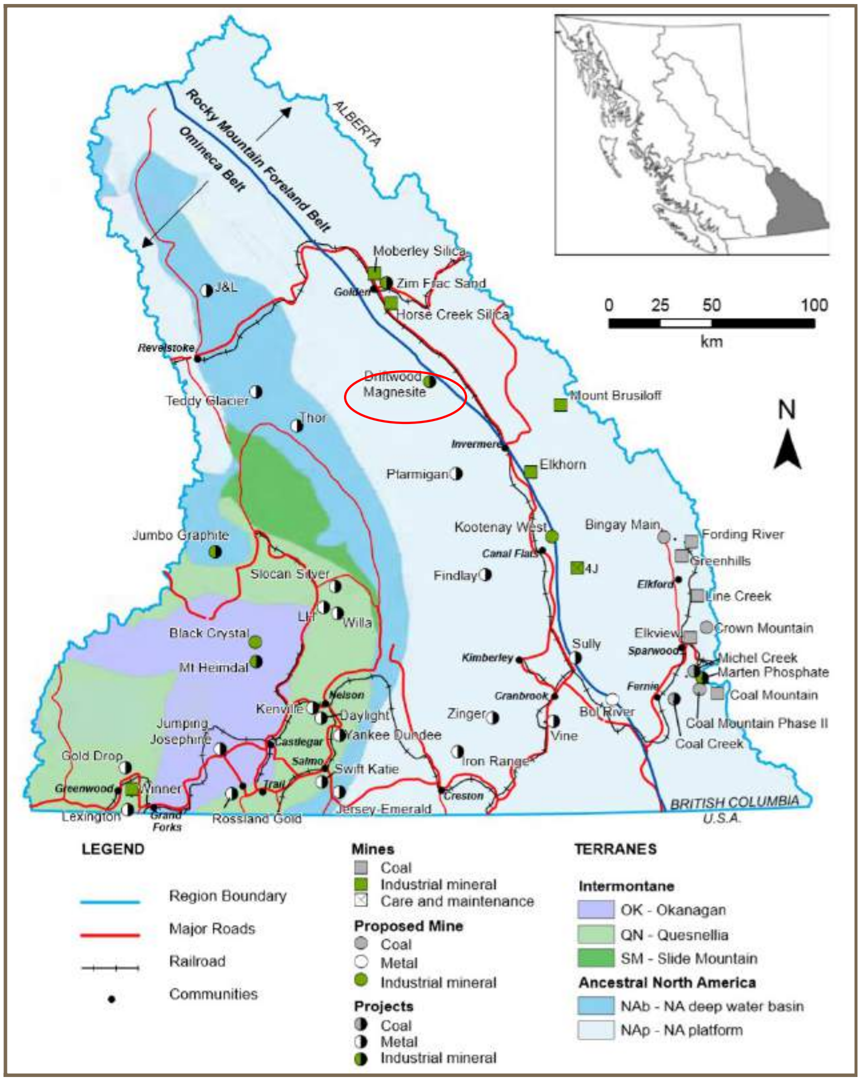
The authors have not verified the legality of any underlying agreement(s) that may exist concerning licenses or other agreement(s) between third parties, but has relied on MGX Minerals Inc. (MGX)'s solicitor to have conducted the proper legal due diligence. Information on the legal agreement between MGX and the Owners was provided by co-owner Andris Kikauka, P.Geo.

Tuun Consulting Inc. (Tuun) notes that the key magnesia industry guidebook '*The Chemistry and Technology of Magnesia'* is available online (Shand, M.A., 2006).

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Description

The Project is located approximately 53 km southeast of Golden, BC, and approximately 164 km northwest of Cranbrook, BC (Figure 4-1). The property is located on National Topographic System (NTS) map sheet 082K/15E, and on TRIM satellite map sheet 082K 098.



Source: *Katay, F.: Exploration and mining in the Kootenay-Boundary Region, BC*

Figure 4-1: Driftwood Creek Location Map

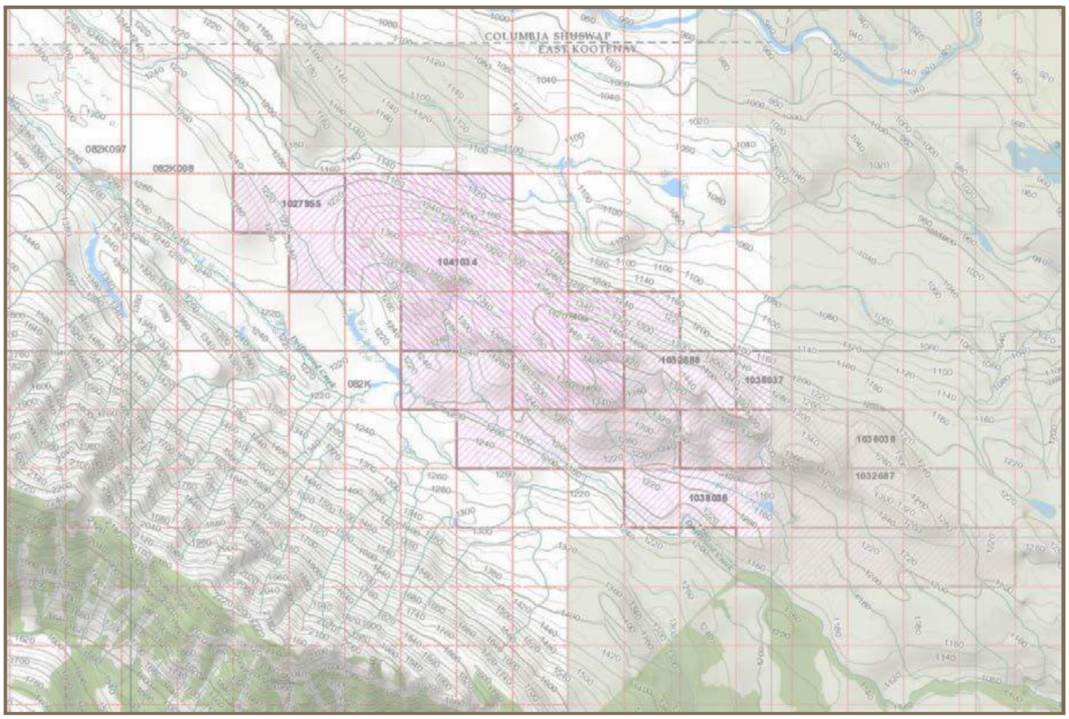
Figure 4-1 shows the location of the Project (red oval) at latitude 50°54'16" N and longitude 116°34'34" W, and its access to both highway and rail to Cranbrook, some 160 km south of the site. Also noted is the Vine Creek Core Storage facility just south of Cranbrook.

Mount Brussilof Magnesite Mine is approximately 62 km to the east-southeast of Driftwood Creek.

4.2 Mineral Tenure

The Project covers part of a prominent isolated ridge that trends about 115° azimuth between Driftwood Creek to the south and Bobbie Burns Creek to the north. The topography is moderate except for the magnesite itself, which locally forms steep cliffs more than 15 m (50 ft) high on both the north and south sides of the deposit. East of the claims and the magnesite, the host dolomite continues as a prominent ridge. Elevations on the claim block range from 1,190 m amsl to 1,370 m amsl.

The Driftwood Creek Magnesite claim group consists of six claims and one mining lease forming a contiguous mineral tenure that is located within the Golden Mining Division (Figure 4-2).



Source: *BC Mineral Titles Online Mineral Map Viewer*

Figure 4-2: Mineral Tenures Map

The total area of the seven mineral tenures that comprise the property is 835.44 ha (2,064.42 acres). Table 4-1 lists the details of the mineral tenures.

Table 4-1: Driftwood Creek Mineral Tenures

Tenure No.	Claim Name	Issue Date	Good To Date	Area (ha)
1027955	Driftwood Road W	2014-Apr-30	2024-Apr-30	61.13
1032687	Driftwood East B	2014-Sep-07	2024-Sep-07	122.31
1032688	Driftwood East A	2014-Sep-07	2024-Sep-07	61.14
1038036	Driftwood South Block	2015-Aug-18	2025-Aug-18	285.40
1038037	MGX N Corner 1	2015-Aug-18	2025-Aug-18	20.38
1038038	MGX N Corner 2	2015-Aug-18	2025-Aug-18	20.38
1041034	(Mining Lease)	2016-Jan-06	2017-Jan-06	264.70

Notes: *ha = hectare*

Details of the status of tenure ownership for the Driftwood property were obtained from the Mineral Titles Online (MTO) electronic staking system managed by the Mineral Titles Branch of the Province of British Columbia. This system is based on mineral tenures acquired electronically online using a grid cell selection system. Tenure boundaries are based on lines of latitude and longitude. There is no requirement to mark claim boundaries on the ground, as these can be determined with reasonable accuracy using a Global Positioning System (GPS). The Driftwood Creek Magnesite claims have not been surveyed.

Information posted on the MTO website indicates that the mining lease (mineral tenure 1041034) is owned 33% by Peter Klewchuk, 33% by Glen Munro Rodgers, and 34% by Andris Arturs Kikauka. Mineral tenures 1027955, 1038036, 1038037, and 1038038 are 100% owned by Mr. Rodgers. Mineral tenures 1032687 and 1032688 are owned 100% by Mr. Kikauka.

4.3 Risks Affecting Legal Access, Title, or Ability to Perform Work

In BC, a mineral title conveys the right to use, enter, and occupy the surface of the claim or lease for the exploration and development or production of minerals or placer minerals, including the treatment of mineralized material and concentrates, and all operations related to the business of mining, providing the necessary permits have been obtained.

The author's review of documentation shows that the mineral tenures have been kept in good standing by conducting required assessment work. Prior to conducting any work, such as drilling, trenching, bulk sampling, camp construction, access upgrading or construction, or geophysical surveys using live electrodes Induced Polarization (IP) on a mineral property a notice of work (NOW) permit application must be filed with and approved by the BC MEM. The filing of the NOW initiates engagement and consultation with all other stakeholders, including First Nations.

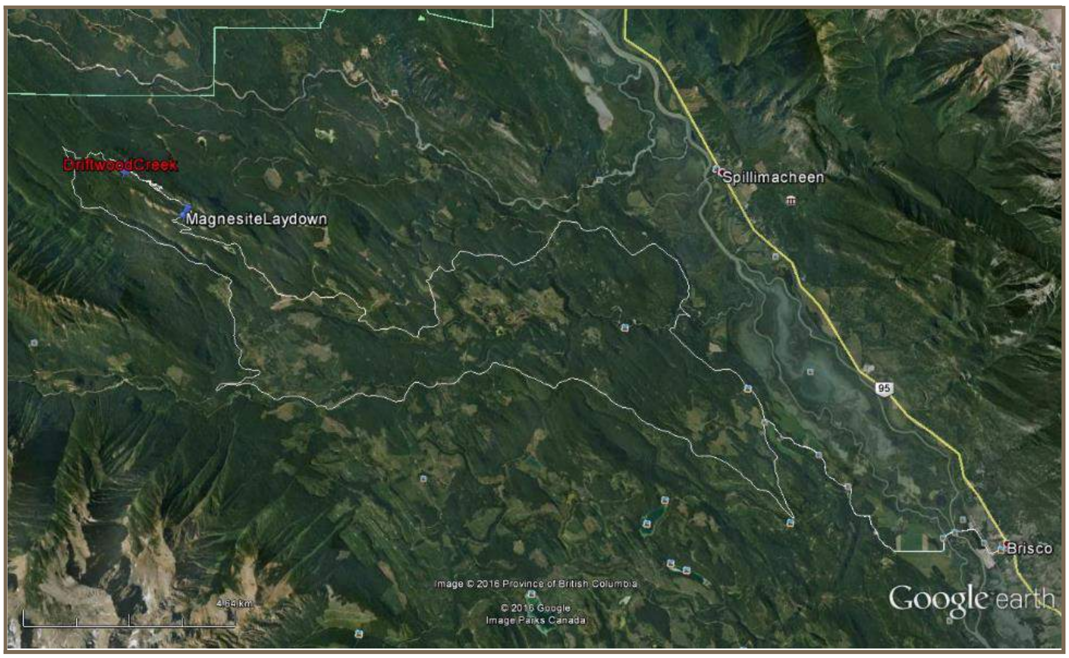
During the site visit, it was noted that surficial disturbances have been minimal, and would not be considered an environmental liability. The author has no reason to believe that there are any risks that would negatively affect the ability of MGX to perform work on the Project.

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

5.1 Accessibility

The Project can be accessed by logging road from either Brisco or Spillimacheen, both of which are located on paved Interprovincial Highway 95 to the east of the Project. From Brisco, the Bugaboo Creek and Driftwood Creek Forest Service Roads (FSRs) are followed for about 39 km. From there, a 1 km access trail leads onto the western edge of the magnesite deposit.

Figure 5-1 shows a second (approved) road constructed to allow access from two directions.

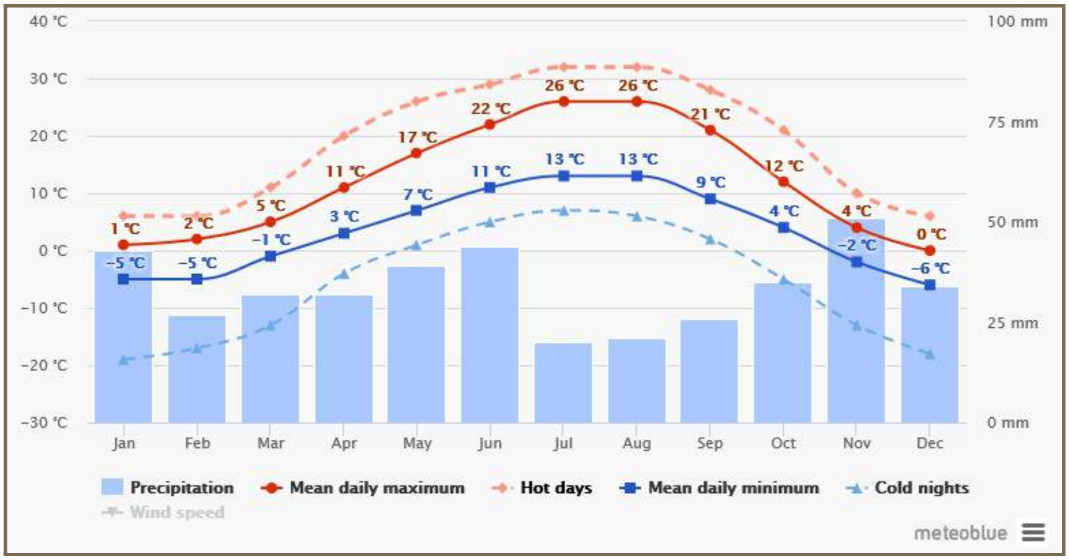


Source: *Google Earth*

Figure 5-1: Forest Service Road Access to the Project

5.2 Climate

Vegetation on the property consists mainly of lodgepole pine, lesser Douglas fir, and western yellow larch, along with minor birch and aspen. The seasonal climate is moderate, with snow in December–January, or earlier at higher levels (Figure 5-2). Work for the Project has primarily focused on the summer season.



Source: *https://www.meteoblue.com/en/weather/forecast/modelclimate/spillimacheen_canada_6153685*

Figure 5-2: Average Temperature and Rainfall of Spillimacheen Area

5.3 Local Resources

The nearest towns are Brisco and Spillimacheen on Highway 95, but both have very limited resources. The nearest population centres with significant services are Golden to the northwest and Invermere to the southeast. The property is 53 km by air from Golden, and 57 km by air from Invermere. Radium Hot Springs (pop. ≈900) is also close to the property, but it is primarily a tourist town with limited services.

Golden (pop. ≈4,200) is a road distance of about 97 km away, with Invermere (pop. ≈3,000) a road distance of approximately 67 km. Both Golden and Invermere have hotels, grocery stores, hardware stores, gas stations, medical services, and heavy equipment service companies that work in the logging and quarrying industries. Helicopter charters are also available from both communities.

5.4 Infrastructure

Both Golden and Invermere are on paved Interprovincial Highway 95, which has a parallel Canadian Pacific Railway (CPR) spur line serving the southeast BC coal fields that run up the Southern Rocky Mountain Trench and along the Columbia River (Figure 5-1). Golden is on the Trans-Canada Highway and the CPR main line. A power transmission line parallels Highway 93/95, and is located approximately 14 km due east of the Driftwood Creek property.

5.5 Physiography

Magnesite weathers prominently, and the Driftwood Creek deposit is well exposed as an isolated ridge at an elevation of 1,250 m amsl within a relatively low valley bottom topography. Numerous cliff exposures are present, with some cliff walls greater than 15 m (50 ft) high. A series of cross-cutting faults produce some offset of geologic contacts, but displacement is minor (pers. comm., A. Kikauka).

6 HISTORY

The following history is based on information contained in readily available public assessment reports that have been filed with the Province of British Columbia. The reports were prepared by both, previous and current property operators, and are listed in Section 27, References. Some selected figures have been incorporated from those reports as reasonable depictions of the results of previous exploration efforts following industry best practices of the times.

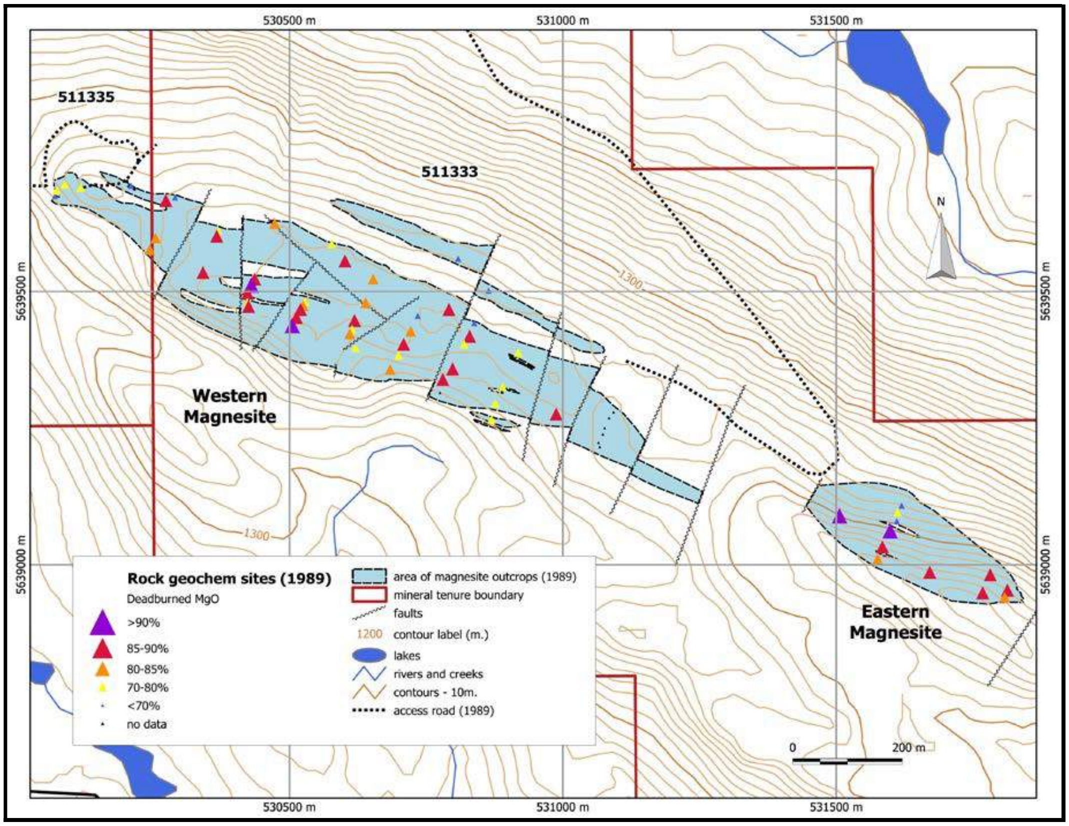
Magnesite was first discovered in the Brisco area in the 1960s and a series of small deposits are described by McCammon (1965) in British Columbia Minister of Mines Annual Report for 1964. The Driftwood Creek Deposit is not included in McCammon's summary, but was evidently discovered about this time as it was first staked in 1968.

In 1978, Kaiser Resources Ltd. (predominantly a coal-mining company) acquired the Driftwood Creek deposit and carried out a program of surface geologic mapping and some very minor and poorly documented diamond drilling. Publicly available reports indicate some minor diamond drilling was done, but no data has been located. According to Rodgers (1989), Kaiser also drilled 12 short holes using a small plugger-type drill between 0.6 m to 2.0 m deep in order to test near-surface purity. The location of this drilling is assumed to be over the East Zone. The property was held for ten years, and then the claims were allowed to expire.

In 1987, the Driftwood Creek Magnesite Deposit was staked by Canadian Occidental Petroleum Ltd. (Canoxy). In 1989, a 2,500 m baseline was established at azimuth 115° that was parallel to the magnesite area (Rodgers, 1989). Cross lines at a 100 m spacing were established across the magnesite, and ranged from 50 m to 500 m in length. The lines were flagged at 50 m intervals. This survey grid was used to do geological mapping and build cross-sections at 1:2,000 and 1:1,000 scales.

As part of the geologic mapping program, a total of sixty-eight 5-km samples of magnesite were also collected along 17 cross-section survey lines (Figure 6-1). Samples were analyzed by Chemex Laboratories Ltd. (Chemex), Vancouver, BC. The analyses were done for SiO_2, Al_2O_3, Fe_2O_3, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, MnO, BaO, and loss on ignition (LOI).

In 1990, Canoxy drilled four NQ diamond holes totalling 219.8 m. This drilling also targeted the Eastern Magnesite deposit. Drill core was split on site, and samples taken at 1.5 m intervals. Only sections through the magnesite were sampled. The core samples were shipped to Chemex in North Vancouver, and were analyzed for major oxides and LOI. As well, a "dead-burned" assay was done for each sample. This involved analysis for MgO% after roasting at 1,000°C for one hour. Canoxy held the claims for 10 years with no additional work, and they were then allowed to lapse.



Source: *D.G. MacIntyre, July 2014.*

Figure 6-1: Dead-burned MgO% Rock Geochemical Samples (after Rodgers, 1989)

In 1999, the magnesite ridge was staked by the present owners, and some additional rock geochemistry was completed on part of the Western Magnesite (Kikauka, 2000). This work involved sampling along north- and northeast-trending lines over exposed outcrop in ten locations within a 325 m x 125 m area (Kikauka, 2000). Weighted average values ranged from 41.1% to 45.5% MgO, and 0.4% to 8.3% SiO_2.

Additional geochemistry was conducted in 2001, along with bulk sampling and access trail construction (Klewchuk, 2002). Twenty samples collected in 2001 provided the following range of values shown in Table 6-1. Follow-up rock sampling and drilling results have been similar.

In 2008, SGS Lakefield Research (SGS) conducted some preliminary metallurgical testwork to test beneficiation for samples from the East and West zones of the Driftwood Creek Magnesite Deposit (Rodgers, 2008). Details are provided in Section 13.

Table 6-1: Range of values for Magnesite Analyses

Oxide	Range of Values (%)
MgO	39.98 to 44.42
SiO_2	2.48 to 13.1
$A1_2O_3$	0.05 to 1.11
Fe_2O_3	0.71 to 1.11
CaO	0.34 to 3.21
TiO_2	<0.01 to 0.1
P_2O_5	0.09 to 0.19
MnO	0.02 to 0.04
Cr_2O_3	001 to 0.12

Source: *Klewchuk, 2002*
Notes: *% = percent; < = less than*

In the fall of 2008, a program of trail access construction and diamond drilling was also completed on the property. This work was under the direction of Peter Klewchuk, P.Geo., one of the property owners, on behalf of Tusk Exploration Ltd. (Tusk) of Vancouver, BC. Trails were constructed from the existing access at the west end of the magnesite ridge onto the Western Magnesite where the thickest zone of magnesite exists, and an additional trail was constructed to access the Eastern Magnesite. In total, about 3,300 m of trails were constructed.

In late October and early November 2008, seven NQ diamond drill holes were completed from an area near the thickest part of the Western Magnesite, for a total of 692 m of diamond drilling. Core from this drilling was bagged and prepared for shipment to a lab, but was never submitted. This core was subsequently analyzed in 2012 by Torch River Resources (Torch River), who were considering an option on the property. Torch River decided not to proceed with the option.

By 2013, the property was under agreement with Manto Gold Corp. (Manto), which filed a Notice of Work for 2014 diamond drilling. In late 2013, MGX signed an arrangement with Manto and on July 7, 2014, the agreement was completed, making Manto a subsidiary.

MGX conducted the diamond drill program, and followed up with a re-assay of the 2008 drill core. In 2015, MGX conducted another diamond drill campaign, and received permission to build a short access road. The primary purpose of the road was to provide a secondary access route and shorten the haul distance for a proposed bulk sample.

In 2016, MGX drilled and blasted a bulk sample site at the East Zone and hauled approximately 100 tonnes of material to a laydown area for stockpiling, sorting, then hauled via highway truck to a crusher in Invermere, BC. Sixteen 25-L pails of sample materials (taken on a grid over the stockpile) were shipped to ALS Minerals (ALS) in North Vancouver for assaying and specific gravity (SG) analysis. A pilot plant was acquired, which is currently being shipped on six tractor-trailers from the Yukon to Invermere, BC. The pilot plant is a 50 t/d flotation plant previously used for concentrating lead-zinc-bearing material. Once the plant is set up, the crushed bulk material will be processed to remove silica and calcium impurities and then re-assayed. Approximately 10 tonnes of the concentrate material will then be shipped to Industrial Furnace Co. of Rochester, NY, for calcining and kiln testing to produce CCM and DBM samples. Information gained from the testing will be used for kiln design.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geologic Setting

The area of the Driftwood Creek Magnesite Deposit was first mapped by Reesor (1973), although the magnesite deposits west of Brisco are not included in his work. The following regional geologic information is extracted from Simandl and Hancock (1991).

The Brisco and Driftwood Creek deposits are situated west of the Southern Rocky Mountain Trench fault (Figure 7-1). They are hosted by dolomites of the Middle Proterozoic (Helikian) Mount Nelson Formation of the Purcell Supergroup within the Purcell anticlinorium. Stratigraphic sections applicable to the area of the magnesite deposits were established by Walker (1926), Reesor (1973), and Bennett (1985). The geology of the Toby and Horsethief Creek areas has been described by Pope (1989, 1990). The upper part of the Mount Nelson Formation hosts the Project, and only the Mount Nelson and Toby Creek formations are described in this report.

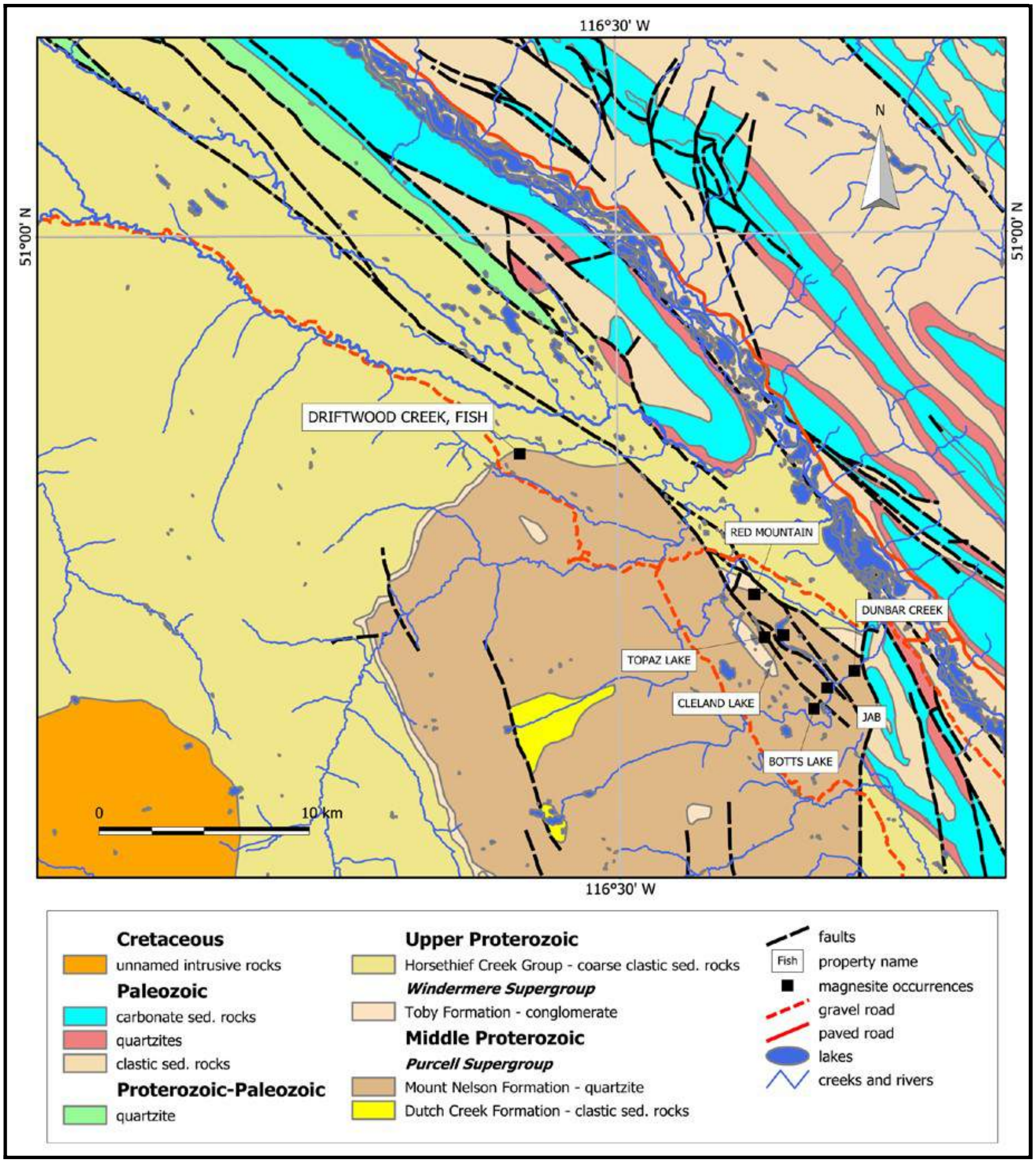
The Mount Nelson Formation is separated from the overlying Toby Formation of the Windermere Supergroup (Hadrinian) by an unconformity (Reesor, 1973; Pope, 1989). This unconformity records the East Kootenay orogenic event, which consisted of regional uplift and thermal metamorphism dated at 750–850 Ma and submarine volcanic activity within the Purcell anticlinorium (Pope, 1989).

The magnesite deposits are located within an area affected by low-grade regional metamorphism (Reesor, 1973; Bennett, 1985). All known magnesite occurrences are located outside the contact metamorphic aureole of Mid-Cretaceous intrusions.

In the Toby-Horsethief Creek map area, the Mount Nelson Formation is at least 1,320 m thick, and is the uppermost unit of the Purcell Supergroup (Pope, 1990). It is divided into seven members (Figure 7-2). The descriptions below, in order from oldest to youngest, are summarized from Pope (1990).

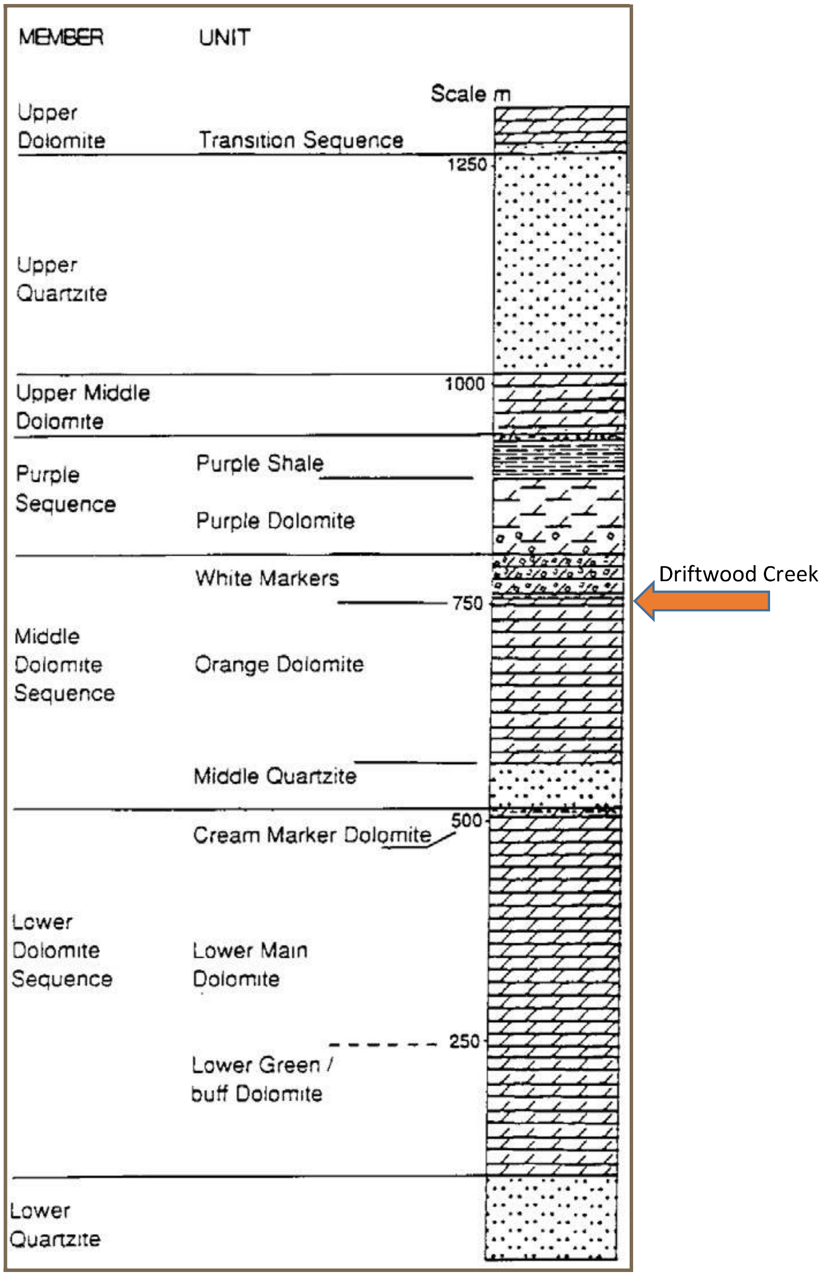
The "lower quartzite" is 50 m to 150 m thick, white, well sorted, thin-bedded (<20 cm), ripple laminated, fine- to medium-grained quartz arenite.

The "lower dolomite sequence" is characterized by its grey colour and light grey weathering surface, laminated beds 20 cm to 50 cm thick, soft sediment features, cryptalgal laminations, and laterally-linked hemispherical stromatolites. This dolomite also contains black argillite layers, 1 cm to 2 cm thick, and oölitic laminae. The top of the sequence is the cream coloured, cherty "cream marker dolomite," which is approximately 20 m thick.



Source: *D.G. MacIntyre, July 2014 (after Massey et al., 2005)*

Figure 7-1: Regional Geology – Driftwood Creek Magnesite Project



Source: *Pope, 1990*

Figure 7-2: Stratigraphic Column, Mt. Nelson Formation

The "middle dolomite sequence" comprises the "middle quartzite," "orange dolomite," and "white markers." The "middle quartzite" is characterized by its apple green colour. It consists of graded, cross bedded, and massive arenites, siltstones, and argillites. Beds are 10 cm to 50 cm thick, with undulate bases and truncated tops. The orange dolomite consists of well-bedded, silty, or light beige to dark grey dolomites, weathering orange-brown or orange-buff. Stromatolitic textures, cryptalgal lamination, chert intercalations, halite casts, solution-collapse breccias, and dewatering features have been described in this unit. The stromatolitic dolomite most commonly forms the footwall to the Driftwood Creek Magnesite Deposit (Simandl and Hancock, 1992).

The "white markers" sequence is less than 70 m thick and conformably overlies the orange dolomite. It consists of cream to medium grey dolomites, and locally contains white magnesite beds up to 1 m thick, as well as purple, green, and buff dolomitic mudstones and beds, with dolomite-replaced halite crystals. It is assumed that the Driftwood Creek magnesite deposit occurs at this stratigraphic level.

The "purple sequence" conformably overlies the white markers. It consists of dolomites as well as dolomitic siltstones and sandstones consisting of 20% quartz, 70% dolomite, and 10% hematite. These rocks contain halite casts, and grade upward into purple shales with green reduction spots. Several mud chip breccias and monomictic conglomerates occur within this sequence. The upper part of the purple sequence is referred to as the "purple shale unit." It consists of purple argillites with or without green reduction spots and laminae. The purple sequence is separated from the overlying upper middle dolomite by a conglomerate consisting of angular to rounded dolomite and quartzite clasts of variable dimensions, cemented by purple sandy argillite.

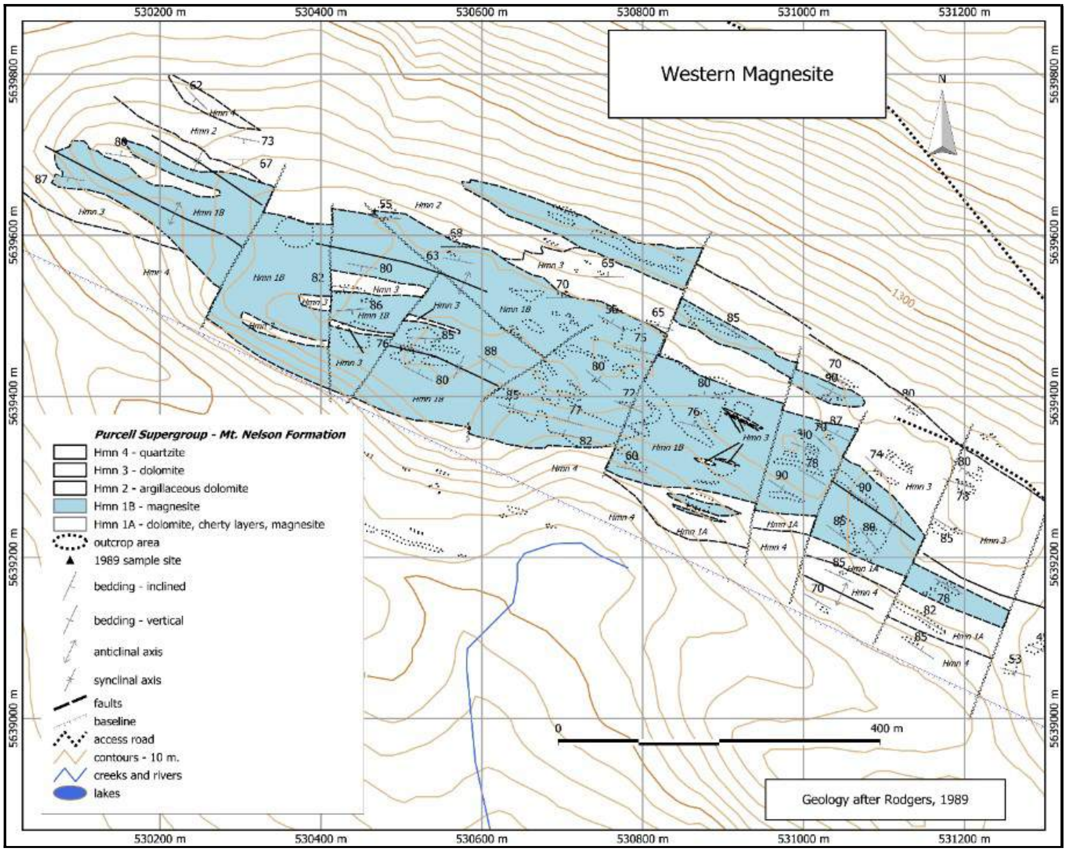
The "upper middle dolomite" is 80 m thick, and similar to the lower main dolomite; however, it contains abundant allochems (oncolites and oölite peloidal and pisolitic laminations) replaced by chert.

The "upper quartzite" is over 260 m thick. It is a cliff-forming, well-sorted, quartz-cemented, medium- to coarse-grained arenite, characterized by massive bedding, and poorly preserved sedimentary features.

The "upper dolomite" has a conformable gradational contact with upper quartzite. Pale beige to dark grey dolomite beds, 10 cm to 50 cm thick, are interbedded with quartz and dolomite-pebble conglomerates and dolomitic sandstones. The unit is characterized by abundant chert layers, cryptalgal structures replaced by black chert, and a distinctive, laminated, strongly contorted, and locally brecciated blue-grey dolomite. The contact with underlying quartzite is transitional, and consists of interbeds of purple argillite, quartzite, and dolomite.

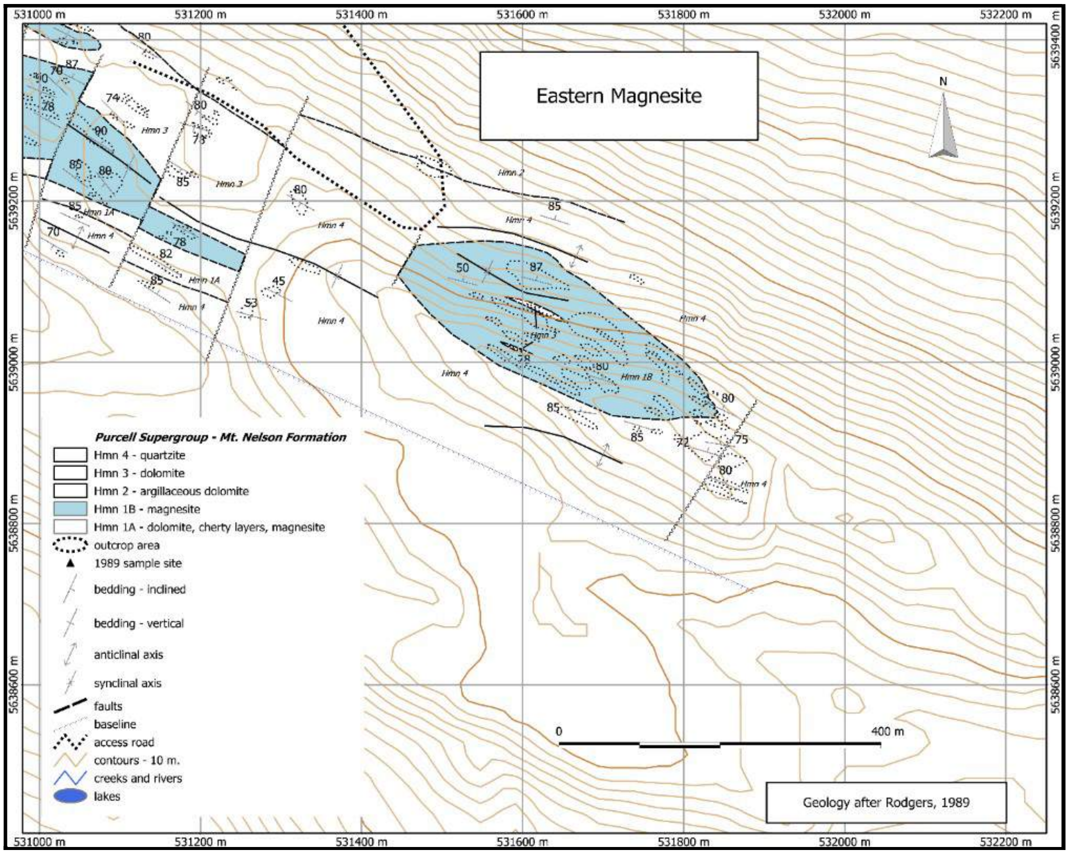
7.2 Structural Geology

The earliest tectonic event in the area responsible for the syncline/anticline development within the Purcell Supergroup is likely related to formation of the Rocky Mountain fold and thrust belt in Late Cretaceous to Early Tertiary time. The northwest-trending fault that parallels the Spillimacheen River, 4 km north of the claims (Rodgers, 1990), probably formed at this time. The magnesite ridge, which trends the same as the main syncline/anticline axes (115°) is frequently cut by north-northeast-trending cross-faults of uncertain age (Figure 7-3 and Figure 7-4). The series of NE-trending cross-cutting faults produce minor offset of geologic contacts (pers. comm. A. Kikauka).



Source: *D.G. MacIntyre, July 2014.*

Figure 7-3: Structural Mapping – West Zone



Source: *D.G. MacIntyre, July 2014*

Figure 7-4: Structural Mapping – East Zone

7.3 Property Geology

Reports by Morris (1978), Rodgers (1989), and Simandl and Hancock (1992) provide the best available geologic information on the Driftwood Creek magnesite deposit. Klewchuk (2002) provides additional detail for the Eastern Magnesite area.

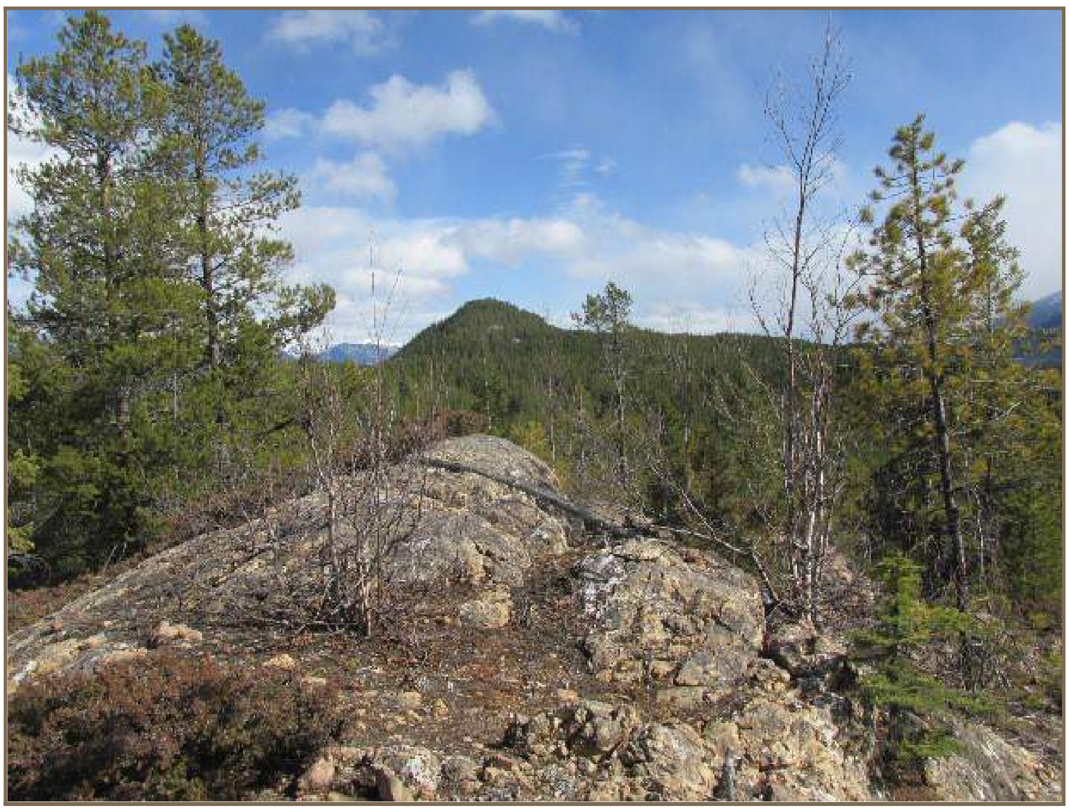
As previously noted, the Project is hosted by the Helikian (Precambrian) age Mount Nelson Formation, with the magnesite deposit occurring in the upper part of the formation.

According to Simandl and Hancock (1992), magnesite and sparry carbonate form stratabound lenses and pockets within the "white marker beds" subdivision of the "middle dolomite" unit of the upper Mount Nelson Formation at the Driftwood Creek property. This middle unit is called unit Hmn1, with the lower part, Hmn1A, describing the buff to light grey dolomite horizon. The upper part, Hmn1B, is applied to the magnesite.

The magnesite is either white, pale grey, or beige, and weathers buff. The unit is characterized by coarse to sparry crystals, and locally contains light green interbeds less than 1 cm in thickness. The interbeds are either regular or disrupted by growth of sparry magnesite crystals within the coarsest magnesite-rich zones (Simandl and Hancock, 1992). Vestiges of hemispherical stromatolites are observed locally in finer-grained magnesite-bearing rocks. Chert, quartz veinlets, and dolomite are the most common impurities especially within the lower part of the deposit. Calcite, pyrite, and talc are typically present in trace amounts. The abundance and proportion of impurities change irregularly, both along strike and across bedding (Simandl and Hancock, 1992).

Magnesite weathers prominently, and the Driftwood Creek deposit is well exposed as an isolated ridge within relatively low valley bottom topography, at an elevation of 1,250 m (Klewchuk, 2010). Numerous cliff exposures are present, with some cliff walls greater than 15 m (50 ft) high.

Magnesite has been mapped over a strike length of 1,900 m and maximum width of about 220 m (Klewchuk, 2010). The magnesite occurs at surface in two discrete bodies; a larger "West Zone" and the smaller "East Zone" (Figure 7-5).



Source: *Rockstone Research 2015*

Figure 7-5: West Zone Outcrop Foreground (East Zone Peak 1.5 km away)

Freshly broken magnesite is typically a milky white colour (Figure 7-6), but weathers to a pale yellow to slightly pinkish colour. Exposures of magnesite are commonly coated with a black lichen that appears to locally favour this rock type.

The host dolomite to the south of the Eastern Magnesite (Hmn2, Argillaceous dolomite) is a much darker buff to reddish-brown colour while the (silty and cherty) dolomite to the north of the thicker Eastern Magnesite is a medium grey colour. Where magnesite contacts with dolomite are exposed, they tend to be quite sharp and are easily recognized. Even where bedding transgressive contacts exist, the boundary tends to be fairly sharp (Klewchuk, 2010).

Source: *Tuun Consulting Inc.*

Figure 7-6: Sparry Pearl White Magnesite in DH2014-05

Texture of the magnesite is variable, ranging from fine- and medium-grained to very coarse-grained. Most of the deposit is of medium- and fine-grained texture, with irregular patches of more coarse-grained texture. Areas of coarse-grained magnesite appear to be irregularly developed within the area of exposed magnesite, and are not obviously related to any structure.

Thin quartz veins are irregularly distributed through the magnesite, in a near-ubiquitous manner, although the concentration of quartz veins does vary. There are areas with no apparent quartz, but these are not extensively developed. The more prominent quartz veins and quartz vein swarms tend to be oriented from N15°E to N60°E. Similar quartz veins are present in the host dolomite (seen mainly to the south of the Eastern Magnesite), indicating that these quartz veins are not related to development of the magnesite. It is possible that they are related to intrusives noted during drilling. These intrusives may have also introduced a second population of Fe_2O_3 that may be considered a potential impurity in the recovery of magnesite.

All the magnesite deposits in the Brisco and Driftwood Creek area are located within the upper half of the Mount Nelson Formation. Most are lenticular and seem to form chains, as illustrated by the Driftwood Creek deposits. All deposits are stratigraphically associated with red to purple dolomites, cherty dolomites, stromatolitic dolomites, dissolution breccias, and other rocks containing dolomite pseudomorphs after halite and lenticular gypsum crystals. Locally, stromatolitic textures are preserved, even within magnesite-bearing rocks. According to Simandl and Hancock (1992), most of the above features are indicative of the evaporitic depositional environment.

Figure 7-7: Lithology of Helikian Units at Driftwood Creek

Unit	Description
Hmn4	Quartzite: purple-green-grey-white; weathers red-green-grey
Hmn3	Dolomite: siliceous, grey/light grey to grey-buff; quartz veinlets/laminae
Hmn2	Argillaceous dolomite: dark red-rusty brown, fine-grained
Hmn1B	Magnesite: white-cream-buff coloured; fine- to coarse-grained
Hmn1A	Stromatolitic Dolomite: buff to light grey; fine-grained/thin-bedded

8 DEPOSIT TYPE

The Driftwood Creek and Brisco magnesite occurrences are classified as sparry magnesite deposits (E09) by the BC MEM (Simandl and Hancock, 1998). This deposit type is characterized by strata-bound (and typically stratiform), lens-shaped zones of coarse-grained magnesite, mainly occurring in carbonates, but also observed in sandstones or other clastic sediments. Magnesite exhibits characteristic sparry texture.

There are two preferred theories regarding the origin of sparry magnesite deposits:

- Replacement of dolomitized, permeable carbonates by magnesite due to interaction with a metasomatic fluid; and
- Diagenetic recrystallization of a magnesia-rich protolith deposited as chemical sediments in marine or lacustrine settings. The sediments would have consisted of fine-grained magnesite, hydromagnesite, huntite, or other low temperature magnesia-bearing minerals.

The main difference between these hypotheses is the source of magnesia; external for metasomatic replacement, and in situ in the case of diagenetic recrystalization. Temperatures of homogenization of fluid inclusions constrain the temperature of magnesite formation or recrystalization to 110°C to 240°C. In British Columbia, the diagenetic recrystalization theory may best explain the stratigraphic association with gypsum and halite casts, correlation with paleotopographic highs and unconformities, and shallow marine depositional features of the deposits (Simandl and Hancock, 1998).

Large-scale (regional) replacement and associated basinal brine, chert intercalations, halite casts, solution-collapse breccias, paleo-karst, and dewatering features at Driftwood magnesite are common features also found in Mississppi Valley lead-zinc deposit types. It is also believed that the stromatolitic biogenic “algal mat” structures at the base of the Driftwood magnesite deposit established a dome-like mass that may have contributed to thickening of the unit. Proterozoic age algal mat biogenic structures are genetically linked to world class mineral deposits in the DRC-Zambia Copperbelt and South Africa Witwatersrand paleo-placer gold-uranium deposits.

9 EXPLORATION

From 2000–2014, the Owners (Kikauka, Rodgers, and Klewchuk) have maintained the claim tenures by filing BC Assessment Reports as noted in Section 6 History. The work reported varied from sampling outcrop, building access trails, and diamond drilling.

MGX has accelerated exploration since July 2014. As noted, the work has consisted of:

- Diamond drilling in 2014 and 2015;
- Rock sampling across the deposit in both 2014 and 2015;
- Re-assay of the 2008 drill core;
- Construction of an alternate access road to the East Zone;
- Surveying of all 2008 to 2015 diamond drill holes along with the 2016 percussion drill holes used for the blasting of the bulk sample;
- Collection of approximately 100 tonnes of bulk sample from the East Zone for further test work;
- Purchase of a 50 t/d pilot plant; and
- Approximately 10 tonnes of the material will be shipped to Industrial Furnace Co. of Rochester NY for calcining and kiln testing to produce CCM and DBM samples.

The information will be used in the final kiln design and the final samples will be shipped to interested customers, primarily in the refractory industry—DBM that have contacted MGX.

9.1 Magnesite Rock Sampling

In 2000, Andris Kikauka, P.Geo., used a maul and mallet to take 45 rock chip samples. About 3 kg of rock chips were collected from a continuous chip channel along 3 m widths for each sample. Rock chip samples were taken from an area of about 6 ha, which was mapped at a scale of 1:500. Rock chip samples were bagged, tagged, and shipped to Pioneer Labs, New Westminster, BC, for multi-element whole rock geochemistry (AR# 26345).

In 2014, an additional 14 rock chip samples were collected by Kikauka (AR#35175), and sent to Pioneer for X-Ray refractory (XRF) lithium borate whole rock analyses. The rock chip samples were taken across 3-m intervals along exposures of bedrock in the West and East Zones. Rock chip samples were taken with rock hammer and chisel, and consist of acorn- to walnut-sized bedrock pieces, for sample weights ranging from 1.35 kg to 2.98 kg. Sample material was placed in marked poly mineralized material bags at 3 m intervals and shipped to ALS, in either Kamloops or North Vancouver. Samples submitted to ALS consisted of 142 split core, 14 rock chip, and 7 blank samples. The blank samples were inserted in the sample stream every 20 samples in order to verify data from the laboratory. The seven blank samples consisted of 0.84 kg to 1.4 kg sized rock chips from a nearly pure boulder of magnesite, and were inserted for quality assurance/quality control (QA/QC) protocol. ALS crushed, split, and pulverized samples using prep-31 code. This involves crushing to better than 70% passing a 2-mm screen. A split of 250 grams is pulverized to better than 85% passing a 75-µm screen. The sample pulps were analyzed using ME-XRF-06 (XRF-26), a lithium borate flux major oxide whole rock geochemical analytical method.

In 2015, Kikauka collected an additional 31 rock samples over the East and West Zones. Those samples were also sent to ALS in Kamloops for assaying.

Rock geochemistry totalled 85 samples across the Project area (≈1,900 m x 220 m, or about 488 ha), of which five were taken in Hmn1A dolomite, and the rest in Hmn1B magnesite. Table 9-1 summarizes the assay results in magnesite.

Table 9-1: Whole Rock Geochemistry Summary

Year	Laboratory	No. Samples	MgO (%)	Al_2O_3 (%)	CaO (%)	Fe_2O_3 (%)	SiO_2 (%)	LOI (%)
2000	Pioneer	40	43.80	0.46	1.34	1.44	2.34	50.26
2014	ALS-Kelowna	9	42.69	0.62	1.45	0.80	5.55	48.39
2015	ALS-Vancouver	31	41.99	0.31	2.76	1.13	4.67	48.61

Notes: *No. = num= ber; % = percent; LOI = loss on ignition*

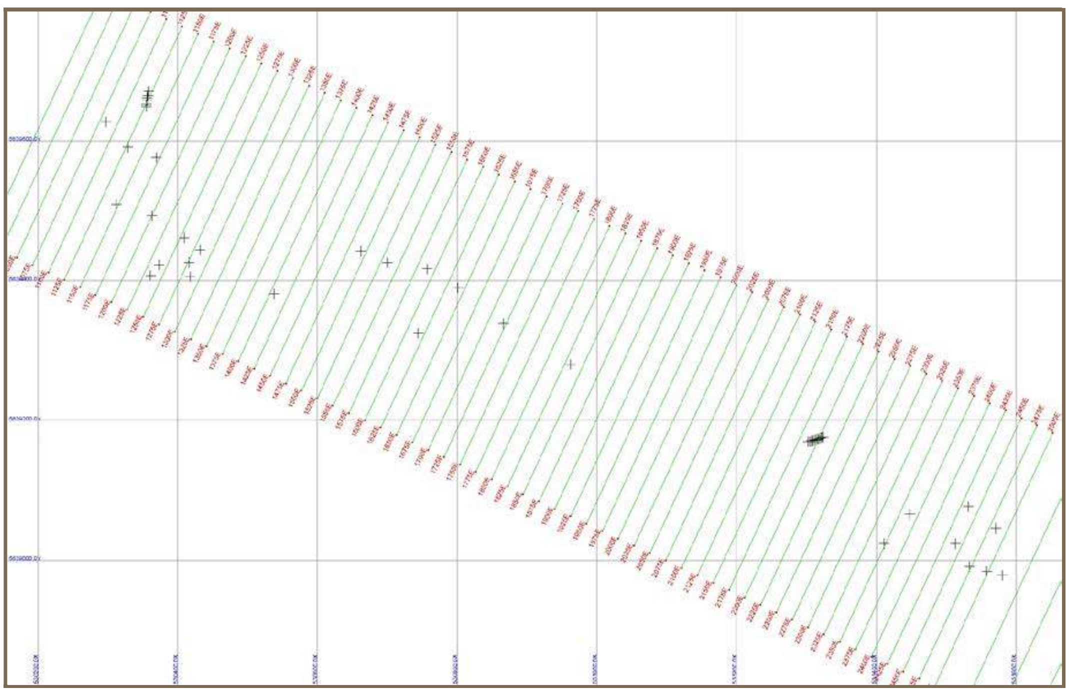


Figure 9-1: Rock Sample Locations

9.2 Road Construction

In 2008, trail access was constructed for the diamond drilling. Part of the improvements involved the construction of additional drill access trails for the 2014–2015 diamond drilling. In Figure 9-2, the southern main logging road is in blue, and accesses the Project at the NW side. From there, the Cat trails for drill access are also marked in blue. In 2016, the (yellow) connector road access was built to tie in with the (red) northern access logging road. The distance between the two zones is approximately 1 km.

Figure 9-2: Driftwood Creek Access Trails

9.3 Exploration Summary

The exploration work was undertaken by professionals registered with APEG BC using industry-standard practices of the time. The sampling practices were focused on outcrops representing the magnesite bed, and the assay results indicate no obvious bias. The sample locations and density are restricted to the two zones shown on Figure 9-2, which comprise a strike length of ≈1,900 m, ≈220 m wide (≈488 ha). The results confirmed the presence of magnesite and were used to define drill exploration programs.

10 DRILLING

Since 1990, a total of 3,654.16 m of diamond drilling in 49 drill holes, and 268.15 m of percussion drilling (25 holes), have been done on the Project, as summarized in Table 10-1.

Table 10-1: Driftwood Creek Diamond Drill Hole Summary

Company	Year	Zone	No. Holes	Type	Avg. Int. (m)	Metres
Canoxy	1990	East	4	NQ	1.54	219.76
Tusk	2008	West	7	NQ	2.13	692.00
MGX	2014	East	8	BTW	2.97	437.52
MGX	2015	West	14	BTW	2.95	1,093.38
MGX	2016	East	25	Perc.	3.05	268.15
MGX	2016	Both	16	BTW	2.93	1,211.50
Total			**74**		**2.73**	**3,922.31**

Notes: *BTW & NQ = Drill Rod sizes; No. = number; m = metres; Avg. Int. = average intervals*

The 1990 drilling by Canoxy targeted the East Zone magnesite. Tusk conducted the 2008 diamond drilling that targeted the West Zone.

MGX took over the option and in 2014–2015, diamond drill-tested both the East and West Zones.

In 2016, a bulk sample location on the East Zone was selected, and twenty-five 4" diameter percussion blast holes were drilled and sampled. Also in 2016, the WSP Group was contracted to survey all of the drill holes. The only holes that could not be verified by the survey were the four 1990 Canoxy drill holes; however, original documents were provided by Rodgers for review and verification.

The drill holes were collared within the magnesite bed and the majority of the assays (at 2 m to 3 m intervals) are near surface. Recoveries within the magnesite were good, and there are no known factors that would affect the accuracy and reliability of the results.

Figure 10-1 is an overview plan of the diamond drilling (the red-dashed line is the outcrop-mapped magnesite outline from Figure 7-3 and Figure 7-4). The section lines are at a 25 m spacing, apart from with some at 12.5 m spacings in the East Zone.

Figure 10-2 is a close-up view of the West Zone hole locations and Figure 10-3 a close-up view of the East Zone hole locations.

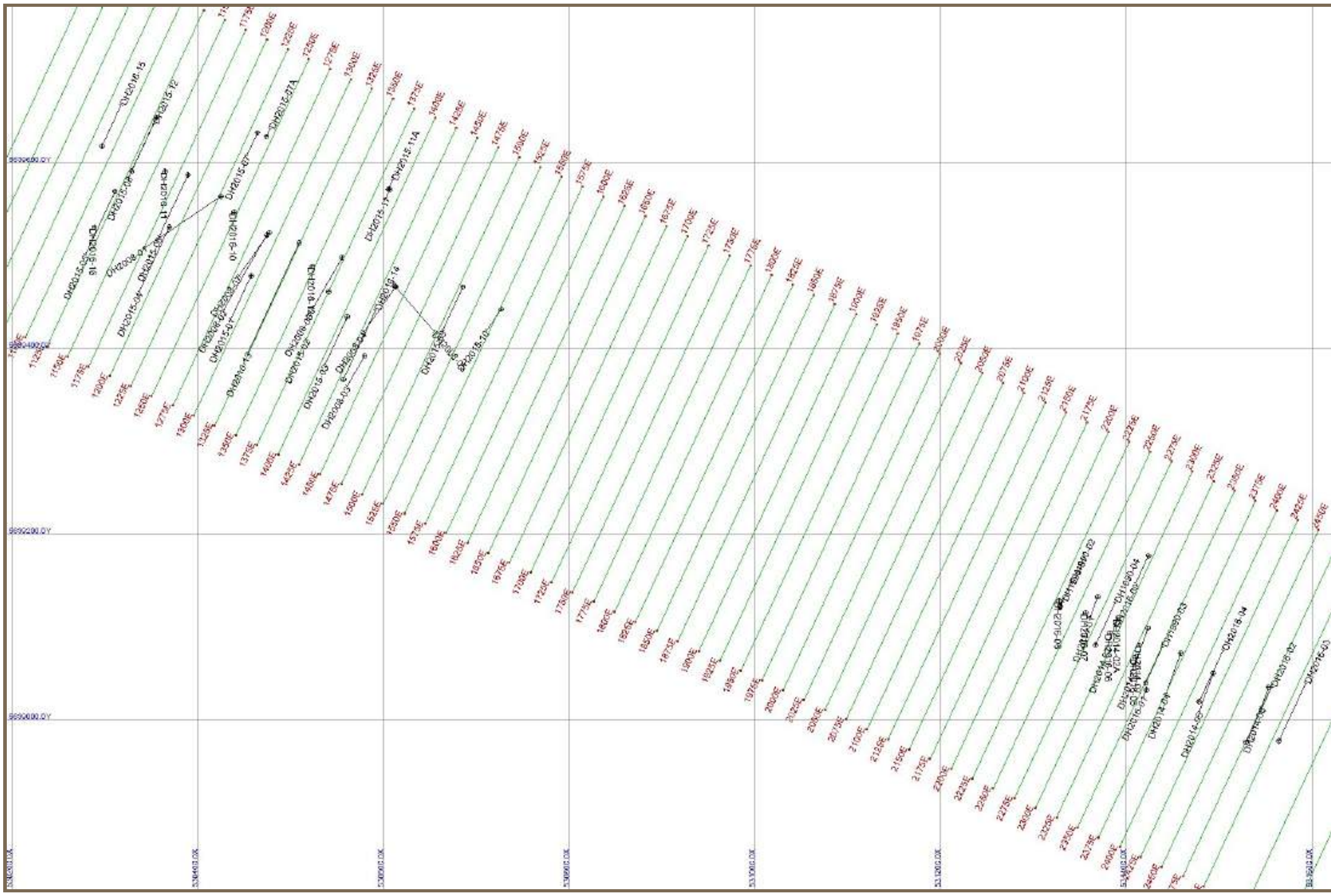


Figure 10-1: Plan View of Driftwood Creek Diamond Drill Holes

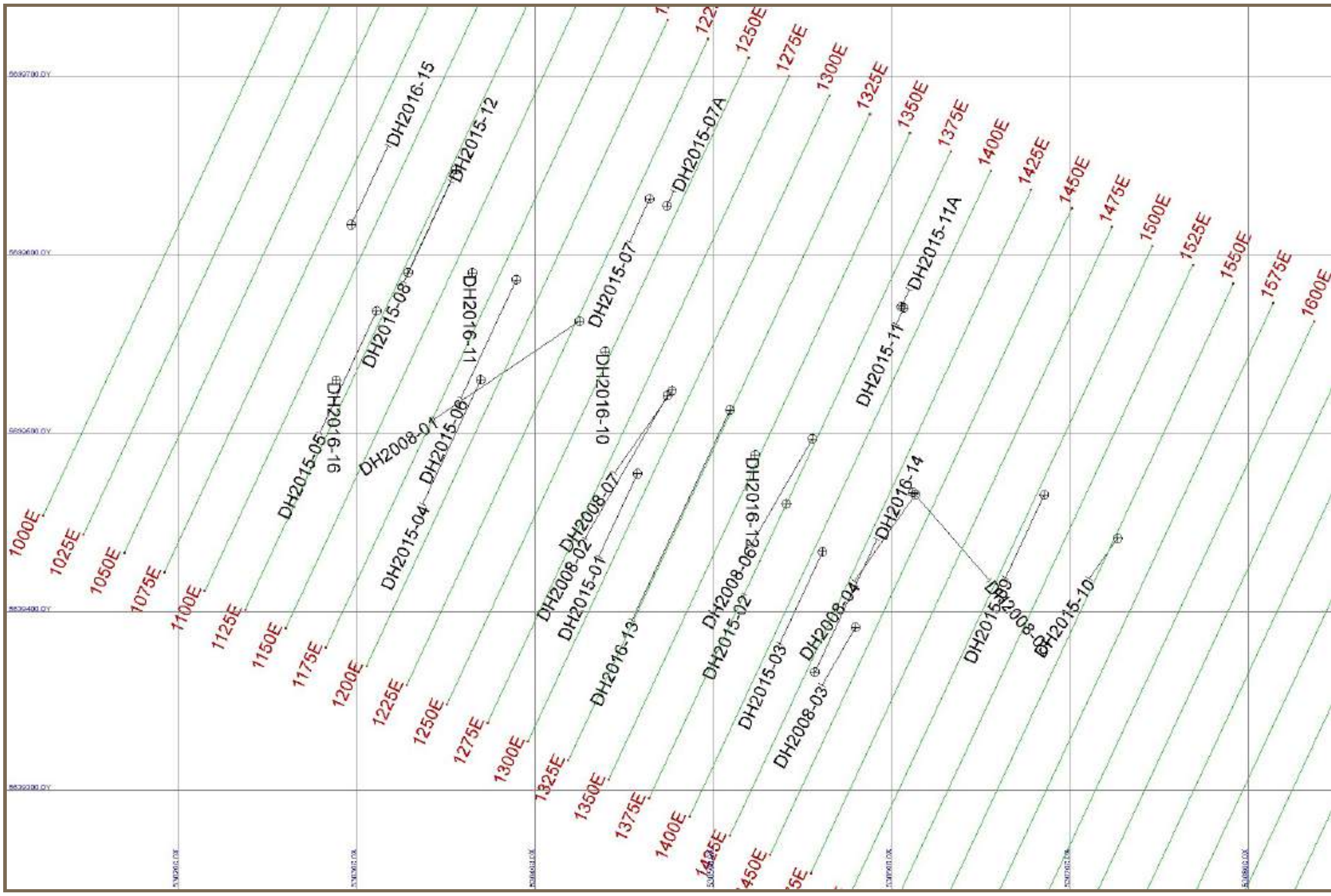


Figure 10-2: Plan View of the West Zone Diamond Drill Holes

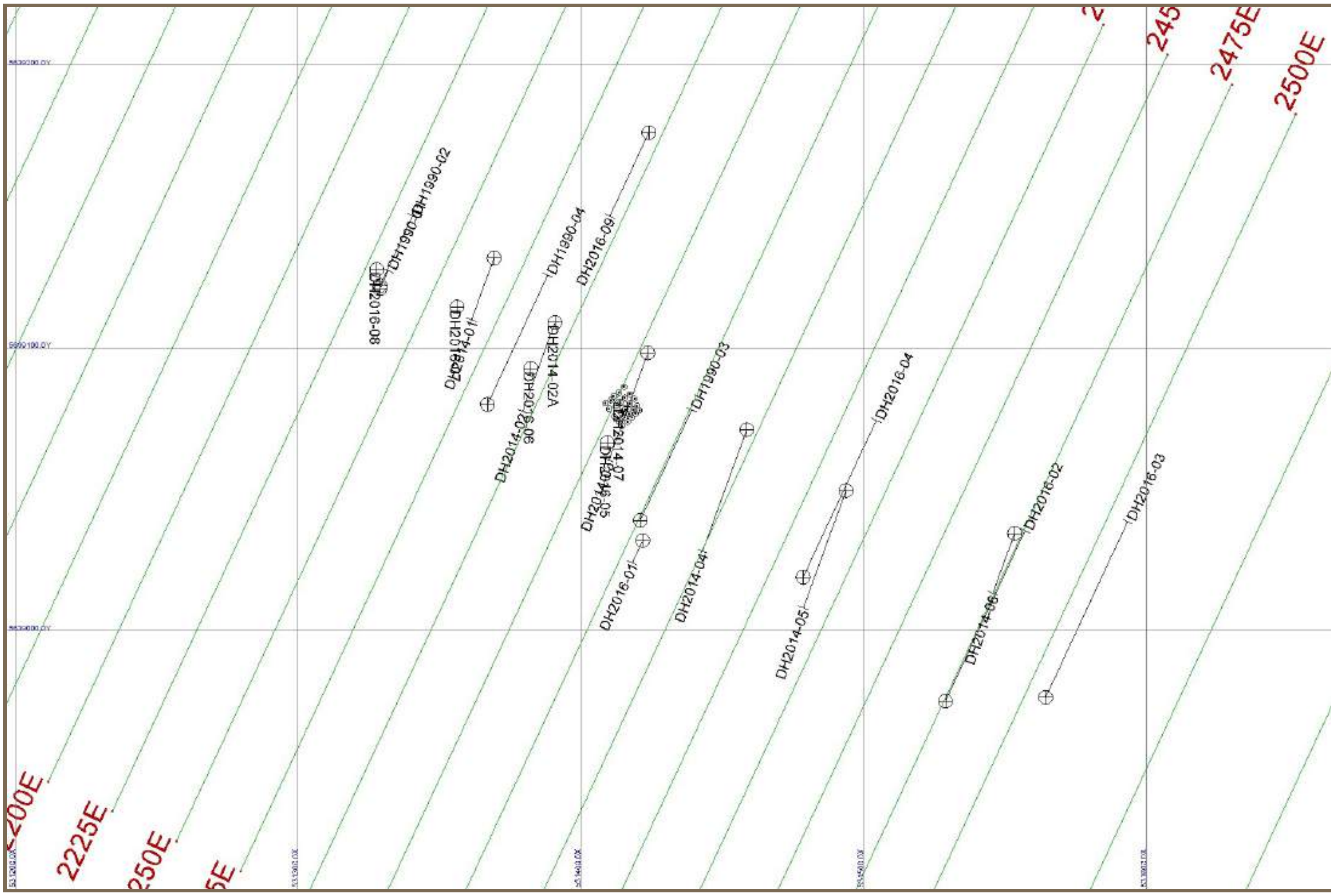
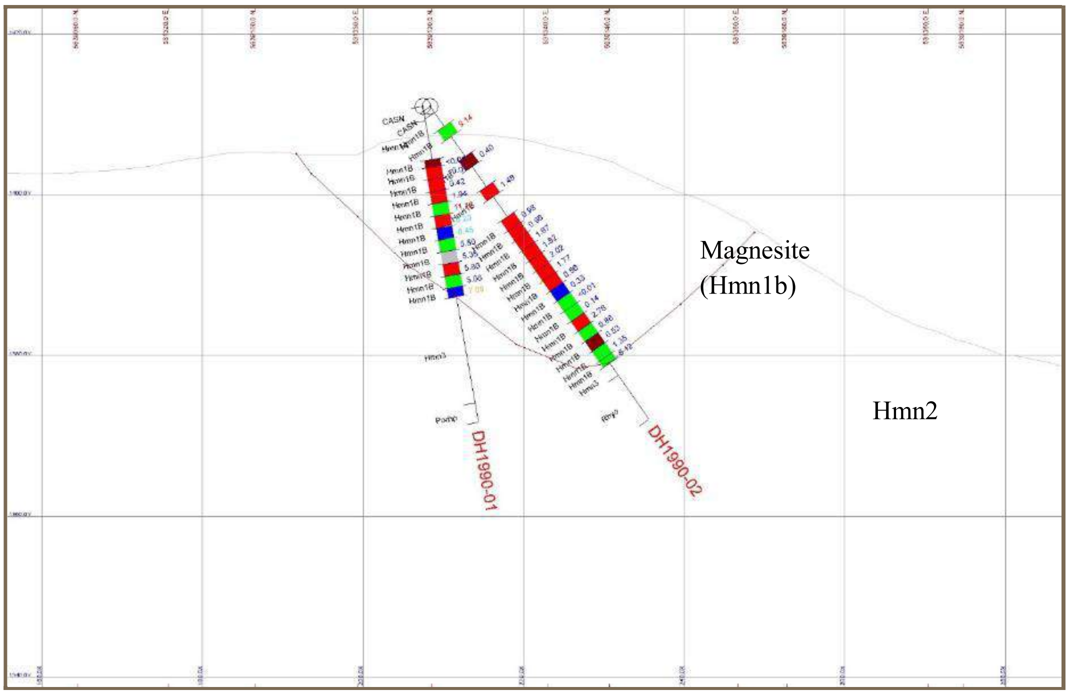


Figure 10-3: *Plan View of the East Zone Diamond Drill Holes*

10.1 1990 Canadian Occidental Petroleum Ltd. Diamond Drilling

In 1990, Canoxy completed 219.8 m of NQ diamond drilling in four holes at the western edge of the East Zone (Figure 10-3). Drill core was split on site, and samples taken at 1.5 m intervals. Only sections through the magnesite were sampled. The core samples were shipped to Chemex in North Vancouver, and were analyzed for major oxides and LOI. As well, a dead-burned assay was done for each sample. This involved analysis for MgO% after roasting at 1,000°C for one hour.

Pure magnesite (magnesium carbonate or $MgCO_3$), has a theoretical magnesia (MgO) content of 47.61%. Some of the 1990 samples were approaching this magnesia content, indicating some very high-grade magnesite occurs in the Eastern Magnesite deposit. Figure 10-4 shows the 1990 geologic interpretation for two of the holes (lithology at left, silica at right, and MgO% coloured for >40%).



Source: *Tuun Consulting Inc.*

Figure 10-4: Section 2237E (1990 DH Geologic Interpretation)

The drill results also showed that there is a higher concentration of silica and alumina along the bottom contact of the magnesite with the dolomite (Figure 10-4). The interpretation was that the best magnesite grades appear to be in the core of the syncline that forms the Eastern Magnesite deposit. The iron oxide content is generally less than 2% overall. Follow-up drilling suggests that the folding is fairly tight and slightly overturned.

10.2 2008 Tusk Exploration Diamond Drilling

Between October 25 and November 5, 2008, Tusk drilled seven NQ holes from six sites within the West Zone (Klewchuk, 2010). Holes were generally drilled southerly across the steeply north-dipping magnesite, at angles close to -45°. Hole depth ranges from 52.2 m to 141.5 m with a total of 692 m drilled.

The central part of the Western Zone is the area of thickest known magnesite on the property, and it forms a local topographic high on the magnesite ridge. According to Klewchuk (2010), about 325 m of east-west strike length of the Western Zone was partially tested by the 2008 diamond drill program, and a maximum thickness of approximately 140 m was tested.

During logging, core was marked up for sampling, but no samples were shipped for analysis. Field mapping suggests that anticlinal folding and faulting may have a bigger impact on accessible magnesite than was originally thought.

The northern margin of the magnesite deposit was not tested, and apparently, a thick band of magnesite that forms cliffs along the southern boundary of the deposit was only tested by one drill hole (DH2008-2).

The main lithology encountered by drilling was magnesite, but there are also a number of other lithologies, including dolomite, quartzite, siltstone, and a number of fine-grained intrusive (volcanic-associated?) units. Quartz veining is generally common in the magnesite, with a few narrow zones of more intense veining intersected.

Contacts between magnesite and other non-carbonate lithologies are typically quite sharp to narrowly gradational, and these contacts are typically more disturbed by late tectonic activity. These zones of broken ground and faulting at lithologic contacts proved difficult to drill through.

The magnesite intersected in drill core is generally white, pale grey, or slightly yellowish in colour. Texture is typically massive to mottled, and grain size ranges from coarsely to finely crystalline. Faint banding, which may reflect original bedding, is rarely evident. Very minor wavy to styolitic grey talc laminae are present through the magnesite in a seemingly irregular manner. White to very light grey quartz veins are scattered through the magnesite; in the fresh core, quartz veins are generally very similar in colour to magnesite, and are thus quite difficult to differentiate, except by their cross-cutting character and greater hardness. Because of this similarity in colour of magnesite and quartz in the fresh core, no attempt was made to estimate silica content during core logging.

The intrusive lenses (dikes or sills?) encountered by drilling are generally fine-grained felsic, intermediate, and mafic composition, and are probably volcanic-associated. These intrusive lenses have been described as trachytes, rhyolites, and mafic dikes in the drill logs. In places, these lenses of intrusives are more broken up than other lithologies, and they were often problematic for drilling. Drill holes DH2008-1, DH2008-6, and DH2008-7 ended in or just below bands of intrusives, which caused drilling problems.

As mentioned previously, drill core was split and sampled in 2008, but the samples were not submitted for chemical analyses. This was presumably due to a lack of money to pay for the analyses. These samples were subsequently stored by the property vendors until they could be submitted to AGAT Laboratories Ltd. (AGAT) for assay. In 2012, Torch River Resources agreed to pay for analyzing the samples as part of their due diligence in evaluating the property. Torch River decided not to option the property.

10.3 2014–2015 MGX Resources Diamond Drilling

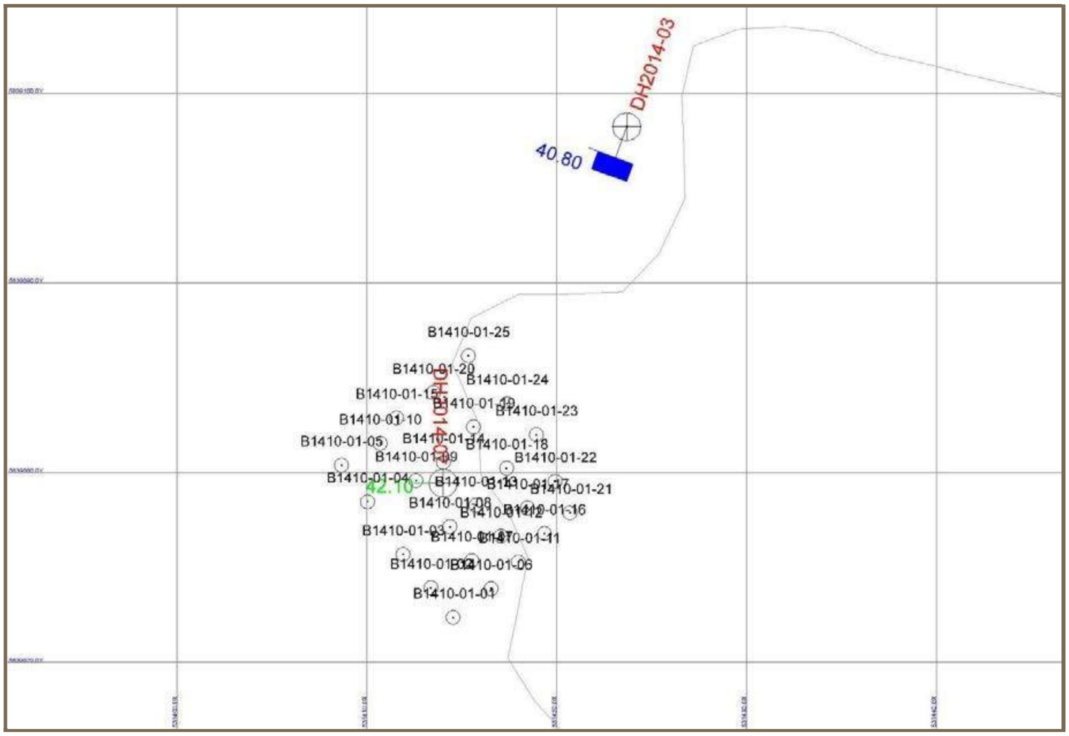
A total of 437.52 m (1,435.07 ft) from eight holes were drilled in a 100 m x 300 m area on the East Zone in September 2014. The intent was to test the results of the 1990 Canoxy drilling.

MGX followed up in 2015 with fourteen BQTW drill holes located on the west portion of the mineral property along the ridge top. A total of 1,093.38 m of core was recovered.

10.4 2016 MGX Resources Percussion Blast Hole Drilling

In May 2016, a site was selected in the East Zone for a bulk sample. Twenty-five 6" (152 mm nominal) diameter percussion drill holes were used as blast holes. The holes were sampled, and the assays compared to grab samples from the magnesite bulk sample stockpile. The holes were relabelled to reflect the quarrying bench elevation and numeric order of the blast. For example, B1410-01-13 would be hole number 13 of the first blast on the 1,410 m elevation bench.

Figure 10-5 shows the location of the percussion holes, and their proximity to the nearby diamond drill holes. DH2014-07 is in the middle of the bulk sample blast pattern.



Source: *Tuun Consulting Inc.*

Figure 10-5: Plan View of 2016 Percussion Drill Blast Holes

10.5 Diamond Drilling Assays

Project drill assaying laboratories have varied, from Chemex (1990), to AGAT (2012), and currently ALS (aka ALS Global). Looking at the average 36.67% MgO value for the 2012 AGAT assays of the 2008 data flagged a concern to be checked. The outcrop rock geochemistry indicated grades between 41.99% and 43.80% MgO. The 1990 Chemex assaying averaged 40.13% MgO.

MGX decided to re-assay 245 of the 2008 drill core samples (Hmn1B) to determine if the 2012 assays were biased to the low side or truly indicative of the area drill-tested. Figure 10-6 shows that the AGAT work did appear to have a low bias for MgO%. The other minerals seemed reasonable, as can be seen for SiO_2% in Figure 10-7.

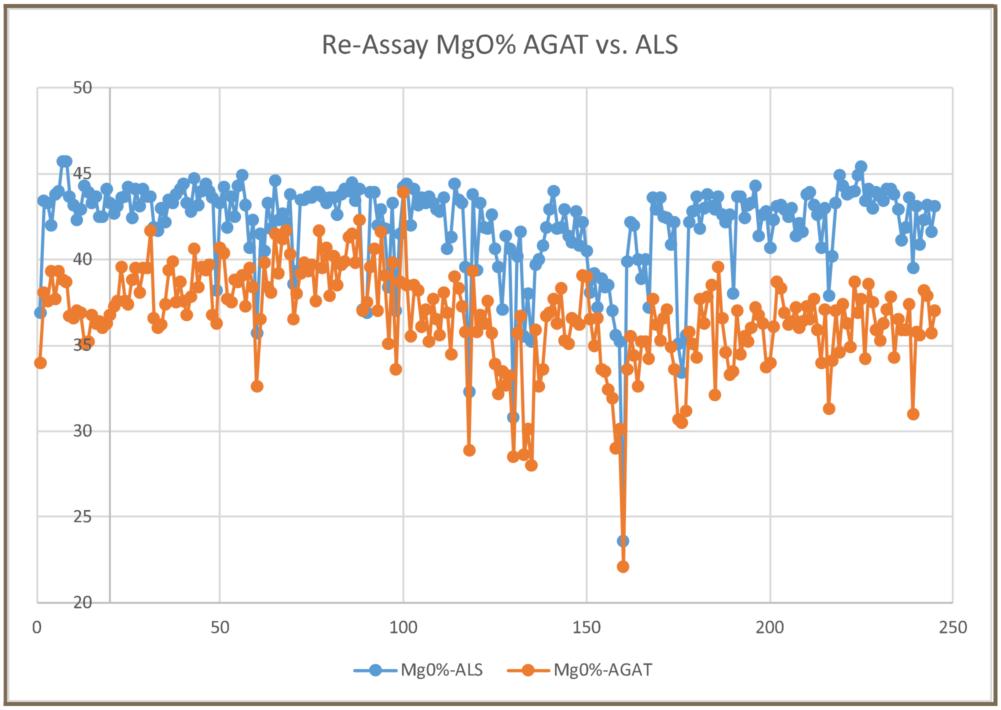


Figure 10-6: ALS vs. AGAT MgO% Re-Assay

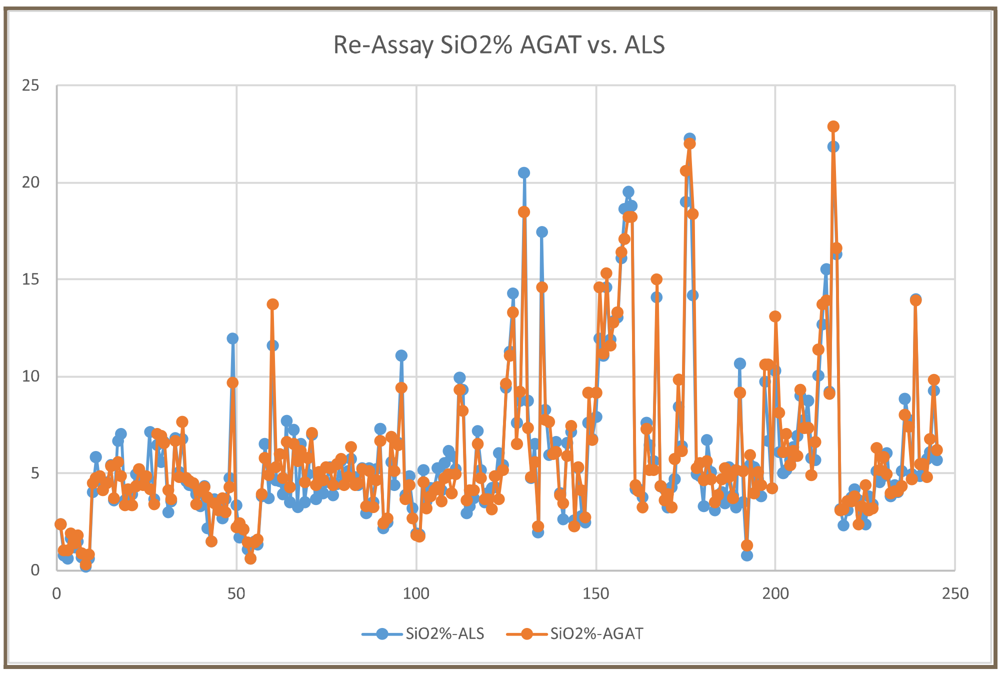


Figure 10-7: ALS vs. AGAT SiO_2% Re-Assay

MGX has since standardized its choice of ALS for all follow-up assaying. ALS states that the laboratorymeets International Standards ISO/IEC 17025:2005 and ISO 9001:2015. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2005 for specific analytical procedures.

After the completion of the maiden resource estimate, MGX drilled a total of 16 of the recommended infill drill holes.

Table 10-2 summarizes the assay results by year for all rock types intersected, while Table10-3 summarizes magnesite intercepts by hole.

Table 10-2: Diamond Drill Assays by Year (all lithologies)

Year	Laboratory	MgO %	Al_2O_3 %	SiO_2 %	CaO %	Fe_2O_3 %	LOI %
1990	Chemex	40.13	1.77	10.23	1.18	0.90	44.81
2008	ALS	37.48	2.17	11.11	3.06	2.07	42.97
2014	ALS	40.13	1.63	9.07	1.84	1.01	45.41
2015	ALS	37.41	1.94	12.73	3.44	1.58	42.98
2016	ALS	39.52	1.54	9.37	3.00	1.34	44.69

Notes: *% = percent; LOI = loss on ignition*

Table10-3: Significant Magnesite Intercepts

HOLE-ID	From (m)	To (m)	MgO %	Al_2O_3 %	SiO_2 %	CaO %	Fe_2O_3 %	LOI %	Interval
DH1990-01	6.71	24.08	43.10	1.15	5.53	1.85	1.08	46.95	17.37
DH1990-02	3.05	39.01	44.40	0.53	1.88	1.62	0.78	49.46	35.96
DH1990-03	1.22	60.96	36.55	2.89	16.37	0.87	0.90	41.05	59.74
DH1990-04	7.62	60.66	40.19	1.57	9.85	0.99	0.88	44.92	53.04
DH2008-01	2.00	141.50	35.66	2.46	10.92	5.37	2.19	42.49	139.50
DH2008-02	2.00	132.00	42.46	0.96	4.82	1.39	1.49	48.30	130.00
DH2008-03	1.00	52.20	26.98	2.51	14.30	12.31	1.95	39.54	51.20
DH2008-04	2.00	82.70	34.16	1.76	18.40	2.92	1.94	39.54	80.70
DH2008-05	2.90	99.40	39.91	1.36	8.92	1.39	1.68	45.65	96.50
DH2008-06	0.70	100.00	36.23	4.36	16.80	0.67	3.17	36.94	99.30
DH2008-07	3.50	82.70	40.17	2.43	9.34	1.14	2.26	43.65	79.20
DH2014-01	1.00	37.80	39.86	2.22	9.79	1.00	0.92	45.06	36.80
DH2014-02	2.00	54.25	42.76	1.00	5.93	1.08	0.75	47.91	52.25
DH2014-02A	0.35	39.00	40.57	1.69	9.22	1.46	1.05	45.08	38.65
DH2014-03	2.80	65.53	38.22	3.00	12.52	1.13	1.56	42.12	62.73
DH2014-04	0.80	69.00	39.91	1.33	7.54	2.78	0.95	46.36	68.20

HOLE-ID	From (m)	To (m)	MgO %	Al_2O_3 %	SiO_2 %	CaO %	Fe_2O_3 %	LOI %	Interval
DH2014-05	2.80	71.63	38.78	1.27	11.17	2.42	0.88	44.79	68.83
DH2014-06	3.00	36.58	38.16	1.85	11.41	3.82	0.96	43.32	33.58
DH2014-07	0.70	57.91	42.72	0.94	5.60	1.18	0.93	48.11	57.21
DH2015-01	2.10	121.92	41.10	1.15	6.62	2.75	1.58	46.27	119.82
DH2015-02	2.74	91.44	40.99	1.88	11.27	0.73	1.43	43.19	88.70
DH2015-03	0.61	65.53	40.71	2.37	13.16	0.92	1.49	40.97	64.92
DH2015-04	3.00	128.02	38.79	1.32	9.16	4.28	1.48	44.62	125.02
DH2015-05	3.00	125.88	27.93	3.79	21.56	7.84	2.24	34.21	122.88
DH2015-06	1.00	114.30	39.97	1.51	10.88	2.61	1.36	43.22	113.30
DH2015-07	0.80	18.00	40.58	0.25	11.95	0.52	1.28	45.28	17.20
DH2015-07A	3.00	18.29	35.86	0.81	21.40	0.98	0.80	40.02	15.29
DH2015-08	1.80	90.00	38.31	2.14	13.64	1.48	1.57	41.91	88.20
DH2015-09	6.00	79.86	42.42	0.83	6.85	0.88	1.48	47.34	73.86
DH2015-10	9.00	43.28	41.25	0.62	9.78	0.64	1.48	45.94	34.28
DH2015-11	3.00	12.00	40.73	0.38	11.86	0.79	0.67	45.46	9.00
DH2015-11A	1.50	15.00	39.28	0.48	14.96	0.72	0.71	43.65	13.50
DH2015-12	3.50	112.78	40.96	1.73	9.40	1.28	1.26	44.78	109.28
B1410-01-01	0.00	9.14	41.30	0.80	8.42	0.75	0.87	46.76	9.14
B1410-01-02	0.00	12.19	42.00	0.73	8.16	0.79	0.87	46.99	12.19
B1410-01-03	0.00	12.19	41.30	1.14	9.80	0.67	0.72	45.99	12.19
B1410-01-04	0.00	12.19	42.40	0.92	5.40	2.15	0.89	48.19	12.19
B1410-01-05	0.00	9.14	41.90	1.03	8.16	1.73	1.05	46.01	9.14
B1410-01-06	0.00	12.19	42.10	0.81	8.37	0.85	0.73	46.89	12.19
B1410-01-07	0.00	9.14	42.60	1.11	7.66	0.74	0.70	47.05	9.14
B1410-01-08	0.00	12.19	42.70	0.92	7.22	0.80	0.76	47.49	12.19
B1410-01-09	0.00	9.14	41.00	0.76	6.42	2.83	0.97	47.62	9.14
B1410-01-10	0.00	12.19	43.20	0.70	6.32	0.94	0.97	47.93	12.19
B1410-01-11	0.00	9.14	43.50	0.57	5.85	0.68	0.90	48.43	9.14
B1410-01-12	0.00	12.19	43.30	0.98	5.48	1.00	0.79	48.36	12.19
B1410-01-13	0.00	9.14	42.30	0.98	6.44	0.83	0.81	47.76	9.14
B1410-01-14	0.00	12.19	38.30	0.97	13.39	2.27	0.97	44.02	12.19
B1410-01-15	0.00	9.14	42.20	1.41	7.21	1.06	1.23	46.58	9.14
B1410-01-16	0.00	12.19	43.10	0.70	6.24	0.79	0.78	48.03	12.19
B1410-01-17	0.00	9.14	42.40	1.05	7.91	0.77	0.77	47.02	9.14
B1410-01-18	0.00	12.19	40.40	1.76	10.65	0.86	0.89	45.1	12.19

HOLE-ID	From (m)	To (m)	MgO %	Al_2O_3 %	SiO_2 %	CaO %	Fe_2O_3 %	LOI %	Interval
B1410-01-19	0.00	9.14	40.80	1.64	9.57	0.74	1.04	45.52	9.14
B1410-01-20	0.00	12.19	41.10	1.60	9.26	0.78	0.97	45.91	12.19
B1410-01-21	0.00	12.19	44.40	0.16	4.43	0.76	0.95	49.28	12.19
B1410-01-22	0.00	12.19	40.20	1.72	10.16	0.96	1.11	45.17	12.19
B1410-01-23	0.00	9.14	42.10	0.84	5.38	1.66	1.29	48.11	9.14
B1410-01-24	0.00	12.19	41.10	1.73	8.27	1.32	1.18	46.09	12.19
B1410-01-25	0.00	9.14	40.70	1.62	9.29	0.87	1.15	45.87	9.14
DH2016-01	3.00	33.00	38.06	1.34	8.34	4.22	1.40	45.55	30.00
DH2016-02	9.00	94.00	41.56	1.44	7.91	1.48	1.04	46.20	85.00
DH2016-03	1.00	27.00	43.22	1.12	4.78	2.00	1.12	47.28	26.00
and	54.00	87.00	43.66	1.15	4.38	1.27	0.87	48.25	33.00
DH2016-04	2.50	51.00	40.98	1.27	7.62	2.42	1.00	46.22	48.50
and	60.00	78.00	41.97	1.54	8.40	0.77	1.01	46.13	18.00
DH2016-05	0.10	46.00	41.26	1.24	7.87	1.22	1.11	46.55	45.90
DH2016-06	0.10	45.00	43.07	1.35	5.58	0.89	0.89	47.92	44.90
DH2016-07	1.50	48.00	43.81	0.91	3.74	1.02	0.82	49.31	46.50
DH2016-08	0.50	30.00	42.45	1.23	5.19	1.42	0.86	48.32	29.50
DH2016-10	0.10	57.00	43.53	1.22	4.79	0.62	1.43	48.07	56.90
DH2016-11	3.00	60.00	42.52	0.83	6.32	1.65	1.36	47.04	57.00
DH2016-12	0.10	18.00	43.68	0.68	5.08	0.52	1.48	48.29	17.90
and	27.00	75.00	42.43	1.46	6.97	1.17	1.38	46.33	48.00
DH2016-13	1.00	162.00	41.98	1.67	10.52	0.76	1.38	43.21	161.00
DH2016-14	1.00	123.00	39.96	1.50	8.78	3.14	1.44	44.54	122.00
DH2016-15	6.00	42.00	37.43	0.83	17.50	1.21	1.31	41.55	36.00
DH2016-16	0.50	80.00	42.40	0.96	3.82	2.12	1.55	48.57	79.50

Notes: *m = metres; LOI = loss on ignition; % = percent*

10.6 Qualified Persons' Observations

The apparent thickness of the antiform magnesite bed is about 30 m. The assay and logging information shows that the bed has a quite consistent grade, with minor silica flooding along some contacts. At this time, the source of the silica is unclear.

11 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

11.1 1990 Canadian Occidental Drill Program

Canoxy (Rodgers 1990, 1990a) used Chemex of North Vancouver to provide analytical services for their 1989 and 1990 exploration programs. The 1990 drilling program involved splitting of NQ core, bagging half of the core in 1.5 m intervals, and shipping the samples to the Chemex laboratory. There are no notes regarding special sample security procedures used for drill core sampling programs, but the author is confident that applicable industry best practices were followed by the property operators.

Rodgers (1990) reports that the samples were crushed and pulverized at the laboratory to -80 mesh, then a representative split was taken and this was pulverized to -150 mesh. The +150 sample material was saved. Samples were digested using a perchloric-nitric-hydrofluoric acid mixture. The samples were analyzed for SiO_2, Fe_2O_3, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, MnO, BaO, and LOI using the Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES) technique. Detection limits for these major oxides was 0.01%, with the upper limit of detection at around 99%. MgO was also analyzed using Atomic Absorption Spectrometry (AAS), with an upper detection limit of 100%. As well, a dead-burn analysis was done for each sample, which involved analysis for MgO% after roasting at 1,000°C for one hour.

Copies of the original assay certificates issued by Chemex were included in Assessment Report 19416 (Rodgers, 1990), and examination of these certificates indicates that Chemex did sufficient QA/QC procedures (including the analysis of blanks and standards) to ensure the accuracy and precision of the analytical results. The author hand-checked every assay to ensure that the provided database was accurate, but did not end up using the data because of survey location concerns.

11.2 2008 Tusk Exploration Drill Program

Drill core samples from the 2008 drilling program were collected in a similar manner to the 1990 samples. However, the samples were not immediately sent to the lab, and remained in bags for four years. There is no information on how the samples were stored, or if any special security precautions were taken. It is assumed that the samples were stored at the Vine Creek storage facility with the remaining drill core until Torch River Resources offered to pay for shipping and analytical work. The authors have visited the Vine Creek facility, and it seems unlikely that the samples would have been tampered with at the facility, although they may have weathered slightly. The 2008 drill core samples were shipped to AGAT (accredited to the ISO 9001 standard) in Burnaby, BC, on April 10, 2012.

The analytical results were reported on Certificate of Analysis 12V589981 on April 27, 2012. The analytical method involved a lithium borate fusion and analysis by ICP-OES. Results were reported for SiO_2, Fe_2O_3, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, MnO, BaO, and LOI. AGAT did perform QA/QC procedures, including the analysis of blanks and standards to ensure the accuracy and precision of the analytical results.

MGX opted to re-assay the remainder of the drill core at the ALS laboratory in Kamloops. The ALS laboratories have attained ISO 9001:2008 certification. Final assays (file KL1504783) were similar to previous drill programs, and were used for resource estimation.

11.3 MGX 2014–2016 Drill Programs

For the 2014–2015 BTW drilling, Neill's Mining Ltd. of Burns Lake provided a Longyear 28 drill, and Woodside Excavating Ltd. of Langley provided a Bobcat for access trails, drill moves, and reclamation.

The diamond drill core was photographed and logged by A. Kikauka, P.Geo. A screw-feed, blade-equipped core splitter was used to split the core in half. Each piece of core was split, with half of the core placed in marked poly bags at 3 m intervals and shipped to ALS in Kamloops or North Vancouver, and the other half placed, in a duplicate orientation position, back into the core box for storage at the Vine Creek facility.

Blank samples (nearly pure silica) were inserted in the diamond drill core sample stream every 20 samples in order to verify data from the laboratory. The seven high-grade standard samples consisted of 0.84 kg to 1.4 kg sized rock chips from a nearly pure boulder of magnesite, and were inserted for QA/QC protocol.

ALS crushed, split, and pulverized samples using prep-31 code. This involves crushing to better than 70% passing a 2 mm screen, then a split of 250 g is pulverized to better than 85% passing a 75 μm screen. The sample pulp is analyzed using ME-XRF-06 (XRF-26) lithium borate flux major oxide whole-rock geochemical analytical methods (Appendix A).

11.4 MGX June 2016 Bulk Sample

In June 2016, a bulk sample was collected at the East Magnesite location by workers from Dominion Excavating of Invermere, BC (red-flagged location in Figure 11-1). It was recognized that there would be contamination from topsoil (unavoidable for the first bench). To rectify the issue of contamination, the overburden and coarse blast rock were relocated outside the edges of the blast by a Cat 336D excavator.

Once finer material was exposed, the material was loaded and hauled by Cat D300E articulated truck to a stockpile ≈3 km away in the valley bottom (Figure 11-2). Note that the steep magnesite cliff faces can be easily seen in Figure 11-2.

Source: *Tuun Consulting Inc.*

Figure 11-1: East Zone Quarrying

At the stockpile location, sixteen 25-L pails of whole rock magnesite grab samples were collected on a rectangular grid pattern for assaying (and SG determination) as previously described. The pails were sealed and shipped to ALS Kamloops.

Approximately 100 tonnes of material were then reloaded into highway dump trucks for transport to the Dominion Excavating gravel pit south of Radium. There the fines were to be screened out and used as a sub base, with cleaner uncrushed material to be used as a base for magnesite that will be crushed and screened to specifications.

Source: *Google Earth*

Figure 11-2: Magnesite zones and Access to Stockpile

As a pre-testwork check of magnesite bulk sample grade, MGX assayed the percussion drill holes (PDH) that were used as blast holes, along with taking sixteen 25-L pails of rock samples in a grid from over the stockpile.

Kikauka also logged the percussion chips from the 2016 bulk sample drilling. Samples were bagged, and like the diamond drill samples, submitted to ALS in North Vancouver for assaying.

Both the PDH and grab sample pails were shipped to ALS in Kamloops for assay by the same methodology as described for the diamond drill programs.

11.5 Sample Security – Vine Creek Core Storage Facility

The Vine Creek Core Storage Facility near Cranbrook, BC (lat. 49.39938 N; long.-115.818112 W; see Figure 4-1) was visited by Tuun. The owner is David L. Pighen, P.Geo., and the facility is both well managed and secure on his property, which is off Highway 3 to the south.

Storage of core is on custom-built wooden racks with roofs (Figure 11-3). Core logging and sampling facilities (rock saws and core splitters) are also available.

Figure 11-3: Vine Creek Core Storage Facility

11.6 Blanks, Duplicates, and Standards

There are no commercially available standard certified reference materials (CRMs) for magnesite or magnesium oxide resources. Therefore, there historically was a reliance upon the various laboratories' own internal standards and procedural checks to verify results. Given the high quality of the ISO-certified laboratories, this was a reasonable approach.

In 2014 MGX chose to use a non-certified "standard" that was sourced from a high-grade magnesite boulder obtained from a roadcut some 250 m west of the westernmost drill hole in the West Zone (the roadcut is located next to the Fish Zone). A total of seven of these standards were inserted every 20th sample (Figure 11-4). There were no blanks inserted in 2014.

These control samples diverge from industry best practices, but given the lack of available CRMs, did provide useful information. Note that the 2014 high-grade samples all fell within two standard deviations (SD) of the mean.

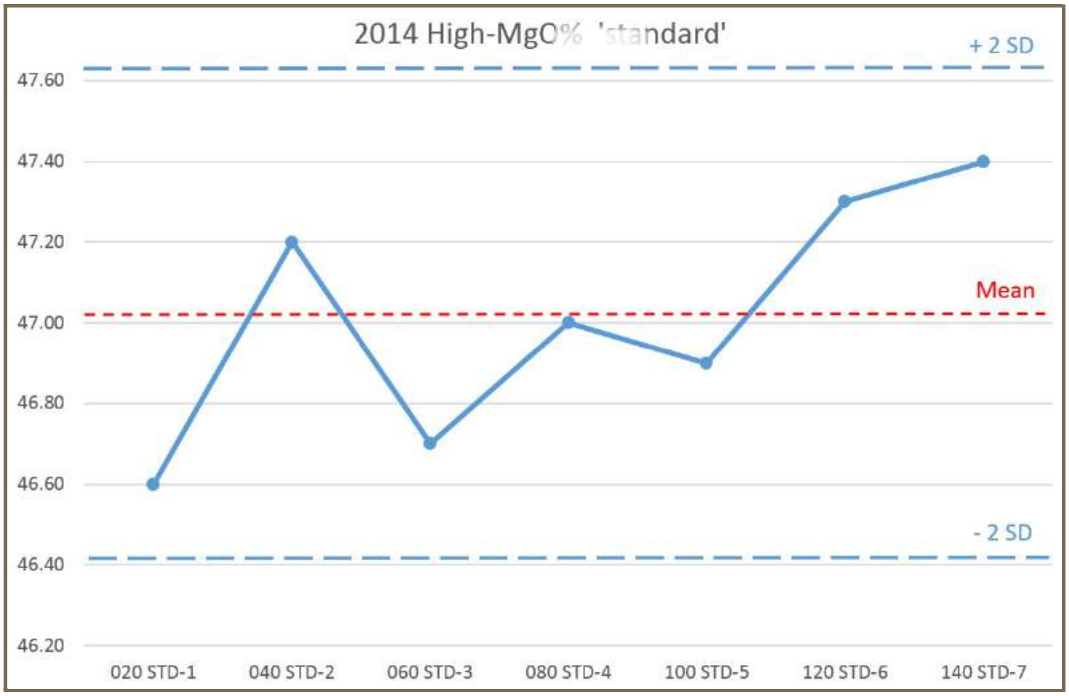


Figure 11-4: 2014 High-MgO% "Standard"

The 2015 "standards" and "blanks" were obtained from a roadcut near Driftwood Creek approximately 1,000 m south of the West Zone, and high silica samples (i.e., true blanks of near zero-grade magnesium) were sourced from Wonah Formation quartzite located 60 km SE of Canal Flats.

In 2015, the nine low-grade "standards" inserted (Figure 11-5) were from a low-magnesium dolomite, and again the laboratory results were essentially within two SD of the mean. (The one outlier in 2015 was only 0.02 above two SD, which difference is considered immaterial).

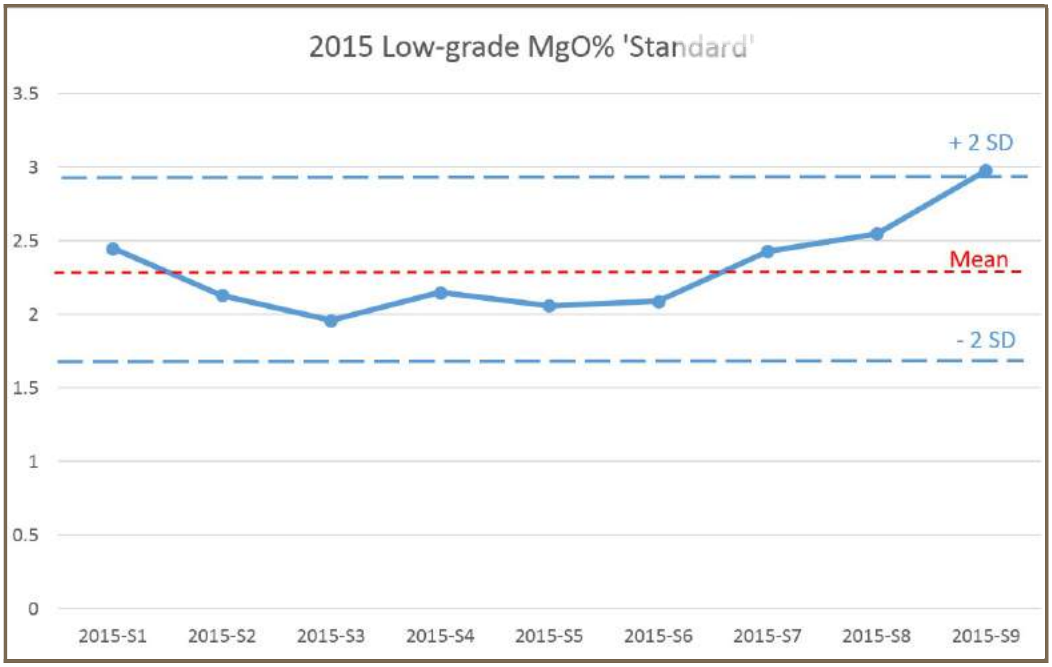


Figure 11-5: 2015 Low-MgO% 'Standard'

Figure 11-6 shows the blanks obtained from the Wonah Quartzite, which contained less than 1% MgO. In this case, the majority of the samples were within one SD of the mean, and all were within two SD.

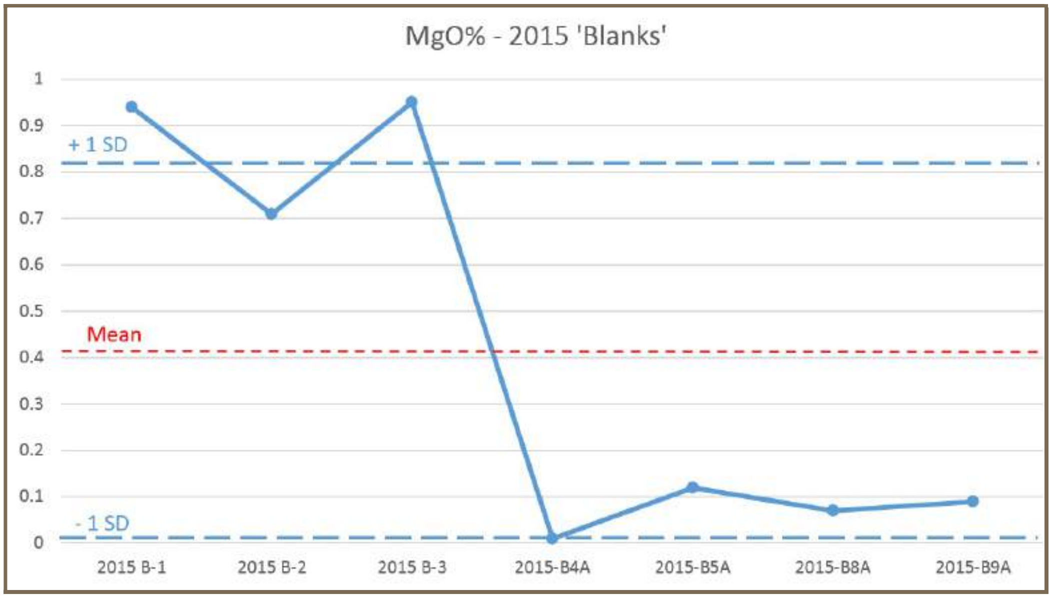


Figure 11-6: 2015 Silica "Blanks"

Overall, the 2015 non-certified control samples (low-grade MgO "standards," and "blanks" (high grade SiO_2) performed very well; however, there is no guarantee that this excellent performance will continue, and a laboratory error could potentially be missed. The key limitation of the controls used is that they have not undergone the rigorous round-robin analytical testing of a CRM to confirm grade and repeatability of the expected grade.

11.7 QP Observations

Tuun has noted that during the various campaigns, one of the Owners (either Kikauka, Klewchuk, or Rodgers) was present for the collection and security of core samples for shipment to the assay laboratory. The drill core was split into halves, with one-half staying with the drill box, and the other half put into poly bags matching the assay interval and tag.

Given there are no known MgO standards available, Kikauka utilized samples of unrelated barren rock for the blanks, as well as samples of another magnesite deposit as standards. Tuun has recommended that MGX work toward creating a set of certified reference standards with a respected laboratory.

The consistency of the historical assay results between the various laboratories used gives a level of confidence that the magnesite bed warrants further investigation.

12 DATA VERIFICATION

12.1 Sample Coordinates

In 2015, some of the drill collars were surveyed by Focus of Cranbrook. Focus became part of the WSP Group and followed up in 2016, surveying all but the 1990 drill collars. WSP uses a DGPS, with corrections for diurnal variations using a GPS base station.

12.2 Downhole Surveys

There is no evidence that any downhole surveys were taken in any of the drill programs. The lack of downhole survey information means that any potential hole deflection would be missed, thus impacting the accurate locating of geologic contacts.

Tuun recommends that downhole surveys be instituted in all future diamond drill campaigns.

12.3 Assay Data

The assay database consisted of several Excel worksheets cross-referenced to the drill holes by year. Tuun collated all assays into one master spreadsheet, and checked the data against the assay sheets. No errors were found.

From the master spreadsheet, a subset of the data was imported into GEMS™. The GEMS import module can be set to reject samples that fall outside of limits. No errors were detected during the import process.

12.4 Opinion of the Qualified Persons

MGX (and the Owners) have complied by performing BC Assessment Report work on the Project to maintain the status of the mineral tenures. Work was done to exploration industry standards by competent registered professionals. It is the author's opinion that the data collected and tracked is of adequate quality for the purposes of this PEA.

13 MINERAL PROCESSING AND METALLURGICAL TESTWORK

To date, there has been only one testwork program conducted on the deposit. In the 2008 BC Assessment Report #30243, it was reported that SGS Lakefield conducted preliminary beneficiation testing of two composite samples in 2007/2008. The SGS program and results are summarized in Section 13.1.

A simplified flowsheet schematic, illustrated in Figure 17-1, was used to represent the main process steps for this report. The flowsheet is partially based on the development work completed by SGS and a proposal from Industrial Furnace Company Inc. for the calcination and sintering operations. The process operations can be divided into a three-stage approach, where the mineralized material would initially be sized and screened, upgraded and calcined into CCM, and then sintered into DBM. It should be noted that limited metallurgical testwork has been completed to date, and the flowsheet prepared for this report is highly conceptual.

13.1 SGS Lakefield Research (SGS)

In 2007, four outcrop samples, each of ≈50 kg, were taken from the West and East Zones, as shown in Figure 13-1. The purpose was to conduct preliminary beneficiation tests, and the samples were bagged and shipped to SGS Lakefield.

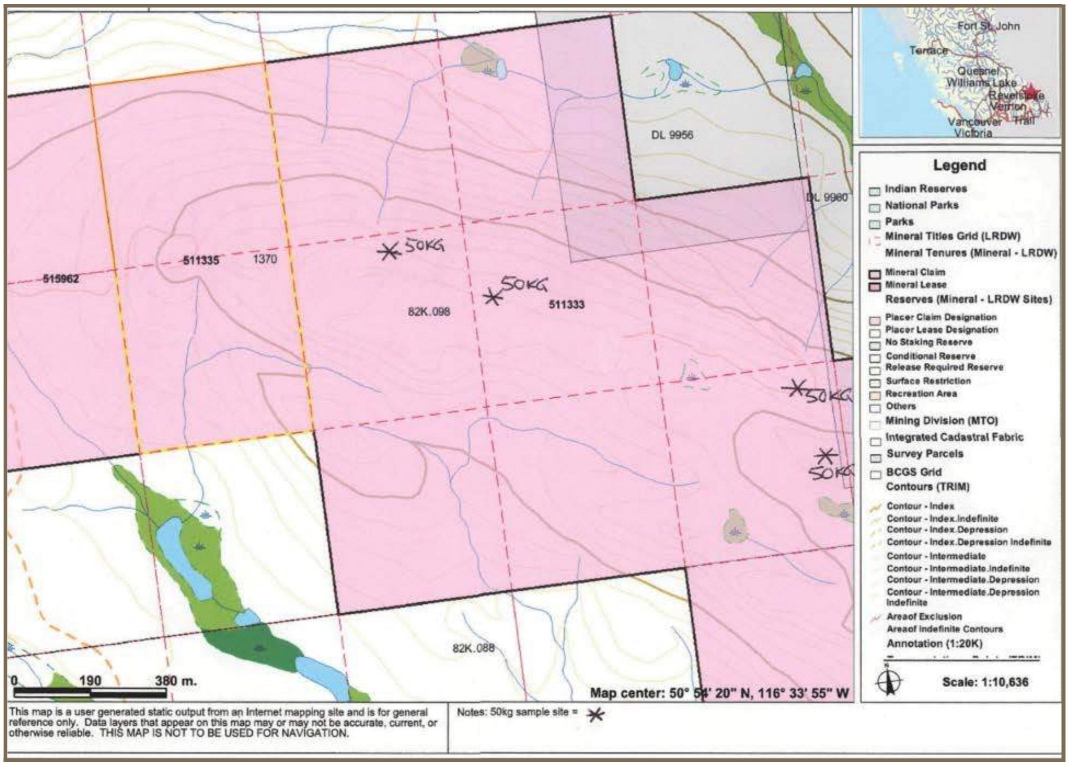


Figure 13-1: Magnesite Metallurgical Test Sample Locations

13.1.1 Beneficiation Objectives

The first phase of beneficiation on two composite magnesite samples (West and East) from the Driftwood property, located in Brisco, BC, has been completed. The objective of this phase was to develop a process to recover magnesite from the sample material. During this scoping study, a preliminary flotation flowsheet and reagent scheme was developed. This flowsheet consisted of pyrite and silicate flotation circuits. Magnesite concentrate will be recovered as silicate flotation tailings. The concentrate assays from this flowsheet are shown in Table 13-1.

Table 13-1: Magnesite Reverse Flotation

Test No.	Product Name	Weight (%)	Assay %					Distribution %				
			SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO
West Zone Comp.	Magnesite Conc.	85.6	0.52	0.07	1.07	1.60	44.5	7.53	30.03	85.7	84.5	90.9
	Head Assay	-	5.91	0.17	1.06	1.68	42.6	-	-	-	-	-
East Zone Comp.	Magnesite Conc.	89.8	0.56	0.10	0.91	0.91	46.4	19.8	22.6	87.4	90.5	92.4
	Head Assay	-	2.59	0.38	0.82	0.89	43.6	-	-	-	-	-

Source: *SGS (2008)*

Notes: *% = percent; Comp. = composite; Conc. = concentrate*

The magnesite recoveries from the West and East Zone composites using reverse flotation were 91% and 92%, respectively.

From the results obtained, Aghamiriam and Imeson (2008) concluded that:

- This material has a high magnesite grade, estimated at 93.4% for the East Zone and 86.3% for the West Zone. It responded well to beneficiation by silicate flotation, with magnesite concentrate generated as silicate tailings.
- All the efforts to reduce the iron content of the magnesite concentrate were unsuccessful. It is believed that this is due to the presence of iron in magnesite crystal structure as solid solution.
- Heavy media separation (HMS) can be considered as a potentially suitable process for primary upgrading, to reject a large portion of silicate minerals, at approximately 73% to 80%, and calcite, at nearly 40% in a coarse fraction.
- Grinding and screening to different fractions failed to generate an acceptable magnesite concentrate.
- High intensity dry and wet magnetic separations to separate iron-containing minerals were attempted. These methods failed to perform a reasonable task to reduce the iron content of the magnesite concentrate.

It should be noted that both the flowsheet and reagent scheme developed in this investigation are preliminary in nature, and more detailed testwork should be conducted to optimize the flotation process.

13.1.2 Sample Receipt and Preparation

The samples were received in November 2007 in several rice bags from the East and West Zones of the deposit, as shown in Figure 13-2. Samples from each zone were removed from the bags and crushed to -10 mesh. Samples from each individual zone were then blended, homogenized, riffled, and rotary split into 2 kg charges. A 250 g subsample was riffled out of each composite from a 2 kg charge and submitted for chemical analysis.

Figure 13-2: Samples as Received in Rice Bags

13.1.3 Head Sample Characterization

13.1.4 Chemical Analysis

Head samples from the East and West Zones were submitted for whole rock analysis (WRA). The results are presented in Table 13-2. Based on these results, mineral estimates are given in Table 13-3. These estimations are based on the assumption that all the measured calcium oxide is either in calcite or in dolomite. In reality, the presence of both of these minerals should be expected. This kind of analysis, however, can provide upper and lower limits of magnesite assays in the composite samples.

Table 13-2: Head Assay of the Composite Samples

Sample ID	Compound %	
	East Zone Composite	West Zone Composite
SiO_2	2.59	5.91
Al_2O_3	0.38	0.17
Fe_2O_3	0.82	1.06
MgO	43.6	42.6
CaO	0.89	1.68
Na_2O	<0.01	0.02
K_2O	0.06	0.03
TiO_2	0.03	0.02
P_2O_5	<0.01	<0.01
MnO	0.04	0.04
Cr_2O_3	<0.01	<0.01
V_2O_5	<0.01	<0.01
LOI	50.2	48.4
Sum	98.6	99.9

Source: *SGS (2008)*
Note: *LOI = loss on ignition; < = less than*

Table 13-3: Estimated Mineral Assays in Composite Samples

Sample ID	Magnesite (%)	SiO_2 (%)	Calcite(*) (%)	Dolomite(**) (%)	MgO in Dolomite	Corrected Magnesite (%)
East Composite	91.19	2.59	1.59	2.93	0.64	89.9
West Composite	89.10	5.91	3.00	5.53	1.21	86.6

Source: *SGS (2008)*
Notes: **Assuming that all CaO is in Calcite. **Assuming that all CaO is in Dolomite. % = percent;*

13.1.5 X-Ray Diffraction Analysis

Semi-quantitative XRD was performed on the composite samples, and the results are shown in Table 13-4. It should be noted that XRD results are only semi-quantitative and inaccurate when the mineral contents are less than 5%.

With conservative estimations, it appears that the magnesite assays of the West and East Zones are approximately 86% to 89%, and 90% to 91%, respectively. The silicate contents of the composite samples were 6% and 3% in the West and East Zone composite samples, respectively. Iron assays of the West and East Zone composites were 0.61% and 0.57%, respectively.

Table 13-4: Results from XRD Semi-Quantitative Phase Analysis on Head Samples

Mineral Name	Semi-Quantitative (wt%)	
	West Zone Composite	East Zone Composite
Quartz	6.00	2.50
Dolomite	6.20	2.70
Magnesite	86.3	93.4
Siderite	0.60	0.50
Pyrite	0.80	0.90
Total	**99.9**	**100**

Note: *Software used was Bruker AXS Difrac Plus EVA.*

13.1.6 Screening and Fractional Analysis

A 2 kg charge (-10 mesh) was screened into four fractions without any primary grinding. A sample from each fraction was submitted for chemical analysis. The results are shown in Table 13-5, and illustrated in Figure 13-3.

In the East Zone sample, MgO assays were fairly consistent in all fractions, whereas the SiO_2 assay was higher (3.5%) in the -10/+48 fraction. In this fraction, the alumina distribution had significantly increased to much higher than the corresponding mass distribution. Silica showed the same trend, but to a lesser extent. In the West Zone sample, the MgO grade was reduced slightly in the -200 mesh fraction. The silica assay increased slightly in both the coarsest (-10/+48 mesh) and finest (-200 mesh) fractions. Similar to the East Zone composite sample, the alumina distribution increased in the -10/+48 mesh fraction, larger than the corresponding mass distribution. The distributions of the other elements followed the mass distribution fairly closely. In conclusion, pre-screening cannot generate any reasonable magnesite upgrading.

Table 13-5: Fractional Analysis

Sample ID Mesh	Weight		SiO_2 %		Al_2O_3 %		Fe_2O_3 %		MgO %		CaO %		LOI (%)		Sum
	(g)	(%)	Grade	Dist.	Grade	Dist.	Grade	Dist.	Grade	Dist.	Grade	Dist.	Grade	Dist.	(%)
WZ -10 +48	316	62.8	6.54	67.6	0.36	82.4	1.12	64.8	42.8	63.2	1.93	67.6	48.1	62.4	101
WZ -48 +100	66.9	13.3	4.75	10.4	0.16	7.75	1.07	13.1	42.4	13.2	1.49	11.0	49.3	13.5	99.3
WZ -100 +200	46.4	9.22	5.37	8.15	0.12	4.03	1.02	8.66	43.0	9.32	1.59	8.17	48.9	9.32	100.1
WZ -200	73.7	14.7	5.77	13.9	0.11	5.87	1.00	13.5	41.5	14.3	1.62	13.2	48.7	14.7	98.8
Head (Calc.)	503	100	6.08	100	0.27	100	1.09	100	42.6	100	1.79	100	48.4	100	100.4
EZ -10 +48	332	66.0	3.48	74.4	0.86	84.4	0.92	69.5	44.3	65.8	0.87	65.8	49.6	65.7	100
EZ -48 +100	59.4	11.8	2.23	8.53	0.32	5.62	0.75	10.1	44.9	11.9	0.84	11.4	50.5	12.0	99.7
EZ -100 +200	40.4	8.03	2.16	5.62	0.29	3.46	0.80	7.35	45.1	8.16	0.87	8.01	50.7	8.17	100
EZ -200	70.6	14.0	2.51	11.4	0.31	6.47	0.81	13.0	44.5	14.1	0.92	14.8	50.4	14.2	99.5
Head (Calc.)	502.0	100	3.09	100	0.67	100	0.87	100	44.4	100	0.87	100	49.8	100	99.8

Source: SGS (2008)

Notes: *WZ = West Zone; EZ = East Zone; Dist. = distribution; LOI = loss on ignition; Calc. = calculated; ID = identification; g = grams; % = percent*

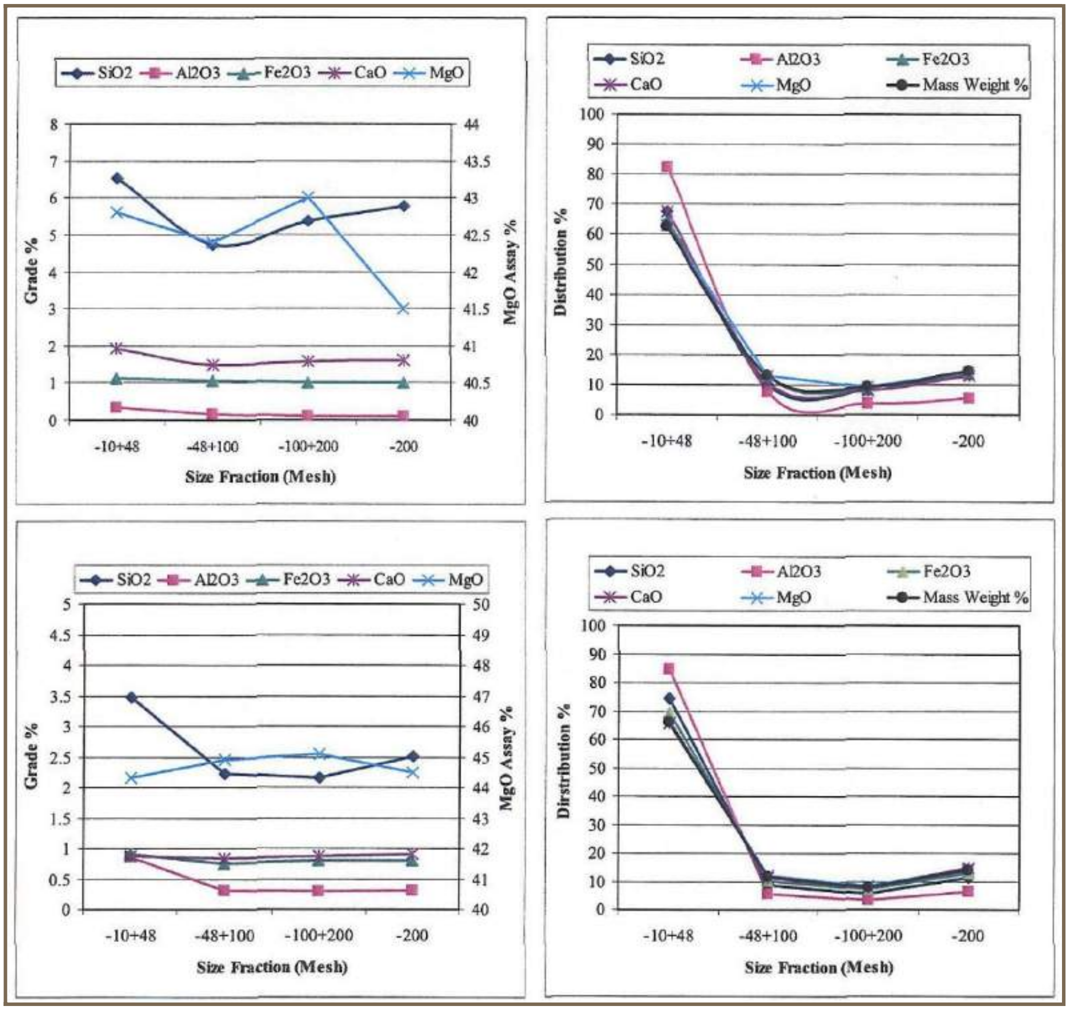


Figure 13-3: Grade and Distribution of Composite Components in Different Fractions

13.1.7 Heavy Liquid Separation

Heavy liquid separation was performed on the West Zone sample to evaluate the potential of gravity and HMS. The first series of heavy liquid tests were performed on the -10/+20 mesh fraction from the West Zone composite. The densities of the heavy liquids tested were 2.80, 2.85, 2.90, 2.95, 3.00, 3.05, and 3.10 g/cm^3. The results are presented in Table 13-6 and Figure 13-4. From the figure, it can be concluded that the density of the media in a dense medium separator (DMS) should be around 2.92 g/cm^3. At such a density, approximately 80% of the quartz and 42% of the dolomite can be separated with a loss of about 10% magnesite. There is, however, no success in reducing the iron content of the magnesite concentrate. Due to the similar separation behaviour of iron and magnesium in these tests, it can be speculated either that there is a liberation issue, or that the iron is substituted in a magnesite crystal structure.

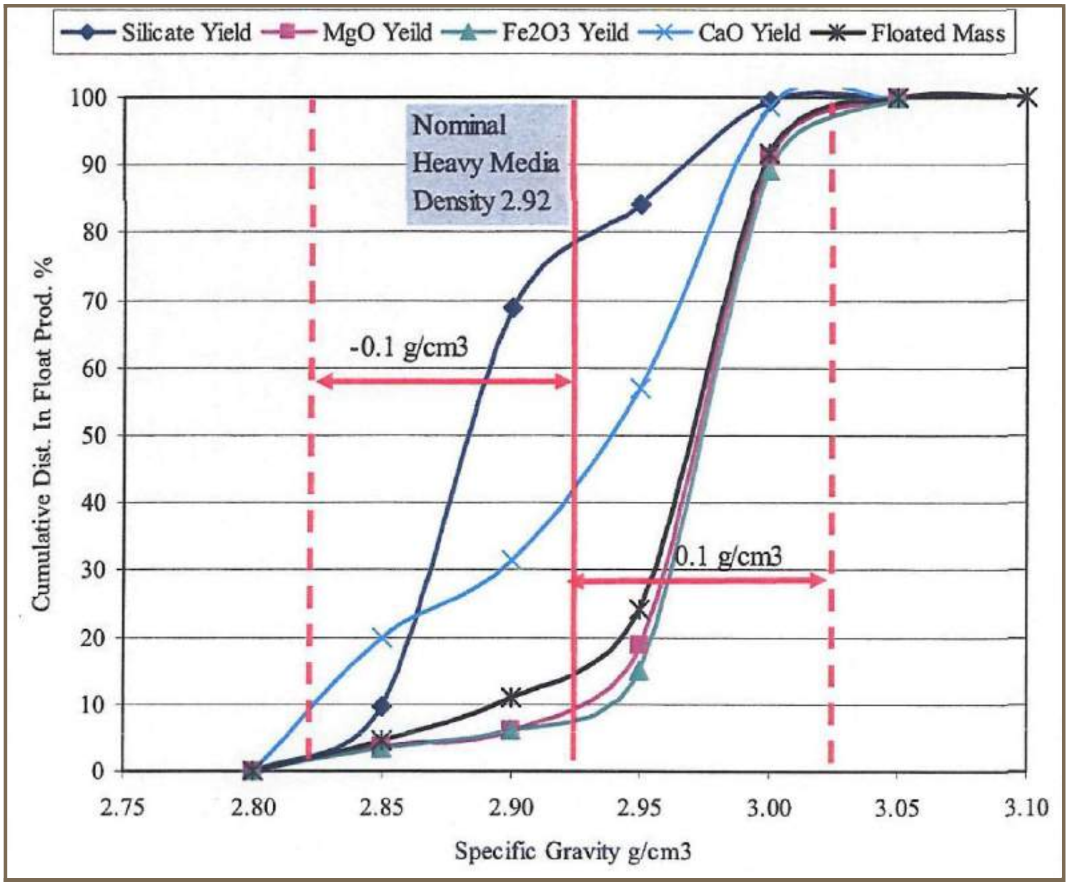


Figure 13-4: Plots of Heavy Liquid Results on Coarse Fraction (-10/+20Mesh)

Table 13-6: Heavy Liquid Separation Test Results on -10/+20 Mesh Fraction

Heavy Liq. Den. SG (g/cm³)	Weight (g)	Weight (%)	Separation Density (g/cm³)	Cumulative Float (%)	Cumulative Sink (%)	Assay	Dist.	Cum. Dist. in Float	Assay of Cumulative Floated Production	Assay of Cumulative Sunk Production
					SiO_2 %					
-2.80	0.02	0.00	2.80	0.00	100	0.00	0.00	0.00	0.00	6.24
-2.85 +2.80	29.7	4.68	2.85	4.68	95.3	12.8	9.59	9.59	12.8	5.92
-2.90 +2.85	39.5	6.22	2.90	10.9	89.1	59.4	59.2	68.8	39.4	2.19
-2.95 +2.90	84.8	13.3	2.95	24.2	75.8	7.20	15.4	84.2	21.7	1.30
-3 +2.95	428	67.4	3.00	91.6	8.36	1.42	15.3	99.5	6.78	0.34
-3.05 +3.00	53.0	8.35	3.05	100	0.00	0.35	0.47	100	6.24	0.00
-3.10 +3.05	0.02	0.00	3.10	100	0.00	6.24	0.00	-	-	-
+3.10	0.02	0.00	-	100	0.00	0.00	0.00	-	-	-
Total (Calc.)	**635**	**100**	-	-	-	**6.24**	**100**	-	-	-

Heavy Liq. Den.	Weight		Separation	Cumulative		Assay	Dist.	Cum. Dist.	Assay of Cumulative	
MgO %										
-2.80	0.02	0.00	2.80	0.00	100	0.00	0.00	0.00	0.00	43.5
-2.85 +2.80	29.7	4.68	2.85	4.68	95.3	35.4	3.80	3.80	35.4	43.9
-2.90 +2.85	39.5	6.22	2.90	10.9	89.1	16.6	2.37	6.17	24.7	45.9
-2.95 +2.90	84.8	13.3	2.95	24.2	75.8	41.6	12.7	18.9	34.0	46.6
-3.00 +2.95	428	67.4	3.00	91.6	8.36	46.6	72.1	91.0	43.3	46.7
-3.05 +3.00	53.0	8.35	3.05	100	0.00	46.7	8.96	100	43.5	-
-3.10 +3.05	0.02	0.00	3.10	100	0.00	43.5	0.00	-	-	-
+3.10	0.02	0.00	-	100	0.00	0.00	0.00	-	-	-
Total (Calc.)	**635**	**100**	**-**	**-**	**-**	**43.55**	**100**	**-**	**-**	**-**
Fe_2O_3 %										
-2.80	0.02	0.00	2.80	0.00	100	0.00	0.00	0.00	0.00	1.14
-2.85 +2.80	29.7	4.68	2.85	4.68	95.3	0.85	3.50	3.50	0.85	1.15
-2.90 +2.85	39.5	6.22	2.90	10.9	89.1	0.47	2.57	6.07	0.63	1.20
-2.95 +2.90	84.8	13.3	2.95	24.2	75.8	0.77	9.04	15.1	0.71	1.27
-3.00 +2.95	428	67.4	3.00	91.6	8.36	1.25	74.1	89.2	1.11	1.47
-3.05 +3.00	53.0	8.35	3.05	100	0.00	1.47	10.8	100	1.14	0.00
-3.10 +3.05	0.02	0.00	3.10	100	0.00	1.14	0.00	-	-	-
+3.10	0.02	0.00	-	100	-0.01	0.00	0.00	-	-	-
Total (Calc.)	**635**	**100**	**-**	**-**	**-**	**1.14**	**100**	**-**	**-**	**-**
CaO %										
-2.80	0.02	0.00	2.80	0.00	100	0.00	0.00	0.00	0.00	1.66
-2.85 +2.80	29.7	4.68	2.85	4.68	95.3	7.02	19.8	19.8	7.02	1.39
-2.90 +2.85	39.5	6.22	2.90	10.9	89.1	3.12	11.7	31.5	4.79	1.27
-2.95 +2.90	84.8	13.3	2.95	24.2	75.8	3.15	25.4	56.9	3.89	0.94
-3.00 +2.95	428	67.4	3.00	91.6	8.36	1.03	41.9	98.8	1.79	0.24
-3.05 +3.00	53.0	8.35	3.05	100	0.00	0.24	1.21	100	1.66	0.00
-3.10 +3.05	0.02	0.00	3.10	100	0.00	1.66	0.00	-	-	-
+3.10	0.02	0.00	-	100	0.00	0.00	0.00	-	-	-
Total (Calc.)	**635**	**100**	**-**	**-**	**-**	**1.66**	**100**	**-**	**-**	**-**

Source: *SGS (2008)*

Notes: *Calc. = calculated; % = percent; g/cm^3 = grams per cubic centimetre; Dist. = distribution; g = grams; SG = specific gravity; Cum. = cumulative*

The next series of heavy liquid separations was conducted on the finer fraction, -20/+48 mesh, of the West zone composite. The results are shown in Figure 13-5 and Table 13-7 and are fairly consistent with those reported for the coarser fraction (-10/+20 Mesh).

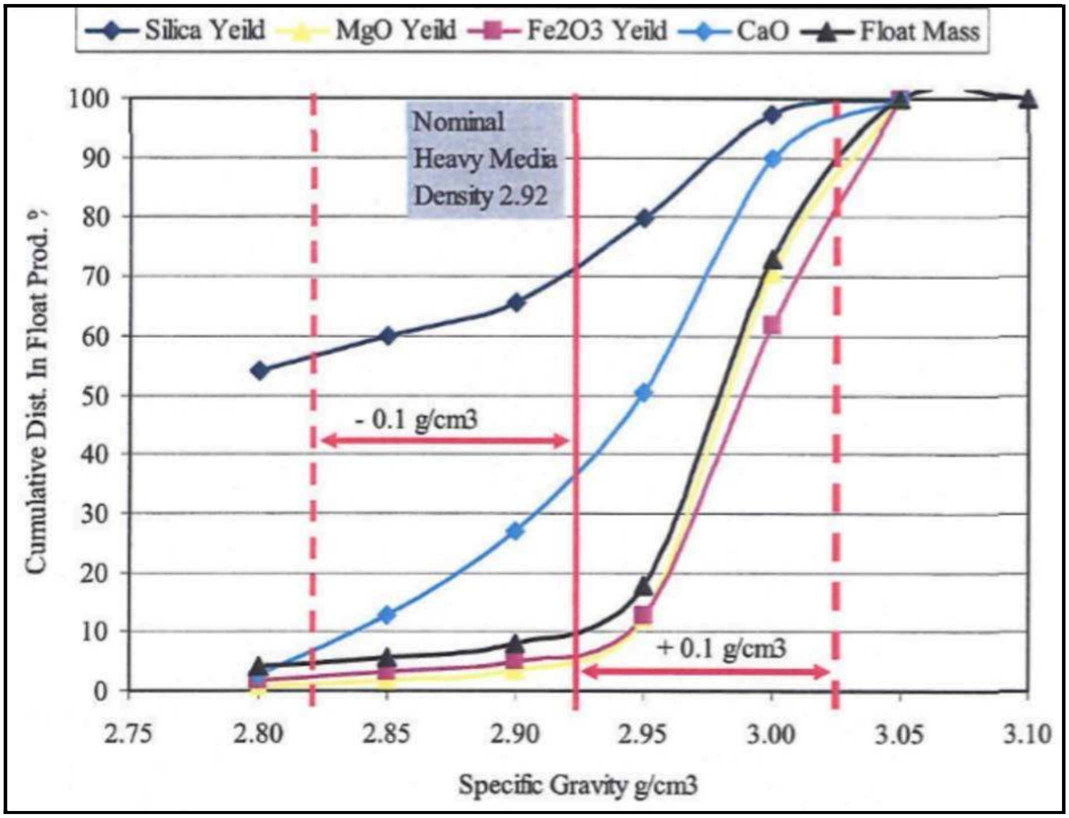


Figure 13-5: Plots of Heavy Liquid Results on Intermediate Fraction (-20/+48Mesh)

Table 13-7: Heavy Liquid Separation Test Results on -20/+48 Mesh Fraction

Heavy Liq. Den. SG (g/cm³)	Weight (g)	Weight (%)	Separation Density (g/cm³)	Cumulative Float (%)	Cumulative Sink (%)	Assay	Dist.	Cum. Dist. in Float	Assay of Cumulative Floated Production	Assay of Cumulative Sunk Production
						SiO_2 %				
-2.80	28.0	4.11	2.80	4.11	95.9	80.3	54.0	54.0	80.3	2.93
-2.85 +2.80	10.2	1.49	2.85	5.60	94.4	23.8	5.82	59.8	65.2	2.60
-2.90 +2.85	16.7	2.44	2.90	8.04	92.0	14.3	5.71	65.6	49.8	2.29
-2.95 +2.90	67.2	9.85	2.95	17.9	82.1	8.91	14.4	79.9	27.3	1.49
-3 +2.95	375	55.0	3.00	72.9	27.1	1.92	17.3	97.2	8.15	0.63
-3.05 +3.00	185	27.1	3.05	100	0.00	0.63	2.80	100	6.11	0.00
-3.10 +3.05	0.07	0.01	3.10	100	0.00	0.00	0.00	-	-	-
+3.10	0.06	0.01	-	100	0.00	0.00	0.00	-	-	-
Total (Calc.)	**682**	**100**	**-**	**-**	**-**	**6.11**	**100**	**-**	**-**	**-**
Initial Weight	**684**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**

Heavy Liq. Den.	Weight		Separation	Cumulative		Assay	Dist.	Cum. Dist.	Assay of Cumulative	
MgO %										
-2.80	28.0	4.11	2.80	4.11	95.9	8.42	0.81	0.81	8.42	44.3
-2.85 +2.80	10.2	1.49	2.85	5.60	94.4	25.1	0.88	1.68	12.9	44.6
-2.90 +2.85	16.7	2.44	2.90	8.04	92.0	31.8	1.81	3.49	18.6	45.0
-2.95 +2.90	67.2	9.85	2.95	17.9	82.1	38.9	8.94	12.4	29.8	45.7
-3.00 +2.95	375	55.0	3.00	72.9	27.1	45.5	58.3	70.8	41.6	46.2
-3.05 +3.00	185	27.1	3.05	100	0.00	46.2	29.2	100	42.8	00
-3.10 +3.05	0.07	0.01	3.10	100	0.00	0.00	0.00	-	-	-
+3.10	0.06	0.01	-	100	0.00	0.00	0.00	-	-	-
Total (Calc.)	**682**	**100**	**-**	**-**	**-**	**42.8**	**100**	**-**	**-**	**-**
Fe_2O_3 %										
-2.80	28.0	4.11	2.80	4.11	98.9	0.46	1.87	1.87	0.46	1.03
-2.85 +2.80	10.2	1.49	2.85	5.60	94.4	0.84	1.24	3.12	0.56	1.03
-2.90 +2.85	16.7	2.44	2.90	8.04	92.0	0.83	2.01	5.13	0.64	1.04
-2.95 +2.90	67.2	9.85	2.97	17.9	82.1	0.78	7.62	12.7	0.72	1.07
-3.00 +2.95	375	55.0	3.00	72.9	27.1	0.90	49.1	61.8	0.86	1.4
-3.05 +3.00	185	27.1	3.05	100	0.00	1.42	38.2	100	1.01	0.00
-3.10 +3.05	0.07	0.01	3.10	100	0.00	0.00	0.00	-	-	-
+3.10	0.06	0.01	-	100	0.00	0.00	0.00	-	-	-
Total (Calc.)	**682**	**100**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
CaO %										
-2.80	28.0	4.11	2.80	4.11	95.9	1.12	2.65	2.65	1.12	1.76
-2.85 +2.80	10.2	1.49	2.85	5.60	94.4	11.6	9.97	12.6	3.91	1.61
-2.90 +2.85	16.7	2.4	2.90	8.04	92.0	10.3	14.5	27.1	5.85	1.38
-2.95 +2.90	67.2	9.85	2.95	17.9	82.1	4.10	23.2	50.3	4.89	1.05
-3.00 +2.95	375	55.0	3.00	72.9	27.1	1.25	39.5	89.9	2.14	0.65
-3.05 +3.00	185	27.1	3.05	100	0.00	0.65	10.1	100	1.74	0.00
-3.10 +3.05	0.07	0.01	3.10	100	0.00	0.00	0.00	-	-	-
+3.10	0.06	0.01	-	100	0.00	0.00	0.00	-	-	-
Total (Calc.)	**682**	**100**	**-**	**-**	**-**	**1.74**	**100**	**-**	**-**	**-**

Source: *SGS (2008)*

Notes: *Calc. = calculated; % = percent; g/cm³ = grams per cubic centimetre; Dist. = distribution; g = grams; SG = specific gravity; Cum. = cumulative*

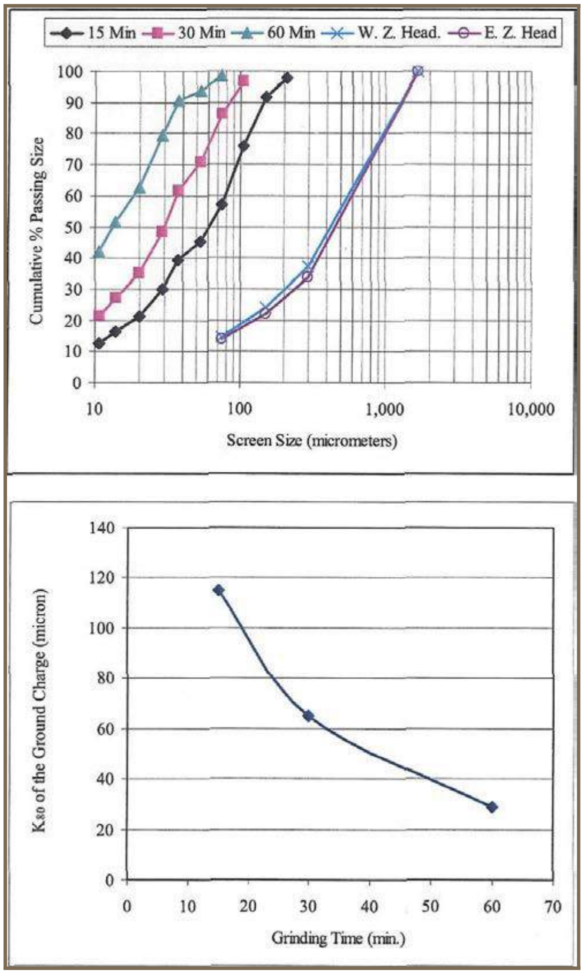
In this fraction, about 73% of the silicate minerals (mainly quartz) and 40% of the dolomite were rejected on the float product of heavy liquid separation at the density of 2.92 g/cm^3, with a loss of only 6% to 7% magnesite. Once again, it was observed that magnesia and iron oxide have similar separation profiles. If iron is found mainly in siderite, it is then reasonable to expect that siderite with a density heavier than magnesite (3.7 g/cm^3) would report to the sink product at the density of 3.1 g/cm^3. This was not the case. In comparison with the results from the coarser fraction (-10/+20 mesh), it is expected to have at least slightly better liberation between all the minerals, including magnesite and siderite, and consequently improved siderite separation. On the contrary, the weight of the material reporting to the sink product of the heavy liquid at a density of 3.10 g/cm^3 was negligible. Thus, either fine siderite grains were disseminated in the magnesite (which was not seen under the stereoscope), or the iron was in a magnesite crystal structure. A solid solution of iron present in a magnesite mineral is fairly common, due to the close radius of the iron and magnesium atoms.

The conclusions from these tests are:

- DMS can potentially be an effective method to remove a large portion of silicate (about 73% to 80%) and a moderate amount of dolomite/calcite (about 40%) from magnesite.
- The density of heavy media should be adjusted to 2.92 g/cm^3. The sink product can be considered as magnesite concentrate, with a grade of at least 45% MgO. About 10% of magnesite will be lost to the float fraction, mainly due to the interlocking with silicate and calcium minerals. The iron oxide content of the magnesite concentrate is expected to be approximately 1% Fe_2O_3 and most probably even higher (up to 1.25% Fe_2O_3) in the coarser fraction (10 to 20 mesh). The silicate assay of the magnesite concentrate is predicted to be at least 1.3% SiO_2, and most probably even higher (1.7% to 2%), whereas the calcium oxide assay is expected to be in the range of 1.1% to 1.3% CaO.
- Due to the fact that the weight percentage within ±0.1 g/cm^3 of the nominal media density of 2.92 g/cm^3 is greater than 25%, only DMS with close cut-point control can be used as a suitable gravity separation method.
- DMS is expected to have poor performance in reducing the iron content of the magnesite concentrate. This is most probably due to presence of iron in magnesite crystal structure.

13.1.8 Grinding Tests

Figure 13-6 shows the 2 kg batch milling curves obtained for the three grinding times on the West Zone sample. The milling curve was used to determine the grinding time required to obtain a feed size of 80% passing 150 µm to 200 µm. A milling time of 10 minutes was sufficient to achieve a feed size of 80% passing 150 µm.



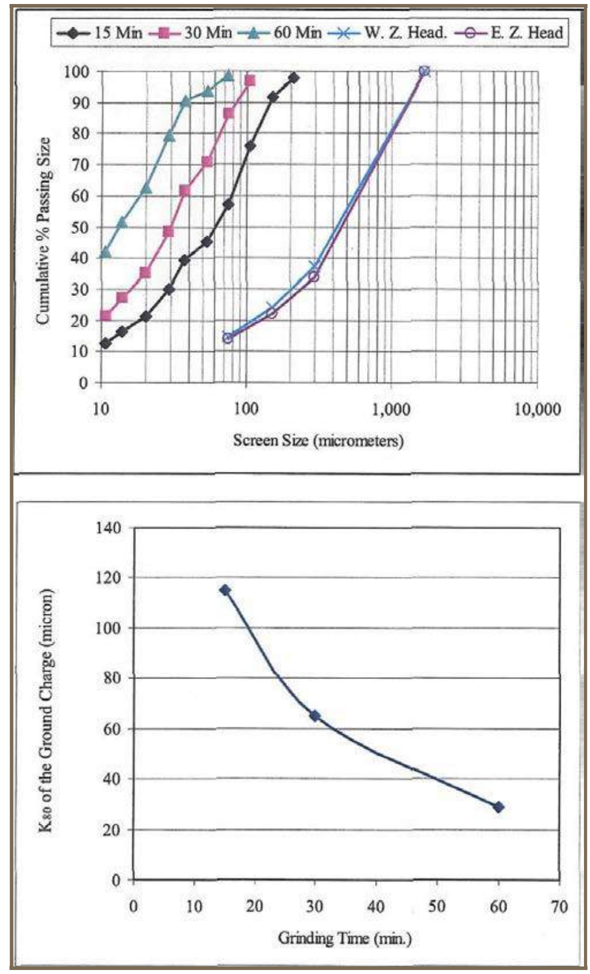


Figure 13-6: Particle Size Distributions and K_{80} as a Function of Grinding Time in a Laboratory Ball Mill, West Composite Sa

13.1.9 Flotation Tests

The development of a flotation flowsheet was undertaken through a series of rougher kinetic tests and cleaner tests on 2 kg composite charges. The objective of these tests was to evaluate the response of the composite sample to metallurgical drivers and produce high-grade magnesite concentrate. Both reverse and direct magnesite flotation were examined. The main gangue minerals in both the West Zone and East Zone composite samples were quartz and silicate minerals. As shown in Table 13-4, assay values for the silicate minerals are much lower than those of the magnesite in both composites. Thus, reverse magnesite flotation seemed to be even more attractive due to a lower mass pull. In reverse magnesite flotation, the silicate minerals were floated first, and the magnesite concentrate was recovered as silicate flotation tailings. Direct magnesite flotation was also examined on the West Zone composite samples. In these tests magnesite was floated, while other gangue minerals were depressed.

A minor amount of pyrite detected in composite samples was initially floated to reduce iron and sulphide content of the final magnesite concentrate.

Since the level of impurities in the West Zone composite was higher, it was decided to start the testwork on that composite. The flotation scheme developed was then used to evaluate the response of the East Zone composite samples to the flotation tests. Unless otherwise stated, in the following sections the word iron is loosely used as equivalent to iron oxide (Fe_2O_3).

Flotation Testwork on West Zone

Reverse Magnesite Flotation

Results from reverse magnesite flotation tests are shown in Table 13-8, Figure 13-7, and Figure 13-8. The effects of different silicate collectors, inorganic salts, and grind sizes were examined. Warm water was added to the flotation cell to raise the pulp to the target level and to adjust the pulp temperature to around 30°C. Other than in tests F2 and F5, conditioning for silicate flotation was performed after decantation and thickening to a 50% pulp density.

Flotation Test 1 (Fl) was conducted on a deslimed product using DA 16 collector at a dosage of 500 g/t at a natural pulp pH of 9.2. Desliming was done by decantation after pulp settlement. The K_{80} of the flotation feed was 200 µm. Under these conditions, about 54% of the silicates were separated, with a magnesite loss of 11%. About 16% of the silicate and 11% of the magnesite reported to the slime fraction. In the final magnesite concentrate, or silicate tailings, the MgO graded 45.4%, and recovery was 78%.

Table 13-8: Magnesite Reverse Flotation, West Zone

Magnesite Reverse Flotation													
Test No.	Products	Weight	Assay %					Distribution %					
	Name	%	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO
Test 1	SiO_2 Ro. Conc	13.3	23.4	0.39	0.95	1.07	35.4	53.6	31.7	11.6	8.44	10.8
500 g/t DA 16	Magnesite Conc.	74.9	2.33	0.07	1.06	1.71	45.4	30.0	31.9	72.5	75.8	78.0
k80 = 195	Slime	11.7	8.15	0.51	1.49	2.27	41.6	16.4	36.4	15.9	15.7	11.2
pH: 92	Head (calc)	100	5.82	0.16	1.10	1.69	43.6	100	100	100	100	100
Test 2	SiO_2 Ro. Conc 1	0.76	45.9	3.71	1.04	1.46	22.5	6.01	22.0	0.71	0.67	0.39
	SiO_2 Ro. Conc 1-2	1.19	46.0	3.37	0.99	1.52	22.6	9.41	31.2	1.05	1.09	0.62
k80 = 195	SiO_2 Ro. Conc 1-3	1.35	43.7	3.12	1.01	1.55	23.8	10.1	32.6	1.21	1.25	0.73
120 g/t ITU	SiO_2 Ro. Conc 1-4	5.30	43.2	0.96	0.80	1.31	24.9	39.2	39.6	3.79	4.16	3.01
60 g/t ENA	Magnesite Conc.	81.4	3.03	0.01	1.08	1.60	45.4	42.2	6.31	78.4	77.9	84.3
45 g/t DA 16	Slime	13.2	8.08	0.52	1.45	2.23	41.7	18.2	53.0	17.0	17.6	12.5
	Pyrite Conc.	0.19	9.61	0.68	4.34	3.03	38.3	0.32	1.01	0.75	0.35	0.17
	Head (calc)	100	5.83	0.13	1.12	1.67	43.8	100	100	100	100	100
Test 5	SiO_2 Ro. Conc 1	7.91	33.0	0.89	0.93	1.34	30.9	41.6	20.4	6.87	6.25	5.53
	SiO_2 Ro. Conc 1-2	19.4	18.1	0.55	1.06	1.32	38.4	56.1	30.8	19.2	15.0	16.8
k80 = 195	SiO_2 Ro. Conc 1-3	28.5	13.6	0.43	1.03	1.34	40.7	61.7	35.8	27.5	22.5	26.3
420 g/t MG83	Magnesite Conc.	70.2	3.29	0.30	1.06	1.82	45.6	36.9	61.2	69.6	7.0	72.5
1000 g/t $CaCl_2$	Sulphide Conc	1.30	7.11	0.80	2.42	2.81	41.6	1.47	3.01	2.93	2.15	1.22
	Head (calc)	100	6.27	0.34	1.07	1.70	44.2	100	100	100	100	100
Test 6	SiO_2 Ro. Conc 1	7.06	61.5	0.88	0.63	1.17	16.7	72.1	37.4	4.18	5.02	2.82
	SiO_2 Ro. Conc 1-2	19.3	28.6	0.56	0.90	1.84	31.2	91.7	64.7	16.3	21.5	14.4
k80 = 195	SiO_2 Ro. Conc 1-3	30.9	18.5	0.43	1.00	1.95	35.6	94.8	80.1	29.0	36.6	26.3
625 g/t ArmacFlot 109	Mg Conc	66.7	0.21	0.02	1.05	1.47	44.8	2.33	8.03	65.8	59.6	71.4
	Sulphide Conc	2.43	7.05	0.81	2.27	2.60	39.4	2.85	11.9	5.20	3.85	2.29
	Head (calc)	100	6.02	0.17	1.06	1.69	41.8	100	100	100	100	100
Test 8	SiO_2 Ro. Conc 1	12.6	45.9	3.71	1.04	1.46	22.5	43.3	35.0	11.6	10.0	10.5
	SiO_2 Ro. Conc 1-2	21.9	15.3	0.46	1.06	1.44	37.2	58.1	52.2	21.3	18.7	19.5
k80 = 195	SiO_2 Ro. Conc 1-3	25.5	14.2	0.43	1.06	1.46	37.8	62.9	56.6	24.7	22.0	23.0
1650 g/t CaCl2	$FeCO_3$ Ro. Conc 1-4	30.7	15.9	0.40	1.01	1.51	37.0	84.6	62.9	28.5	27.4	27.1
370 g/t DA16	$FeCO_3$ Ro. Conc 1-5	31.2	15.6	0.39	1.02	1.52	37.1	84.6	63.2	29.2	28.1	27.7
90 g/t A845	Magnesite Conc.	66.4	1.10	0.08	1.08	1.74	44.2	12.7	27.2	65.8	68.4	70.0
	Sulphide Conc	2.42	6.53	0.77	2.26	2.50	39.6	2.74	9.56	5.02	3.58	2.29
	Head (calc)	100	5.76	0.19	1.09	1.61	41.9	100	100	100	100	100
Test 9	SiO_2 Ro. Conc 1	7.98	55.6	1.02	0.63	1.31	18.5	75.8	28.2	4.58	6.50	3.53
	SiO2 Ro. Conc 1-2	20.5	26.3	0.78	1.01	2.08	31.6	92.2	55.1	18.9	26.6	15.5
k80 = 195	SiO_2 Ro. Conc 1-3	31.7	17.3	0.56	1.04	1.99	35.8	93.5	61.3	30.2	39.3	27.1
625 g/t ArmacFlot 109	$FeCO_3$ Ro. Conc 1-4	37.7	14.5	0.49	1.06	1.84	37.2	93.6	63.4	36.3	43.1	33.5
	$FeCO_3$ Ro. Conc 1-5	44.7	12.3	0.42	1.06	1.68	38.5	93.7	65.1	43.2	46.6	41.1
	Magnesite Conc.	52.5	0.35	0.15	1.07	1.50	44.9	3.14	27.3	51.2	49.0	56.2
	Sulphide Conc	2.79	6.71	0.79	2.22	2.55	40.0	3.19	7.63	5.63	4.42	2.66
	Head (calc)	100	5.85	0.29	1.10	1.61	41.9	100	100	100	100	100
Test 10	SiO_2 R. Conc 1	10.5	49.1	1.07	1.24	1.62	22.3	88.3	57.5	7.62	9.74	5.49
	SiO_2 R. Conc 1-2	18.6	29.8	0.70	1.56	1.89	31.1	95.0	66.6	17.0	20.2	13.6
k80 = 195	Magnesite Conc.	81.4	0.36	0.08	1.74	1.71	45.2	5.03	33.4	83.0	79.8	86.4
500 g/t Armac 1225	SiO_2 Feed (calc)	100	5.91	0.17	1.06	1.68	42.6	100	100	100	100	100
Test 11	SiO_2 Ro. Conc 1	10.8	61.5	0.88	0.63	1.17	16.7	76.9	50.4	6.69	7.94	4.45
	SiO_2 Ro. Conc 1-2	29.0	29.0	0.56	0.89	1.83	31.0	97.1	86.0	25.4	33.2	22.1
k80 = 98	SiO_2 Ro. Conc 1-3	30.7	27.5	0.54	0.91	1.84	31.6	97.4	87.9	27.3	35.4	23.8
750 g/t Armacflot 109	Magnesite Conc.	68.2	0.21	0.02	1.05	1.47	44.8	1.65	7.20	70.1	62.7	75.0
100 g/t SC1	Sulphide Conc	1.15	7.05	0.81	2.27	2.60	39.4	0.94	4.92	2.56	1.87	1.11
	Head (calc)	100	8.67	0.19	1.02	1.60	40.7	100	100	100	100	100
Test 13	SiO_2 Ro. Conc 1	12.8	44.2	0.86	0.99	1.92	24.5	92.3	53.0	10.8	14.9	7.32
	SiO_2 Ro. Conc 1-2	22.2	26.0	0.59	1.17	2.18	32.4	94.2	63.4	22.1	29.4	16.8
k80 = 72	SiO2 Ro. Conc 1-3	25.7	22.6	0.53	1.19	2.08	34.1	94.6	66.1	26.0	32.4	20.5
700 g/t Armacflot 109	Magnesite Conc.	72.3	0.28	0.08	1.11	1.47	46.1	3.30	27.8	68.3	64.5	77.8
	Sulphide Conc	1.93	6.78	0.66	3.41	2.65	39.2	2.13	6.12	5.61	3.10	1.77
	Head (calc)	100	5.91	0.17	1.06	1.68	42.6	100	100	100	100	100
Test 17	SiO_2 Ro. Conc 1	8.72	58.4	1.33	0.66	1.27	17.4	86.1	58.7	5.38	6.83	3.62
	SiO_2 Ro. Conc 2	11.0	47.6	1.13	0.81	1.51	22.3	89.0	62.9	8.32	10.3	5.87
k80 = 195	SiO_2 Ro. Conc 3	12.3	43.4	1.04	0.86	1.60	24.2	90.4	65.0	9.94	12.2	7.11
350 g/t Armac 1225	SiO_2 Ro. Conc 4	13.2	40.8	0.99	0.90	1.64	25.4	90.9	65.8	11.1	13.4	7.99
	SiO_2 Ro. Tail	85.6	0.52	0.07	1.07	1.60	44.5	7.53	30.3	85.7	84.5	90.9
	Sulphide Conc	1.21	7.79	0.64	2.84	2.80	38.9	1.60	3.92	3.22	2.09	1.12
	Head (calc)	100	5.91	0.20	1.07	1.62	41.9	100	100	100	100	100

Source: *SGS (2008)*

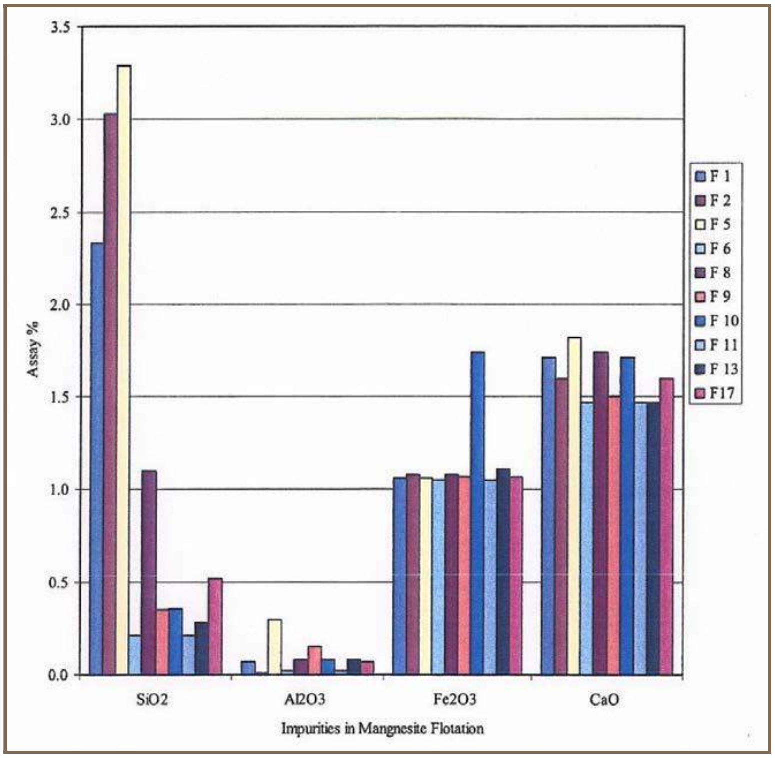


Figure 13-7: Impurities in Magnesite Concentrate Reverse Flotation, West Zone

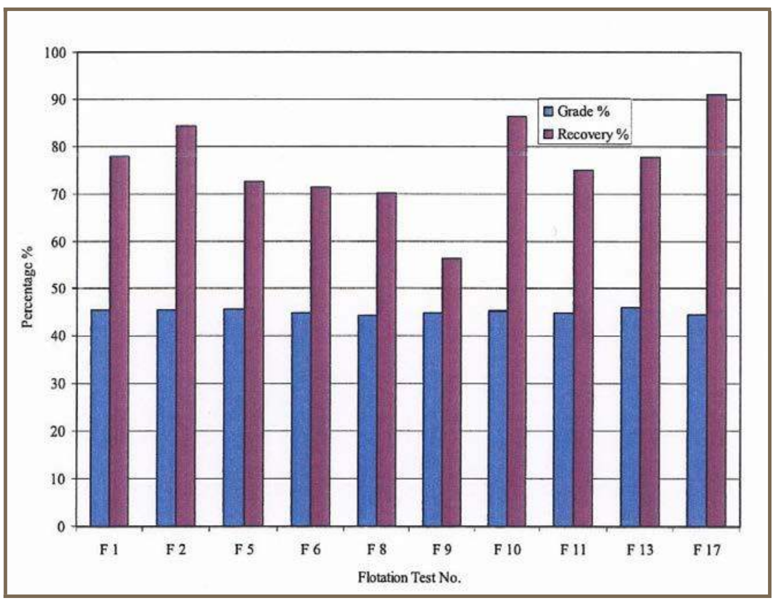


Figure 13-8: Grade and Recovery of Magnesite Concentrate Reverse Flotation, West Zone

In test F2, ITU and ENA collectors were tested, but the results were not that promising. Consequently, it was decided to use DA 16 collector in addition to the aforementioned collectors to improve silicate recovery. A large portion of silicate was recovered at that stage. The silicate grade in magnesite concentrate was about 3% SiO_2, and magnesite recovery reached 84%.

In Test F5, the effect of MG83 collector was examined in combination with an activator for silicate minerals, $CaCl_2$. No attempt was made to separate slimes in this test. About 62% of the silicates were floated and separated from the magnesite. Silicate grade in the tailings, or magnesite concentrate, was still about 3.3%, and magnesite recovery was less than 73%.

Armacflot 109 was used in test F6. The results were promising, with about 95% of the silicate floated and separated from the magnesite. In addition, 80% of the alumina and 36% of the CaO were also removed. About 26% of the magnesite was also lost in the silicate concentrate, resulting in low magnesite recovery in this test, at about 72%. Approximately 65% of the iron reported to the magnesite concentrate. Thus, there was almost no success in reducing the iron content of the magnesite concentrate. In the magnesite concentrate, the grade reached about 45% MgO, and the silicate assay was 0.2% SiO_2.

In test F8, DA 16 and $CaCl_2$ were used to separate silicate, and A-845 to float siderite. According to XRD analysis, the main source of iron in the composite samples is siderite. In this test, the magnesite concentrate grade was 44% MgO containing about 1% silica. The iron assay of this concentrate did not decrease to any significant level in comparison with Test F6.

Due to the poor performance of A-845 collector in test F8, it was decided to try an experimentally developed siderite collector, SCI, in test F9. This collector did not perform any better. Approximately 14% of magnesite reported to the siderite concentrate. Even after such a negative impact on magnesite recovery, the iron assay of the silicate tailings was still above 1% Fe_2O_3.

In test F10, the effect of Armac 1225 collector was tested. This collector appeared to be very selective toward silicate flotation. In silicate tailings (magnesite concentrate), the silicate grade was reduced to 0.36% SiO_2. About 95% of the silicate reported to the silicate concentrate.

Magnesite recovery was about 86%, and its grade reached about 45% MgO. In the magnesite concentrate, the iron assay was 1.7% Fe_2C>3, and 83% of the iron reported to the silicate tailings.

Due to the high iron content of the magnesite concentrate, it was decided to increase primary grinding to possibly improve liberation between magnesite and iron minerals. In test F11, the K_{80} of the silicate flotation feed was reduced to 98 µm. The result was promising in terms of silicate rejection. About 94% of the silica was reported to the silica concentrate, at a loss of 24% of the magnesium oxide. As a result, the grade reached 45% MgO in the magnesite concentrate. The iron content of the magnesite concentrate was still high, slightly above 1% Fe_2O_3. In test F13, the primary grinding was further increased to reduce the K_{80} of the flotation feed to about 73 microns (µm). The iron assay of the magnesite concentrate was still about 1.1% Fe_2O_3. These unsuccessful attempts in reducing the iron content of the magnesite concentrates are consistent with the previous hypothesis that iron is most probably substituted in the magnesite crystal structure.

Test F17 followed the same flotation scheme as test F10 with a finer grind (K_{80} of 140 µm). Results were better than expected. About 91% of the magnesite was recovered in the silicate tailings. Silicate, iron, and magnesium oxide assays were 0.52%, 1.07%, and 44.5%, respectively. Total sulphur in magnesite concentrate was below the detection limit of the sulphur analyzer, which is 0.01% S. About 1.1% of the magnesite was lost in the sulphide pre-concentrate, and the sulphur assay in this stream was 0.51%. In conclusion, Armac 1225 is a very efficient and selective collector to float silicate minerals in this mineralized system. An approximate range from 350 g/t to 500 g/t of this collector is required to achieve good silicate separation from magnesite. Test F17 is considered the best test performed on the West Zone composite sample.

Direct Magnesite Flotation

Results from direct magnesite flotation tests are shown in Table 13-9, Figure 13-9, and Figure 13-10. The effects of different magnesite collectors and inorganic and organic depressants were examined. Warm water was added to the flotation cell to raise the pulp level and adjust the pulp temperature to about 30°C. Other than in test F3, conditioning for magnesite flotation after pyrite flotation was performed after decantation and thickening to reach 50% pulp density.

Table 13-9: Magnesite Direct Flotation, West Zone

Test No	Products	Weight	Assay %					Distribution %				
	Name	%	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO
Test 3	Mg Ro. Conc 1	9.05	1.33	0.11	1.20	1.40	44.9	2.02	4.97	9.23	7.71	9.63
	Mg Ro. Conc 1-2	21.1	1.51	0.12	1.19	1.45	44.6	5.36	12.8	21.4	18.6	22.3
k80 = 195	Mg Ro. Conc 1-3	26.0	1.58	0.12	1.20	1.45	44.5	6.89	16.0	26.5	23.0	27.5
420 g/t Oleic Acid	Mg Ro. Conc 1-4	34.9	1.70	0.13	1.28	1.40	44.5	9.95	22.2	38.1	29.7	36.8
300 g/t FA-2	Mg Ro. Conc 1-5	55.7	2.07	0.12	1.21	1.46	44.2	19.4	34.7	57.4	49.6	58.4
500 g/t Na2SiO3	Mg Ro. Conc 1-6	81.0	2.67	0.13	1.16	1.57	43.9	36.3	51.1	80.2	77.5	84.3
	Mg Ro. Tailing	16.7	21.9	0.50	1.00	1.86	34.2	61.3	41.7	14.2	18.9	13.5
	Sulphide Conc 1	1.61	6.46	0.58	2.67	2.58	40.4	1.74	4.65	3.65	2.52	1.54
	Sulphide Conc 2	0.64	5.68	0.79	3.63	2.64	39.5	0.61	2.54	1.99	1.04	0.60
	Head (calc)	100	5.96	0.20	1.18	1.64	42.2	100	100	100	100	100
Test 4	Mg Ro. Conc 1	11.7	0.76	0.11	1.01	1.22	47.2	1.57	6.49	11.0	8.59	12.6
	Mg Ro. Conc 1-2	38.1	1.14	0.11	1.06	1.55	46.6	7.67	21.1	37.5	35.4	40.4
k80 = 195	Mg Ro. Conc 1-3	52.3	1.58	0.12	1.03	1.53	46.3	14.6	31.8	50.2	48.0	55.1
1585 g/t FA-2	Mg Ro. Conc 1-4	62.7	2.03	0.12	1.04	1.52	46.1	22.5	38.6	60.5	57.2	65.7
1000 g/t Na2SiO3	Mg Ro. Conc 1-5	72.2	2.50	0.13	1.04	1.54	45.7	31.8	46.3	69.9	66.9	75.0
100 g/t Quebraque	Mg Ro. Conc 1-6	87.7	2.98	0.13	1.05	1.64	45.4	46.1	59.6	85.9	86.7	90.6
	Mg Ro. Tailing	11.0	27.0	0.63	1.04	1.66	32.8	52.4	34.9	10.6	11.0	8.21
	Sulphide Ro Conc	1.29	6.70	0.84	2.90	2.93	41.7	1.53	5.47	3.49	2.28	1.23
	Head (calc)	100	5.67	0.20	1.07	1.66	44.0	100	100	100	100	100
Test 7	Mg Ro. Conc 1	36.8	1.96	0.08	0.99	1.58	43.8	12.6	15.8	34.9	35.7	38.5
	Mg Ro. Conc 1-2	73.4	2.30	0.11	1.01	1.62	43.8	29.5	43.2	71.0	73.1	76.5
k80 = 195	Mg Ro. Conc 1-3	82.8	2.90	0.12	1.01	1.60	43.4	42.0	52.8	80.2	81.2	85.7
650 g/t FS2	Mg Ro. Tailing	15.2	20.9	0.48	1.05	1.68	34.2	55.5	39.1	15.3	15.7	12.4
1000 g/t Na2SiO3	Sulphide Ro Conc	2.04	6.89	0.75	2.34	2.52	39.3	2.46	8.18	4.56	3.15	1.91
100 g/t Quebraque	Head (calc)	100	5.91	0.17	1.06	1.68	42.6	100	100	100	100	100
Test 12	Mg 2nd Cl. Conc	60.2	0.41	0.05	1.06	1.59	45.6	4.24	15.0	56.2	61.0	64.0
	Mg 1st Cl. Conc.	71.7	0.63	0.06	1.07	1.62	45.5	7.72	21.2	67.5	74.0	76.0
k80 = 195	Mg Ro. Conc.	86.9	1.33	0.09	1.09	1.62	45.2	19.9	37.8	83.3	89.6	91.5
750 g/t FA2	Mg Ro. Tail	10.4	5.80	1.07	1.26	0.94	24.5	77.1	55.3	11.6	6.24	5.95
1400 g/t Na2SiO3	Sulfide Ro Conc	2.64	3.04	6.82	5.12	4.16	2.51	3.04	6.82	5.12	4.16	2.51
160 g/t Quebraque	Head (calc)	100	5.83	0.20	1.14	1.57	42.9	100	100	100	100	100

Source: *SGS (2008)*

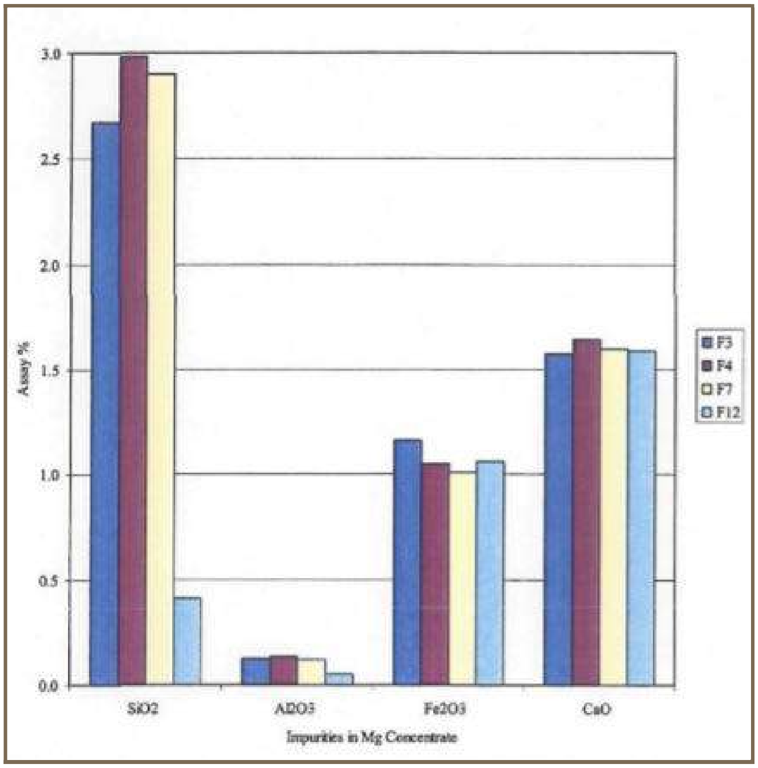


Figure 13-9: Impurities in Magnesite Concentrate, Direct Flotation, West Zone

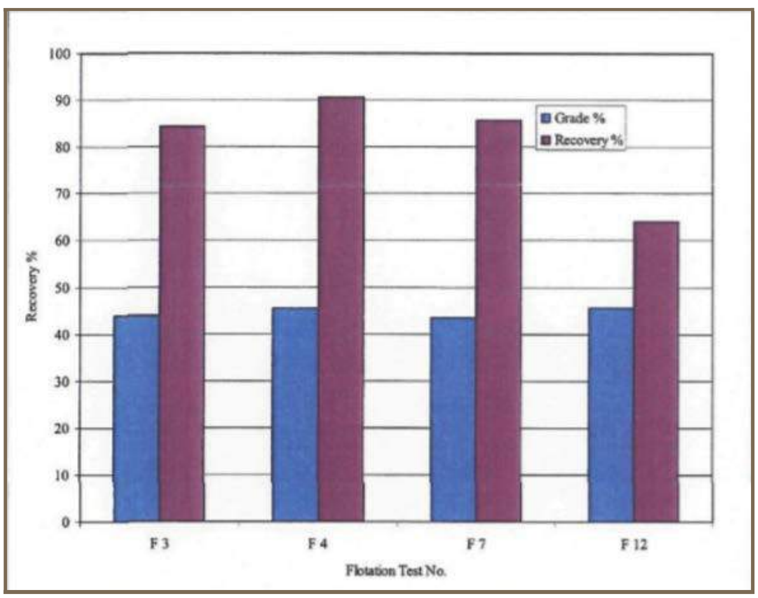


Figure 13-10: Grade and Recovery of Magnesite Concentrate, Direct Flotation, West Zone

Three collectors (FA-2, FS-2, and oleic acid) were attempted for magnesite flotation. Among them, FA-2 gave the best results. Although sodium silicate and Quebracho were used to depress silicate and carbonate minerals, the silicate assay of the rougher concentrate was higher than expected. In all the rougher flotation tests, the silicate assays in the magnesite concentrate were higher than 1.3% SiO_2. In the best case, test F12, approximately 77% of the silicates were rejected. In cleaner flotation, silicate rejection improved significantly and reached 95% after second cleaning, but the loss of magnesite was also high, at about 32%, neglecting any circulating load. No attempt was made to optimize the cleaning process, but it is clear that the efficiency of reverse flotation is far better. Furthermore, it is a simpler flowsheet to operate. As a result, further investigation on direct magnesite flotation was not continued.

Flotation Testwork on East Zone

Based on the experience developed on the West Zone sample, a flotation scheme for the East Zone sample was developed. Only reverse flotation was applied with the East Zone material. The results from these flotation tests are illustrated in Table 13-10, Figure 13-11, and Figure 13-12. These results clearly indicated that Armac 1225 is an excellent collector for selective silicate flotation. Finer grinding (a K_{80} of 84 µm in test F14) did not indicate an improvement in the reduction of gangue minerals in the magnesite concentrate. In test F16, 76% of the silicate was rejected in rougher flotation. Consequently, silicate and alumina assays in magnesite concentrate were reduced to 0.56% SiO_2 and 0.1% AI_2O_3. Iron and calcium oxide both assayed at 0.91% F_2O_3 and 0.91% CaO, and both were not rejected to any significant level in the magnesite concentrate. A high-grade magnesite concentrate, grading 46.4% MgO, with a recovery of 92.4%, was achieved from the East Zone sample.

Table 13-10: Magnesite Reverse Flotation, East Zone

			Assay %					Distribution %				
	Product Name	Weight	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO
Test 14 West Zone	SiO_2 Rougher Conc. 1-3	5.52	31.0	3.61	1.39	0.77	28.8	63.8	47.2	7.69	4.62	3.48
	SiO_2 Rougher Conc. 1-4	8.62	26.0	3.28	1.43	0.83	31.7	83.3	66.9	12.4	7.75	5.97
	Magnesite Conc.	90.1	0.43	0.14	0.93	0.93	47.1	14.4	29.9	84.2	90.8	92.8
	Sulphide Conc.	1.32	4.56	1.04	2.60	1.01	42.7	2.23	3.24	3.44	1.44	1.23
	Head (Calc.)	100	2.68	0.42	1.00	0.92	45.7	100	100	100	100	100
Test 15 East Zone	SiO_2 Ro. Conc. 1	1.89	12.9	7.04	1.25	0.88	34.5	17.1	11.0	2.77	1.64	1.23
	SiO_2 Ro. Conc. 1-2	3.03	16.5	7.72	1.19	0.83	32.3	30.2	17.6	4.43	2.58	1.87
	SiO_2 Ro. Conc. 1-3	3.68	18.7	7.37	1.17	0.81	31.6	35.1	22.5	5.59	3.19	2.31
	SiO_2 Ro. Conc. 1-4	4.40	25.2	6.33	1.08	0.75	29.3	37.6	26.2	6.80	3.96	2.93
	Magnesite Conc.	94.6	2.14	0.56	0.84	0.88	44.1	60.8	71.8	89.9	94.9	96.1
	Sulphide Conc.	1.04	5.03	1.41	2.80	0.98	41.0	1.57	1.98	3.29	1.16	0.98
	Head (Calc.)	100	3.3	0.74	0.88	0.88	43.4	100	100	100	100	100

	Product Name	Weight	Assay %					Distribution %				
Test 16 East Zone	SiO_2 Ro. Conc. 1	1.42	12.9	7.04	1.25	0.88	34.5	7.19	25.1	1.89	1.38	1.08
	SiO_2 Ro. Conc. 1-2	2.67	16.5	7.72	1.19	0.83	32.3	17.3	51.7	3.40	2.45	1.91
	SiO_2 Ro. Conc. 1-3	3.08	18.7	7.37	1.17	0.81	31.6	22.6	57.1	3.84	2.76	2.16
	SiO_2 Ro. Conc. 1-4	3.87	25.2	6.33	1.08	0.75	29.3	38.3	61.5	4.45	3.21	2.51
	Magnesite Conc.	8.33	23.5	3.45	0.98	0.80	31.8	76.9	72.3	8.70	90.5	92.4
	Sulphide Conc.	1.88	4.44	1.09	1.93	1.01	42.0	3.29	5.16	3.89	2.11	1.76
	Head (Calc.)	100	2.54	0.40	0.93	0.90	45.1	100	100	100	100	100

Source: *SGS (2008)*

Notes: *Conc. = concentrate; % = percent; Calc. = calculated*

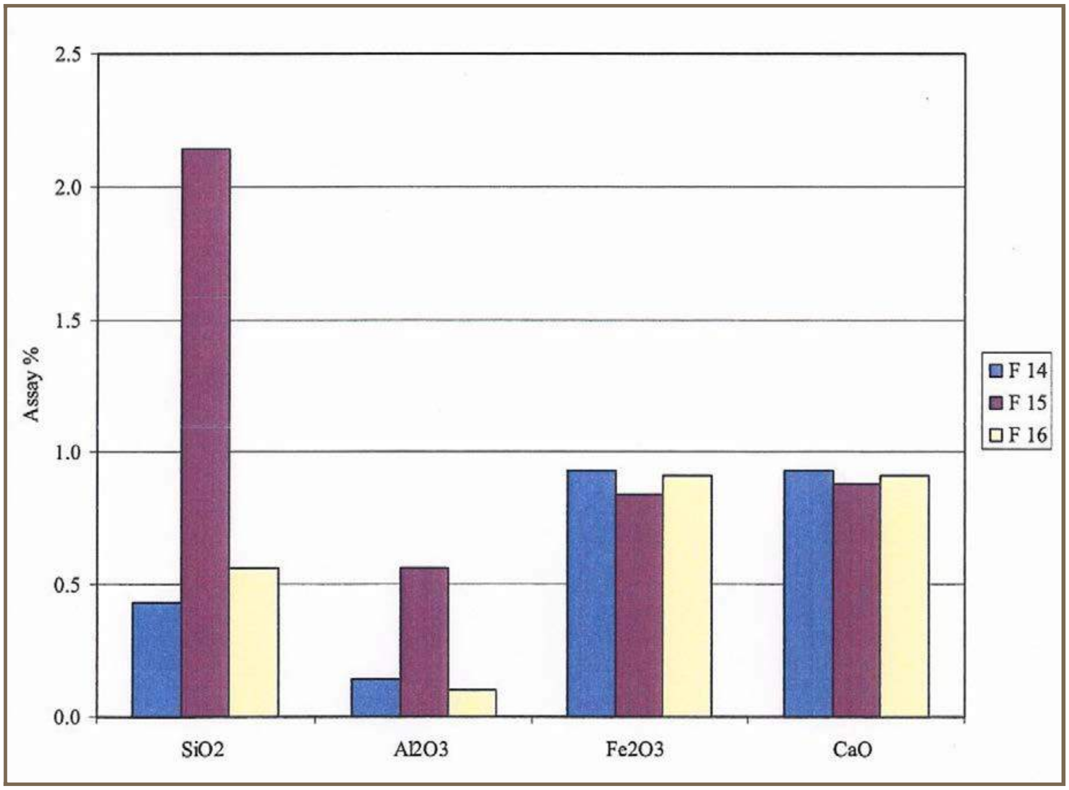


Figure 13-11: Impurities in Magnesite Concentrate, Direct Flotation, East Zone

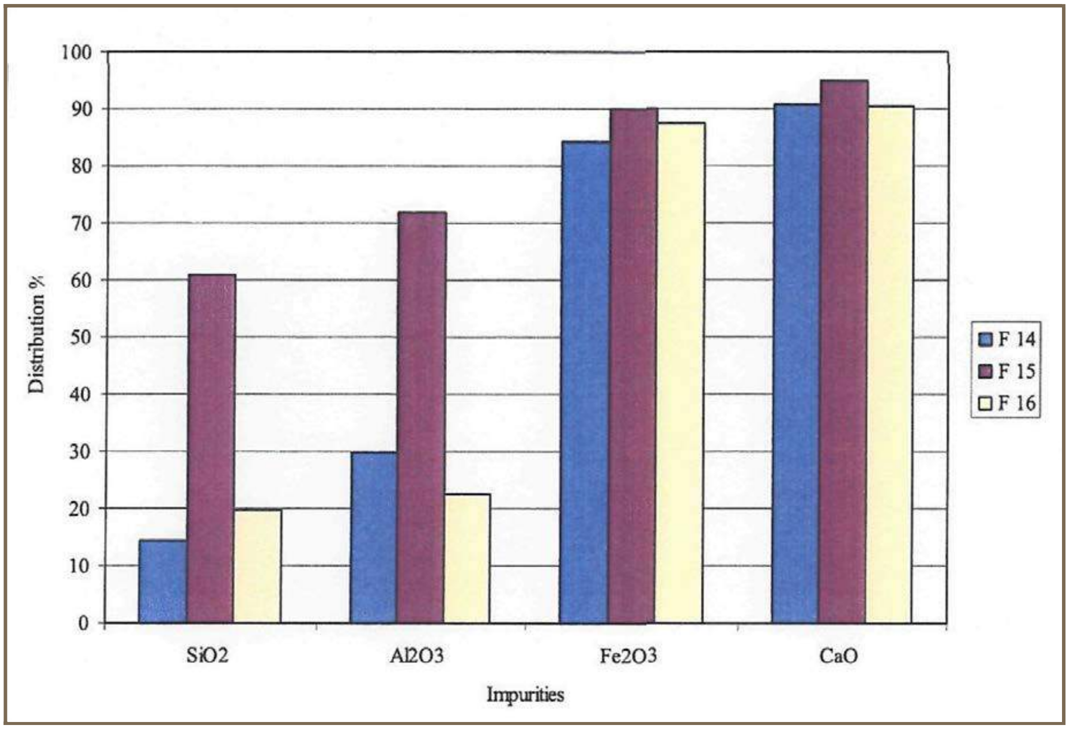


Figure 13-12: Distribution of Gangue Minerals in Magnesite Concentrate, Reverse Flotation, East Zone

Conceptual Flotation Flowsheet

Based on the results from the flotation tests, a conceptual flowsheet has been developed, shown in Figure 13-13. In this flowsheet, only flotation was used to upgrade the magnesite, and it consists of the following components:

- A pyrite flotation circuit; and
- Four rougher stages of silicate flotation.

The fourth stage silicate rougher flotation tailings is the final magnesite concentrate.

In the second phase of this Project, a more elaborate flowsheet should be developed in which HMS could be used in combination with flotation to achieve magnesite concentrate, as HMS is expected to reject a large portion of silicate and calcite gangue minerals.

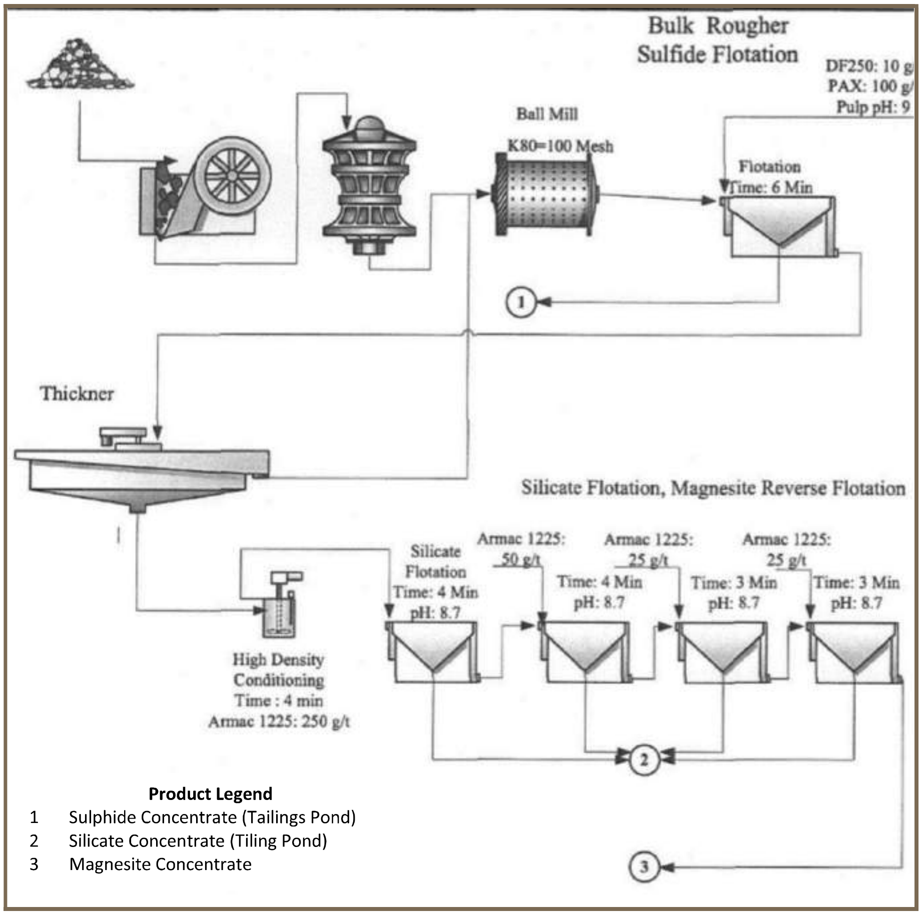


Figure 13-13: Flotation Circuit for Treatment of Magnesite Mineralization

13.2 Conclusions and Recommendations

13.2.1 SGS – Ahgamirian and Imeson, 2008

The preliminary test results indicate that high grades of magnesite concentrate can be produced by froth flotation alone, which would most likely be improved in combination with HMS. Iron contamination of the magnesite concentrate is expected to be mainly due to the presence of iron in magnesite crystal structure as solid solution, which remains to be confirmed through microprobe examination. Reverse flotation is the preferred method to generate magnesite concentrate. In this method, silicate minerals are first floated with Armac 1225 in several stages. The final tailings are considered the resultant concentrate. Pyrite flotation should be conducted, primarily to reduce the iron and sulphur content of the magnesite concentrate. Coarse primary grinding with a K_{80} of 150 µm is sufficient to achieve the required degree of liberation between magnesite and other gangue minerals. Desliming appears to have no positive effect on the quality of the magnesite concentrate or collector consumption reduction. This, however, remains to be confirmed in the second phase of the Project. Based on this scoping testwork, the following recommendations are made:

- Mineralogical studies should be carried out on the magnesite concentrate to determine the nature of the iron contaminant in the magnesite concentrate;
- More batch flotation tests should be conducted with the objective of optimizing the flotation schemes;
- The effect of desliming on the quality of the magnesite concentrate should be revisited;
- Heavy liquid separation on the coarser fractions should be tried; and
- The combination of HMS and flotation should be evaluated. Preconcentrate from HMS should be further processed by flotation or, if the grade is high enough, it may generate a marketable stream without further processing. Flotation will likely be a part of the process to beneficiate the fine fraction not suitable for HMS.

13.2.2 Samuel Engineering

Samuel recommends that further testwork be completed to validate the proposed flowsheet. At this time, only preliminary work has been conducted as outlined in this section. To advance the Project, extensive testwork must be performed examining all areas of the process. This would include, but is not limited to:

- Comminution data testing, including:
  - abrasion testing;
  - crushability index testing;
  - ball mill grindability testing; and
  - Bond impact testing.
- Additional flotation data as recommended by SGS (see Section 13.1.9).
- Thickening and filtration tests, including:
  - sample characterization;
  - flocculant screening;
  - static thickening;
  - dynamic thickening;
  - rheology;
  - pressure filtration; and
  - vacuum filtration.

Additional testwork around the calcination process should be addressed, including:

- Multi-hearth:
    - product quality (CCM grade);
    - product losses; and
    - fuel consumption.
- Pelletization:
    - binder requirements.
- Vertical shaft furnace:
    - product quality (DBM grade);
    - product losses;
    - fuel consumption; and
    - binder requirements.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

This Mineral Resource Statement for the MGX Minerals Inc. Project represents an updated Mineral Resource estimate prepared under the Canadian Securities Administrators' National Instrument 43 101 (NI 43-101) guidelines.

The Canadian Institute of Mining (CIM) Definition Standards on Mineral Resources and Reserves (CIM Definition Standards) establish definitions and guidance on the definitions for Mineral Resources, Mineral Reserves, and mining studies used in Canada. The Mineral Resource, Mineral Reserve, and Mining Study definitions are incorporated, by reference, into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101). The CIM Definition Standards, as adopted by CIM Council on May 10, 2014 states that:

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

The Mineral Resource has been prepared by Tuun based on 49 diamond drill holes, the 25 blast holes used for the bulk sample, and 45 magnesite surface samples. This resource estimation was completed by Allan Reeves, P.Geo., an independent QP as defined in NI 43-101. The effective date of the resource statement is December 31, 2016 and follows the guidelines of the generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines" (as adopted on November 23, 2003).

The resources discussed in this section are considered a reasonable representation of the Project at the current level of prospecting and sampling. The estimate follows the CIM Definition Standards for Mineral Resources and Mineral Reserves (as adopted by CIM Council on May 10, 2014).

Tuun also reviewed the 2003 "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" for Industrial Minerals (pp. 37–44). The QP recognizes that for industrial minerals such as the magnesium oxide product evaluated in this report, there is an *"inter-relationship that exists between: (i) markets, (ii) product evaluation, and (iii) product development"*. MGX has begun the necessary dialogue between themselves and potential buyers at this very early stage in the Project.

Tuun also reviewed the "Guidance on Commodity Pricing used in Resource Estimation and Reporting" adopted by the CIM Council on November 28, 2015. The guidance provides additional clarity on the CIM definition of *"reasonable prospects of eventual economic extraction."*

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. This Mineral Resource Estimate re-examined the existing data for the purposes of producing an updated Resource Estimate following NI 43-101 guidelines that form the basis of this PEA.

For resource estimation, Tuun utilized Geovia GEMS™ 6.7.1 software to model the solid defining the primary magnesite bed. The software was also used for basic statistics, geostatistics, variography, block modelling, estimating grades, and reporting of the resources.

14.2 Resource Database

The Project data were provided as spreadsheets, 2D cross-sections, and surface maps. Tuun audited the data and compiled it into one Access-compatible GEMS™ database. The drill hole dataset used for the estimate contains 74 drill holes as summarized in Table 14-1.

It had been suggested in earlier British Columbia Assessment Reports that the lower part of the Eastern Zone hit a zone of silica flooding attributed to possible thermal sources related to the emplacement/enrichment of the magnesite. Excess silica gangue would require removal.

The drill holes used in the resource estimate are sufficiently reliable to interpret with confidence the boundaries of the Hmn1B lithology hosting the magnesite deposit and estimate the percentages of the contained magnesium oxide (MgO), and the four possible contaminants to the recovery process: Al_2O_3, Fe_2O_3, CaO, and SiO_2.

In addition, Tuun reviewed the 51 rock samples provided, and found that the tenor of grade in the 45 magnesite (Hmn1B lithology) samples was comparable to that of the drill holes. The rock samples had been collected as chip samples in approximately 10 cm wide and 3 m long zones. The location and quality of the rock samples is deemed adequate for supplementing both the geologic interpretation and resource grade estimation without adding any detectable bias to the outcome. The samples are summarized in Table 14-2.

Table 14-1: Drill Holes used in the Resource Estimate

Hole-ID	East	North	Elevation (m)	Length (m)	Azimuth	Dip	Zone	Company	Type
DH1990-01	531,329.0	5,639,121.0	1,411.0	39.93	25.0	-80.0	East	Canoxy	NQ
DH1990-02	531,329.0	5,639,122.0	1,411.0	47.55	25.0	-55.0	East	Canoxy	NQ
DH1990-03	531,421.0	5,639,039.0	1,419.0	60.96	25.0	-45.0	East	Canoxy	NQ
DH1990-04	531,367.0	5,639,080.0	1,418.0	71.32	25.0	-45.0	East	Canoxy	NQ
DH2008-01	530,424.95	5,639,562.95	1,374.18	141.50	236.0	-46.0	West	Tusk	NQ
DH2008-02	530,474.38	5,639,521.33	1,377.49	133.50	210.0	-46.0	West	Tusk	NQ
DH2008-03	530,579.81	5,639,391.19	1,378.64	52.20	210.0	-44.0	West	Tusk	NQ
DH2008-04	530,613.29	5,639,465.78	1,393.60	82.70	215.0	-44.0	West	Tusk	NQ
DH2008-05	530,612.13	5,639,466.88	1,393.75	99.40	139.0	-49.0	West	Tusk	NQ
DH2008-06	530,555.63	5,639,497.00	1,386.91	100.00	210.0	-46.0	West	Tusk	NQ

Hole-ID	East	North	Elevation (m)	Length (m)	Azimuth	Dip	Zone	Company	Type
DH2008-07	530,477.00	5,639,524.00	1,384.00	82.70	215.0	-47.0	West	Tusk	NQ
DH2014-01	531,369.32	5,639,131.78	1,418.00	37.80	200.0	-52.0	East	MGX	BTW
DH2014-02	531,390.88	5,639,108.97	1,422.00	54.25	200.0	-52.0	East	MGX	BTW
DH2014-02A	531,390.88	5,639,108.97	1,422.00	39.62	0.0	-90.0	East	MGX	BTW
DH2014-03	531,423.69	5,639,098.22	1,426.00	65.53	200.0	-52.0	East	MGX	BTW
DH2014-04	531,458.68	5,639,071.05	1,430.00	74.20	200.0	-52.0	East	MGX	BTW
DH2014-05	531,493.78	5,639,049.52	1,433.00	71.63	200.0	-52.0	East	MGX	BTW
DH2014-06	531,553.37	5,639,034.40	1,435.00	36.58	200.0	-52.0	East	MGX	BTW
DH2014-07	531,414.02	5,639,079.45	1,424.00	57.91	0.0	-90.0	East	MGX	BTW
DH2015-01	530,457.52	5,639,477.48	1,384.11	121.92	205.0	65.0	West	MGX	BTW
DH2015-02	530,541.01	5,639,460.43	1,388.92	93.00	205.0	-53.0	West	MGX	BTW
DH2015-03	530,561.33	5,639,433.53	1,387.29	65.53	205.0	-30.0	West	MGX	BTW
DH2015-04	530,369.54	5,639,530.12	1,368.12	128.02	205.0	-53.0	West	MGX	BTW
DH2015-05	530,311.21	5,639,568.64	1,367.78	125.88	205.0	-53.0	West	MGX	BTW
DH2015-06	530,389.55	5,639,606.26	1,371.85	114.30	205.0	-50.0	West	MGX	BTW
DH2015-07	530,464.34	5,639,631.55	1,380.51	44.20	205.0	-53.0	West	MGX	BTW
DH2015-07A	530,474.13	5,639,627.68	1,381.04	18.29	25.0	-60.0	West	MGX	BTW
DH2015-08	530,355.78	5,639,648.01	1,375.52	108.20	205.0	-50.0	West	MGX	BTW
DH2015-09	530,685.67	5,639,465.59	1,378.52	79.86	205.0	-50.0	West	MGX	BTW
DH2015-10	530,726.86	5,639,441.24	1,378.84	43.28	215.0	-50.0	West	MGX	BTW
DH2015-11	530,606.90	5,639,570.42	1,380.00	16.76	205.0	-53.0	West	MGX	BTW
DH2015-11A	530,605.59	5,639,571.03	1,380.45	21.34	25.0	-60.0	West	MGX	BTW
DH2015-12	530,329.02	5,639,590.27	1,359.88	112.80	25.0	-60.0	West	MGX	BTW
B1410-01-01	531,414.55	5,639,072.37	1,424.54	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-02	531,413.37	5,639,073.93	1,423.88	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-03	531,411.91	5,639,075.69	1,423.72	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-04	531,410.04	5,639,078.46	1,423.59	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-05	531,408.66	5,639,080.39	1,423.43	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-06	531,416.55	5,639,073.89	1,424.27	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-07	531,415.52	5,639,075.37	1,424.16	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-08	531,414.37	5,639,077.16	1,424.01	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-09	531,412.59	5,639,079.58	1,423.81	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-10	531,410.70	5,639,081.54	1,423.44	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-11	531,417.98	5,639,075.30	1,424.32	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-12	531,417.06	5,639,076.66	1,424.15	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-13	531,415.74	5,639,078.28	1,424.07	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-14	531,414.03	5,639,080.53	1,423.81	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-15	531,411.57	5,639,082.89	1,423.70	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-16	531,419.35	5,639,076.79	1,425.04	12.19	0.0	-90.0	East	MGX	Perc

Hole-ID	East	North	Elevation (m)	Length (m)	Azimuth	Dip	Zone	Company	Type
B1410-01-17	531,418.46	5,639,078.13	1,424.87	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-18	531,417.36	5,639,080.24	1,424.83	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-19	531,415.63	5,639,082.41	1,424.67	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-20	531,413.50	5,639,084.21	1,424.28	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-21	531,420.70	5,639,077.88	1,425.49	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-22	531,419.92	5,639,079.54	1,425.28	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-23	531,418.93	5,639,082.00	1,425.10	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-24	531,417.39	5,639,083.64	1,425.26	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-25	531,415.35	5,639,086.15	1,424.82	9.14	0.0	-90.0	East	MGX	Perc
DH2016-01	531,421.92	5,639,031.93	1,417.85	39.0	205.0	-77.0	East	MGX	BTW
DH2016-02	531,528.93	5,638,975.09	1,425.80	94.0	25.0	-45.0	East	MGX	BTW
DH2016-03	531,564.35	5,638,976.45	1,430.58	97.00	25.0	-45.0	East	MGX	BTW
DH2016-04	531,478.60	5,639,018.90	1,423.56	86.50	25.0	-45.0	East	MGX	BTW
DH2016-05	531,409.37	5,639,066.46	1,421.32	46.00	0.0	-90.0	East	MGX	BTW
DH2016-06	531,382.31	5,639,092.63	1,418.04	49.00	0.0	-90.0	East	MGX	BTW
DH2016-07	531,356.29	5,639,114.48	1,412.93	49.00	0.0	-90.0	East	MGX	BTW
DH2016-08	531,327.90	5,639,127.73	1,404.39	31.00	0.0	-90.0	East	MGX	BTW
DH2016-09	531,424.03	5,639,176.08	1,376.72	46.00	205.0	-45.0	East	MGX	BTW
DH2016-10	530,439.44	5,639,546.06	1,376.99	64.00	0.0	-90.0	West	MGX	BTW
DH2016-11	530,364.86	5,639,590.10	1,375.06	65.00	0.0	-90.0	West	MGX	BTW
DH2016-12	530,523.55	5,639,487.93	1,386.55	75.00	0.0	-90.0	West	MGX	BTW
DH2016-13	530,509.46	5,639,513.12	1,387.29	184.00	205.0	-45.0	West	MGX	BTW
DH2016-14	530,557.04	5,639,366.05	1,378.40	128.00	25.0	-50.0	West	MGX	BTW
DH2016-15	530,296.97	5,639,617.07	1,361.67	76.00	25.0	-50.0	West	MGX	BTW
DH2016-16	530,288.57	5,639,529.86	1,374.69	82.00	0.0	-90.0	West	MGX	BTW

Notes: *m = metres; ID = identification*

Table 14-2: Rock Samples used in the Resource Estimate

Rock-ID	East	North	Elevation (m)	Length (m)	Azimuth	Dip	Zone	MgO%
501	531302.0	5639170.0	1,382.0	3.0	295.0	0.0	East	43.6
502	531305.0	5639170.0	1,383.0	3.0	295.0	0.0	East	43.8
503	531307.0	5639171.0	1,384.0	3.0	295.0	0.0	East	40.2
504	531309.0	5639171.0	1,385.0	3.0	295.0	0.0	East	44.6
505	531312.0	5639172.0	1,385.0	3.0	295.0	0.0	East	42.9
506	531314.0	5639173.0	1,385.0	3.0	295.0	0.0	East	40.7
801	530355.0	5639649.0	1,376.0	3.0	295.0	0.0	West	35.4
802	530355.0	5639652.0	1,375.7	3.0	295.0	0.0	West	32.9
803	530356.0	5639659.0	1,374.7	3.0	295.0	0.0	West	39.4

Rock-ID	East	North	Elevation (m)	Length (m)	Azimuth	Dip	Zone	MgO%
804	530356.0	5639662.0	1,374.2	3.0	295.0	0.0	West	34.0
805	530357.0	5639665.0	1,372.6	3.0	295.0	0.0	West	38.3
23291	531513.0	5639024.0	1,439.0	3.0	295.0	0.0	East	44.6
23292	531448.0	5639066.0	1,434.0	3.0	295.0	0.0	East	40.6
23293	530538.0	5639381.0	1,385.0	3.0	295.0	0.0	West	43.2
DR-15-01	530418.0	5639406.0	1,338.2	3.0	295.0	0.0	West	43.2
DR-15-02	530361.0	5639407.0	1,314.8	3.0	295.0	0.0	West	45.8
DR-15-03	530373.0	5639423.0	1,331.6	3.0	295.0	0.0	West	38.6
DR-15-04	530416.0	5639426.0	1,350.4	3.0	295.0	0.0	West	37.5
DR-15-05	530432.0	5639444.0	1,380.2	3.0	295.0	0.0	West	44.0
DR-15-06	530409.0	5639461.0	1,383.7	3.0	295.0	0.0	West	37.9
DR-15-07	530363.0	5639493.0	1,384.5	3.0	295.0	0.0	West	43.9
DR-15-08	530312.0	5639509.0	1,368.4	3.0	295.0	0.0	West	44.6
DR-15-09	530369.0	5639576.0	1,377.8	3.0	295.0	0.0	West	44.1
DR-15-10	530297.0	5639627.0	1,365.0	3.0	295.0	0.0	West	40.9
DR-15-11	530118.0	5639662.0	1,352.0	3.0	295.0	0.0	Fish	46.2
DR-15-12	530328.9	5639590.8	1,363.3	3.0	295.0	0.0	Fish	43.5
DR-15-13	530064.0	5639679.0	1,340.0	3.0	295.0	0.0	Fish	45.0
DR-15-14	530016.0	5639706.0	1,325.0	3.0	295.0	0.0	Fish	40.8
DR-15-15	529957.0	5639703.0	1,315.0	3.0	295.0	0.0	Fish	42.0
DR-15-16	529914.0	5639689.0	1,311.0	3.0	295.0	0.0	Fish	42.3
DR-15-17	529892.0	5639673.0	1,301.0	3.0	295.0	0.0	Fish	45.6
DR-15-18	529866.0	5639691.0	1,285.0	3.0	295.0	0.0	Fish	41.5
DR-15-19	531533.0	5638991.0	1,434.0	3.0	295.0	0.0	East	44.9
DR-15-20	531558.0	5638984.0	1,432.0	3.0	295.0	0.0	East	44.3
DR-15-21	531580.0	5638979.0	1,441.0	3.0	295.0	0.0	East	37.7
DR-15-22	531571.0	5639046.0	1,431.0	3.0	295.0	0.0	East	45.0
DR-15-23	531532.0	5639077.0	1,412.0	3.0	295.0	0.0	East	42.6
DR-15-24	531411.0	5639024.0	1,411.0	3.0	295.0	0.0	East	36.4
DR-15-25	530962.0	5639280.0	1,378.0	3.0	295.0	0.0	West	33.4
DR-15-26	530866.0	5639339.0	1,374.0	3.0	295.0	0.0	West	42.9
DR-15-27	530662.0	5639442.0	1,390.7	3.0	295.0	0.0	West	43.5
DR-15-28	530700.0	5639426.0	1,397.6	3.0	295.0	0.0	West	41.6
DR-15-29	530757.0	5639417.0	1,380.1	3.0	295.0	0.0	West	39.9
DR-15-30	530801.0	5639390.0	1,378.3	3.0	295.0	0.0	West	38.1
DR-15-31	530744.0	5639325.0	1,340.9	3.0	295.0	0.0	West	44.0

Notes: *m = metres; ID = identification*

14.3 Assay Data Evaluation

Various statistical tools were used to examine the characteristics of the full dataset. For example, basic or descriptive statistics were calculated with Excel to summarize all assays within the magnesite (Hmn1B) bed, as tabulated in Table 14-1 and Table 14-2, as a crosscheck to GEMS™ calculations.

GEMS™ software contains a comprehensive set of statistical tools to examine the characteristics of a dataset. In addition to basic or descriptive statistics, histograms and probability plots were used to explore further the data.

Table 14-3 shows the Excel statistics for the magnesite rock samples collected, while Table 14-4 summarizes the results for the drill samples. Overall, the mean MgO% grades of the two sample types are very similar, given the difference in the sample set sizes.

Table 14-3: Excel Statistics of Rock Magnesite Assays

Description	MgO %	Fe_2O_3 %	Al_2O_3 %	CaO %	SiO_2 %	LOI
Number	45	45	45	45	45	45
Minimum	32.9	0.56	0.04	0.15	0.54	38.21
Maximum	46.2	1.74	0.81	10.65	25.76	51.11
Mean	41.46	1.08	0.36	2.51	6.27	47.85
Median	42.60	0.97	0.34	0.93	4.35	68.64
Mode	44.60	0.77	0.04	0.55	n/a	47.85
Variance	12.00	0.12	0.07	9.16	29.53	7.71
St. Dev.	3.46	0.35	0.26	3.03	5.43	2.78
Skewness	-0.9	0.3	0.27	1.52	1.57	-1.46
Kurtosis	0.02	-1.34	-1.52	1.01	2.60	2.20

Notes: *% = percent; LOI = loss on ignition; St. Dev. = standard deviation*

Table 14-4: Excel Statistics of Drill Hole Magnesite Assays

Description	MgO %	Fe_2O_3 %	Al_2O_3 %	CaO %	SiO_2 %	LOI
Number	1,083	1,083	1,083	1,083	1,083	1,083
Minimum	10.66	0.04	0.14	0.17	0.4	7.92
Maximum	47.23	15.65	60.44	26	8.43	51.66
Mean	41.69	1.23	7.35	1.48	1.27	46.39
Median	42.90	1.01	5.26	0.72	1.31	47.79
Mode	43.60	0.79	5.06	0.38	1.56	48.14
Variance	18.95	1.14	40.68	7.99	0.21	31.10
St. Dev.	4.35	1.07	6.38	2.83	0.45	4.59
Skewness	-3.62	4.97	2.77	5.32	4.87	-3.03
Kurtosis	16.81	46.50	11.45	32.46	66.60	13.63

Notes: *% = percent; LOI = loss on ignition; St. Dev. = standard deviation*

It is apparent in Table 14-4 that a few drill hole assays are present that have low magnesia or very high contaminants. No corrections for possible rock-type miscoding of the original data were undertaken.

14.3.1 Combined Magnesite Assay GEMS™ Statistics

As noted, the primary product of interest is magnesium oxide (MgO) which is present in the magnesia-rich dolomite identified as Hmn1B or magnesite. The rock samples were combined with the drill hole samples and the GEMS™ descriptive statistics recalculated.

The GEMS™ statistics were generated by passing the drill holes through the interpreted magnesite wireframe and recoding the assays to Hmn1B (and rock code 110). Magnesite and the four primary contaminants (plus LOI), are summarized in Table 14-5.

Table 14-5: GEMS™ Statistics of Combined Database Magnesite Assays

Description	MgO %	Fe_2O_3 %	Al_2O_3 %	CaO %	SiO_2 %	LOI
Number	1,125	1,125	1,125	1,130	1,125	1,125
Minimum	10.66	0.40	0.04	0.00	0.01	7.92
Maximum	47.23	8.43	15.65	10.65	60.44	51.66
Mean	41.68	1.26	1.19	0.12	7.28	46.46
Median	42.90	1.31	0.98	0.00	5.21	47.86
Variance	18.69	1.20	1.12	0.63	5.38	20.64
St. Dev.	4.32	0.45	10.6	0.79	6.35	4.54
COV	0.10	0.36	0.89	6.36	0.87	0.10
Skewness	-3.56	4.76	4.93	9.50	2.74	-3.04
Kurtosis	19.50	68.21	49.25	103.42	14.24	16.83
97.5th percentile	45.40	1.93	3.75	0.98	25.07	50.71
99th percentile	45.75	2.29	4.99	3.63	34.15	51.11

Notes: *% = percent; LOI = loss on ignition; St. Dev. = standard deviation; COV = Coefficient of Variation*

14.3.2 Assay Histograms and Distribution Curves

The magnesite grade range is quite tight, between 40% and 47% MgO in the Hmn1B lithology (Figure 14-1), which typifies the Driftwood Creek magnesite deposit.

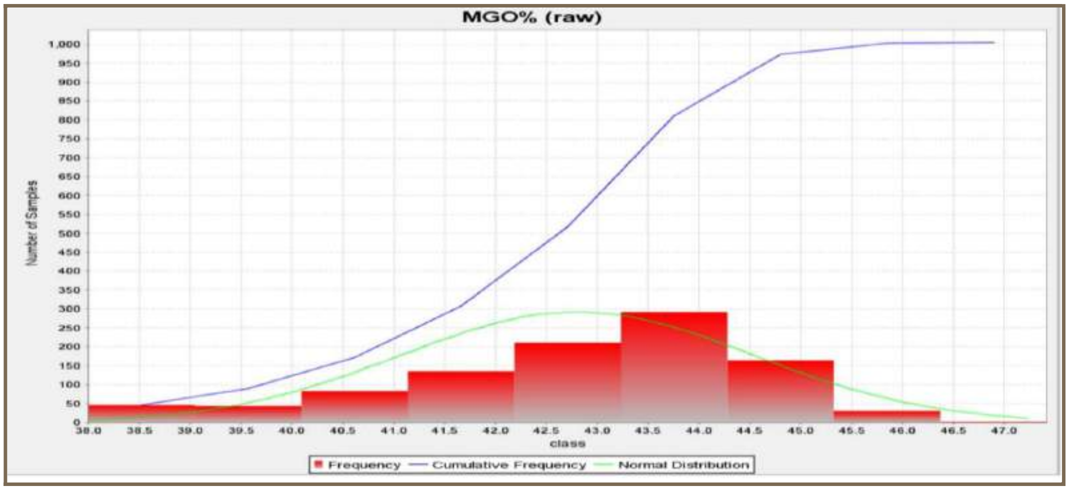


Figure 14-1: MgO% Assays – Hmn1B

Histograms of the four potential contaminants are shown in Figure 14-2 to Figure 14-5.

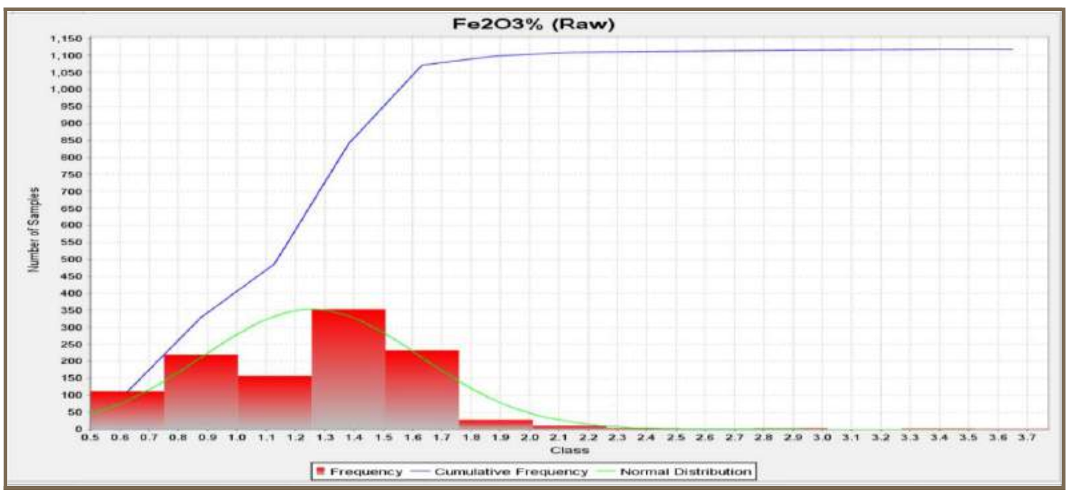


Figure 14-2: Fe_2O_3% Assays – Hmn1B

Fe_2O_3% shows two populations, which may be due to metal remobilization from the K5A/K5B intrusive rocks noted during core logging.

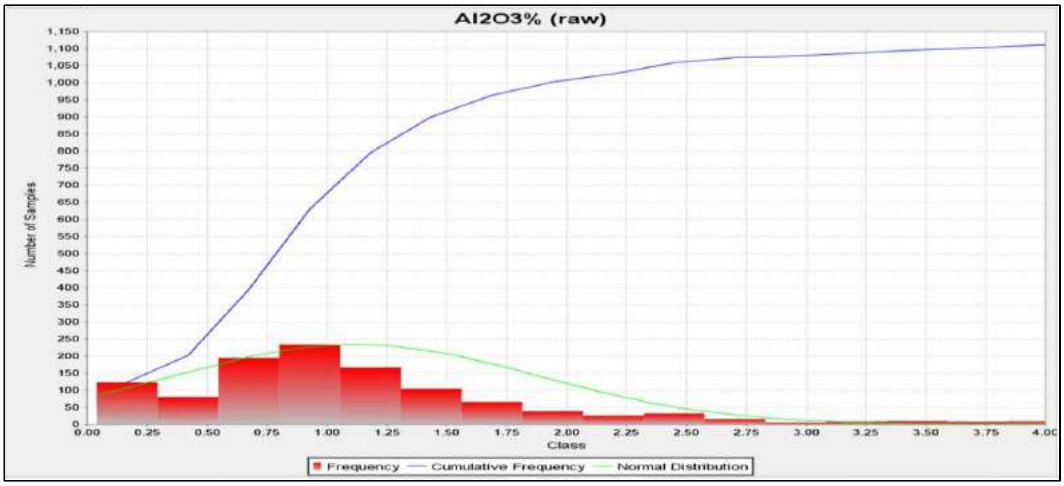


Figure 14-3: Al_2O_3% Assays – Hmn1B

The second population at the low end is likely an artifact of the detection limit.

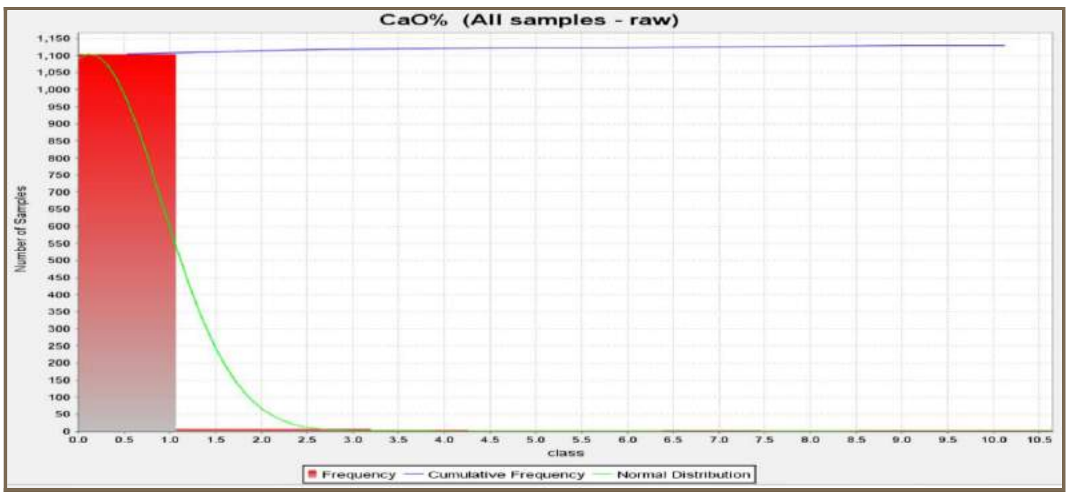


Figure 14-4: CaO% Assays – Hmn1B

These results show that the Driftwood Creek deposit contains very pure magnesite with almost no calcic component.

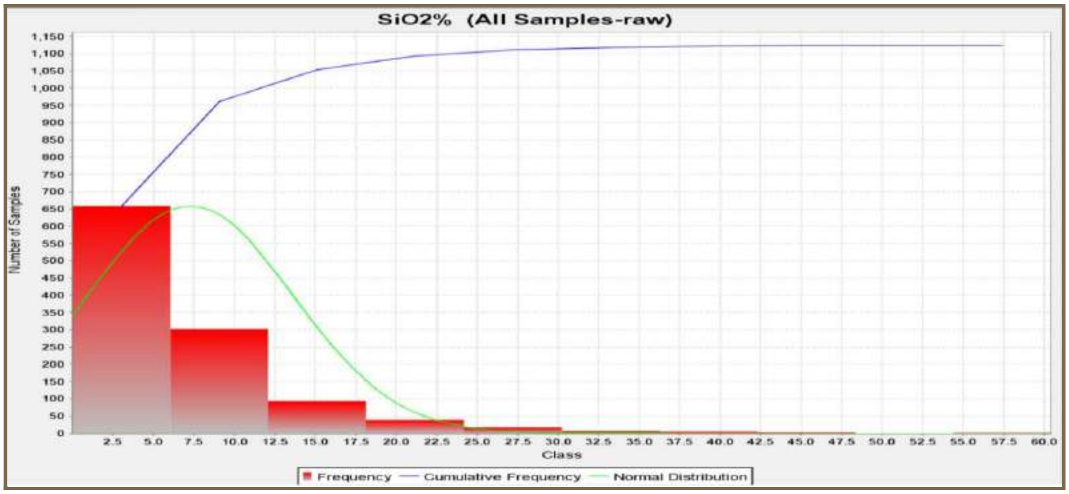


Figure 14-5: SiO_2% Assays – Hmn1B

14.3.3 Compositing

Sampling at the Project has varied between two primary lengths: 2 m and 3 m. During the 2008 drilling, the geologist favoured 2 m lengths, and this created shorter lengths at lithologic contacts. A later geologist increased the sample length to 3 m, and it has remained that way irrespective of lithology for the 2014–2016 diamond drilling campaigns. Outcrop rock-chip sample lengths were 3 m long by about 10 cm wide.

The author has chosen to standardize composite lengths to the weighted average length of ≈2.75 m within the corresponding lithology recorded. This is just over half of the selected block size of 5 m. Composites shorter than 1.4 m were not created during the process, and six miscoded lithologies were corrected. GEMS™ statistics for the composites show only minor changes in Table 14-6.

Table 14-6: Hmn1B Composite Statistics

Description	MgO %	Fe_2O_3 %	Al_2O_3 %	CaO %	SiO_2 %	LOI
Number	1,115	1,115	1,115	1,115	1,115	1,115
Minimum	13.19	0.50	0.04	0.15	0.01	21.57
Maximum	46.20	3.31	6.71	25.30	37.43	51.51
Mean	41.90	1.22	1.13	1.45	7.17	46.60
Median	42.70	1.28	0.99	0.78	5.55	47.72
Variance	12.87	0.12	0.54	6.49	28.49	46.42
Std. Dev.	3.59	0.34	0.73	2.55	5.34	14.50
COV	0.09	0.28	0.65	1.76	0.74	0.08
Skewness	-4.06	0.29	1.90	5.69	2.17	-2.52

Description	MgO %	Fe_2O_3 %	Al_2O_3 %	CaO %	SiO_2 %	LOI
Kurtosis	26.80	4.00	10.01	41.55	9.19	11.84
97.5th percentile	45.27	1.79	3.02	7.89	22.26	50.44
99th percentile	45.57	1.99	3.78	17.90	27.71	51.02

Notes: *% = percent; LOI = loss on ignition; St. Dev. = standard deviation; COV = Coefficient of Variation*

14.3.4 Capping of High Grades

The author considered four ways to treat the outliers, or high-grade samples:

- Apply a cap to the raw assay grade;
- Composite the assays and apply a cap;
- Composite the assays and do not cap; or
- Composite the assays and limit the range influence.

Of the four methods considered for outlier treatment, the author has elected to composite the assays and then cap high grades for:

- MgO% at 45.27% (the 97.5th percentile);
- Fe_2O_3% at the 99th percentile (or 1.99%) to accommodate the bi-modality and conservatively estimate the potential contamination;
- Al_2O_3% at the 99th percentile (or 3.78%);
- CaO% at the 99th percentile (or 17.90%);
- LOI% at the 99th percentile (or 51.02%); and
- SiO_2% at the 99th percentile (or 27.71%).

The use of the 99th percentile caps for the potential contaminants is considered conservative, in that the impurities to be removed should be slightly overestimated, compared to the primary target MgO%.

14.4 Surfaces and Solids

One-metre Light imaging, Detection, and Ranging (LiDAR) topographic contours were made available to the author for this report. The data had been provided by a local logging company and the surface created in GEMS™ is shown in Figure 14-6 complete with all drill holes.

MGX did not have any three-dimensional geologic interpretations, so Tuun created wireframes for both the East and West Zones and six crosscutting faults. The solids were originally based on 2D paper-based geologic interpretations and outcrop mapping provided by Andris Kikauka, P.Geo. They were modified after the 2016 infill drilling was completed, and are shown in Figure 14-7.

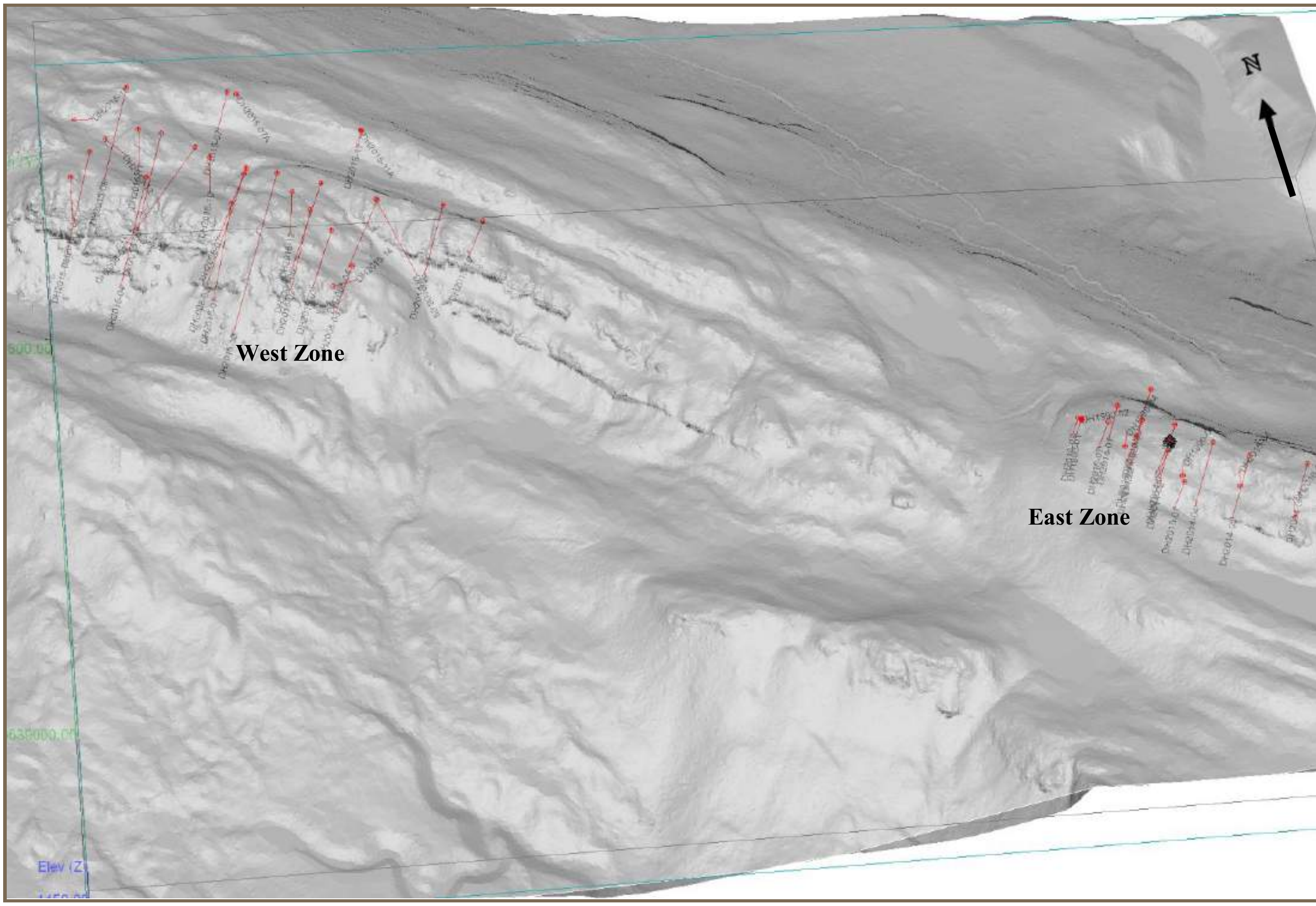


Figure 14-6: Topography and Drill Holes

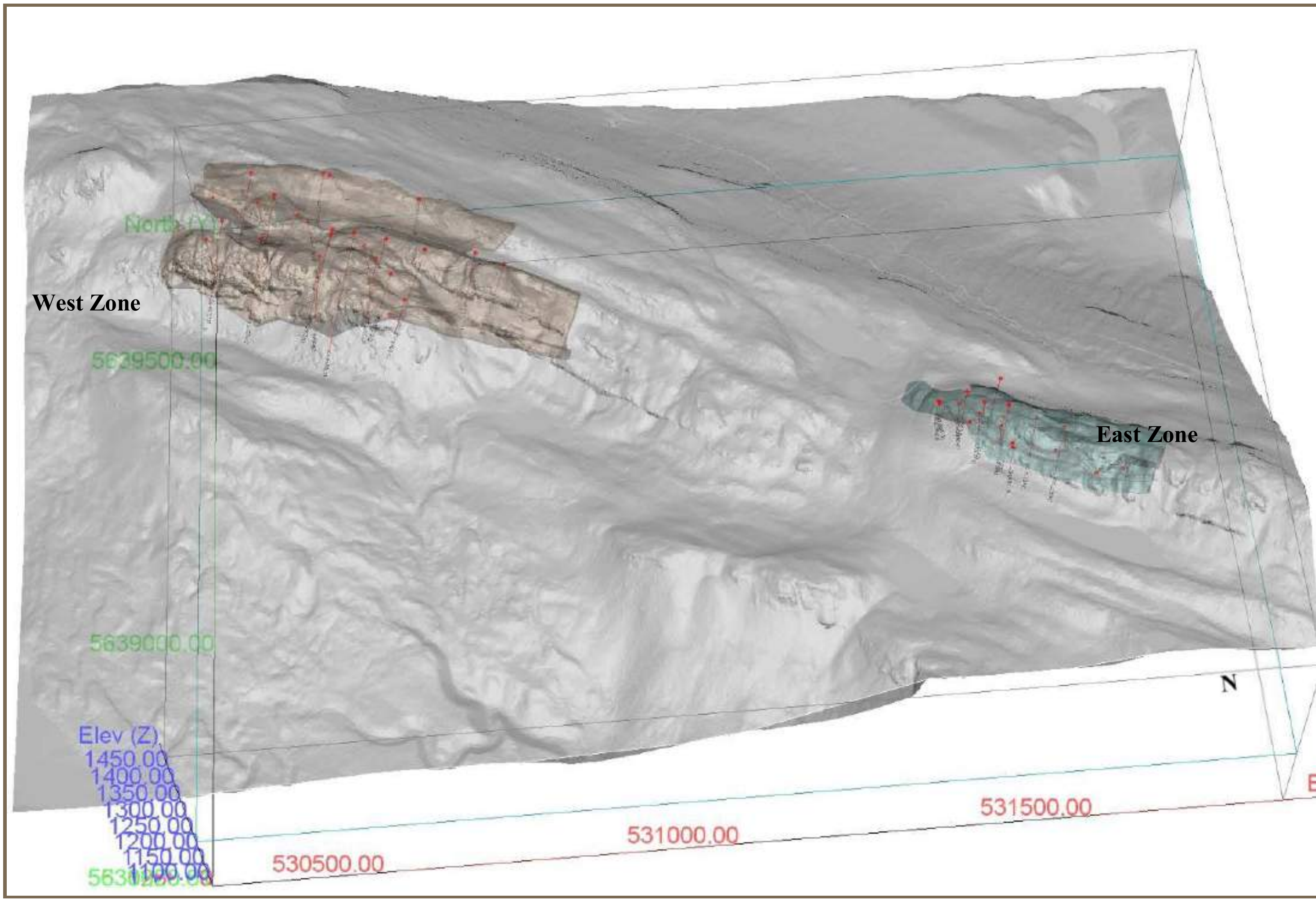


Figure 14-7: Magnesite and Topography Wireframes

14.5 Specific Gravity Estimation

Specific gravity (SG) determinations of select 2014–2015 drill core samples were done by the weight-in-air/weight-in-water method. The laboratory opted to do the weighing after first pulverizing the core, resulting in their final values being questionable (≈8% lower than expected). Also, the laboratory would not warrant their work as NI 43-101 compliant, and so the results have been disregarded in this analysis.

Two grab samples taken by the author at the bulk sample site returned very similar SG results (≈2.95) as the best previous data from the 2008 SGS metallurgical report. These samples were 1 kg to 2 kg lumps and the method chosen was the wax-coated weight-in-air/weight-in-water method.

MGX personnel also collected sixteen 25-L pails of rock samples from the magnesite stockpile created at the site. The samples were collected on a grid pattern over the storage area, and tested by the weight-in-air/weight-in-water method. These tests are considered a reasonable estimation of the SG for the East Zone. The magnesite SG varied from a minimum of 2.89 to a maximum of 3.02, with an average of 2.95.

In the 2016 infill drill campaign, samples were taken for SG analysis (methodology in Appendix A). A total of 18 samples were taken, 15 from the Hmn1B lithology. The average of the 15 magnesite samples was also 2.95, confirming earlier testing. Both, East Zone and West Zone SGs are similar (see Table 14-7 for SG data by lithology).

Table 14-7: Specific Gravity by Lithology

Identifier	Lithology	Specific Gravity	Comment
Hmn1A	Dolomite with some strong atolites	2.83	2 samples
Hmn1B	**Magnesite**	**2.95**	**33 samples**
Hmn2	Argillaceous dolomite	2.83	1 sample
Hmn3	Cherty dolomite	2.70	Supplied
Hmn4	Quartzite	2.60	Supplied
Int =	Grey-black, fine grained intrusive	2.60	Supplied
K 5A =	Dike-Sill, intermediate composition	2.60	Supplied
K 5B =	Dyke-Sill, Mafic composition	2.60	Supplied

The supplied lithologies do not have measured SG data just assumptions supplied by Kikauka, but are considered irrelevant as they are either distal to the magnesite or very thin intercepts (dykes). The primary background waste rock types expected to be encountered during mining are Hmn1A and Hmn2.

It is recommended that additional SG sampling be conducted on the adjacent Hmn1A and Hmn2 lithologies in future infill and geotechnical drilling programs.

14.6 Geostatistical Analysis and Variography

Mineral deposits often have spatial variability that tends to be strongest in one direction. This is termed anisotropy, and samples in this direction have lower variability than samples in other directions. A semi-variogram is a graph used to show this variability.

The horizontal axis of the semi-variogram shows the distance between pairs of samples being compared while the vertical axis shows the variability (half of the variance) of the samples at specific distances (lag intervals).

The semi-variogram model consists of four key parts: the nugget, sill, range, and model type. The nugget (C_0) describes the variability at very short distances and could be a result of emplacement processes, differences in the sampling and assaying techniques, or perhaps contamination. The sill is the point at which the curve approaches a constant value, and the distance to the sill is called the range. The model types that can be used to fit the data are commonly the spherical, exponential, and Gaussian models.

Spatial continuity of all four minerals was evaluated with normalized variograms using Geovia GEMS™ Version 6.7.1 software. The anisotropy was assessed using azimuth, dip, and azimuth (ADA) rotation.

Sixteen directional variograms at 22.5° increments were created at a -15° plunge with the primary dip direction being near vertical in an east–west direction as indicated by geologic interpretations.

A downhole linear semi-variogram was also created to crosscheck the z-range. Nested spherical models were fitted as summarized in Table 14-8. Geostatistical analysis of the MgO%; Fe_2O_3%; Al_2O_3%; CaO%; and SiO_2% composites produced reasonable semi-variograms (Appendix B).

Table 14-8: Semi-Variogram Parameters

Variable	Model	Azimuth	Dip	Azimuth	Co	Cl	C2	X (m)	Y (m)	Z (m)
MgO %	Sph	152.1	-20.4	1.1	1.07	0.79	-	43.2	38.8	23.7
	Sph	152.1	-20.4	1.1	-	-	0.40	217.6	195.5	119.5
Fe_2O_3 %	Sph	322.8	-19.4	175.1	0.01	0.02	-	17.5	13.2	5.1
	Sph	322.8	-19.4	175.1	-	-	0.04	198.4	149.9	58.1
Al_2O_3 %	Sph	285.8	6.9	20.4	0.11	0.29	-	54.6	26.7	27.6
	Sph	285.8	6.9	20.4	-	-	0.15	250	122	126.4
CaO %	Sph	285.4	7.2	45.3	0.0	4.50		95.2	76.2	27.5
	Sph	285.4	7.2	45.3	-	-	0.50	259.2	207.5	74.9
SiO_2 %	Sph	256.9	24.9	9.5	2.27	21.4	-	65.6	45.9	21.4
	Sph	256.9	24.9	9.5	-	-	2.26	159.5	111.7	159.5
LOI	Sph	271.7	-18.5	170.1	0.38	7.97	-	64.2	28.4	49
	Sph	271.7	-18.5	170.1	-	-	6.1	274.7	121.8	209.6

Notes: *m = metres; % = percent; LOI = loss on ignition*

Comparisons of the estimated blocks used three techniques: the nearest neighbour (NN), inverse distance squared (IDS or ID2), and ordinary kriging (OK). They were undertaken to ensure that the estimation output respected nearby composites and overall trends in the bed. During the validation stage, it was determined that the IDS methodology was the most representative of the three.

14.7 Block Model Definition

Personal experience, discussions with MGX representatives, and mining colleagues suggested that a 5 m block size is a reasonable approximation of a selective mining unit (SMU) for a small truck and excavator quarrying fleet. It has therefore been assumed that the quarry gear might be similar to the Cat 336D excavator and Cat D300 articulated truck used for the collection of the bulk sample.

The block model is in Universal Transverse Mercator (UTM) coordinates NAD 1983 11N, and the block model origin coordinates, block size, and rotation are summarized in Table 14-9.

Table 14-9: Block Model Definition

Origin	Block Size	No. of Blocks
530,000 E	5	320
5,659,300 N	5	152
1480 Elev. (Max.)	5	80
Rotation	-25	-

Notes: *Elev. = elevation; Max = maximum; No. = number*

14.8 Grade Estimation

Block model grades were estimated in three passes using IDS with the minimum and maximum 2.75 m capped composite samples and searches as summarized in Table 14-10. The methodology used was that blocks meeting the tabulated criteria would be classified after:

- Pass 1 would be Measured;
- Pass 2 – Indicated; and
- Pass 3 – Inferred.

Unfilled blocks would retain the block model initialization value of zero (0), and would be used only as a guide to determining any targets for future exploration (TFFE).

Table 14-10: Search Ellipse Parameters

Elliptical Search		Orientation Angle			Search Dimensions			No. of Composites		Maximum Comp./Hole
Variable	Pass	Azimuth	Dip	Azimuth	X (m)	Y (m)	Z (m)	Minimum	Maximum	
MgO %	P1	152.1	-20.4	11	50	50	50	7	18	3
	P2	152.1	-20.4	11	125	100	125	5	18	2
	P3	152.1	-20.4	11	200	150	200	1	18	2
Fe_2O_3 %	P1	322.8	-19.4	175.1	17.5	13.2	5.1	7	18	3
	P2	322.8	-19.4	175.1	99.2	75	29	5	18	2
	P3	322.8	-19.4	175.1	198.4	149.9	58.1	1	18	2
Al_2O_3 %	P1	285.8	6.9	20.4	55	27	28	7	18	3
	P2	285.8	6.9	20.4	125	61	63	5	18	2
	P3	285.8	6.9	20.4	187.5	91.5	94.8	1	18	2
CaO %	P1	285.4	7.2	45.3	95.2	76	28	7	18	3
	P2	285.4	7.2	45.3	130	104	37	5	18	2
	P3	285.4	7.2	45.3	194	155.5	56	1	18	2
SiO_2 %	P1	256.9	24.9	9.5	39.9	27.9	39.9	7	18	3
	P2	256.9	24.9	9.5	79.9	55.9	79.7	5	18	2
	P3	256.9	24.9	9.5	119.6	83.8	119.6	1	18	2
LOI	P1	271.7	-18.5	170.1	68.7	30.4	52.4	7	18	3
	P2	271.7	-18.5	170.1	137.4	60.9	104.8	5	18	2
	P3	271.7	-18.5	170.1	206	91.3	157.2	1	18	2

Notes: *No. = number; % = percent; Comp. = composites; LOI = loss on ignition; m = metres*

14.9 Model Validation and Sensitivity

The grade models were visually validated by comparing the blocks estimated with actual drill hole composite data on both section and plan views. Table 14-11 shows the colour key used for figures in this section.

Table 14-11: Magnesium Oxide (MgO%) Colour Legend

>= Lower Bound	< Upper Bound	Colour
38.00000	42.00000	RGB 192 192
42.00000	42.50000	RGB 0 0 255
42.50000	43.00000	RGB 0 255 0
43.00000	45.00000	RGB 255 0 0
45.00000	100.00000	RGB 139 0 0

Notes: *< = less than; > = more than*

Figure 14-8 and Figure 14-9 are section and plan views, respectively, for the West Zone. Figure 14-10 and Figure 14-11 are section and plan views for the East Zone. Overall, composite grades are a good match to the estimated block grades.

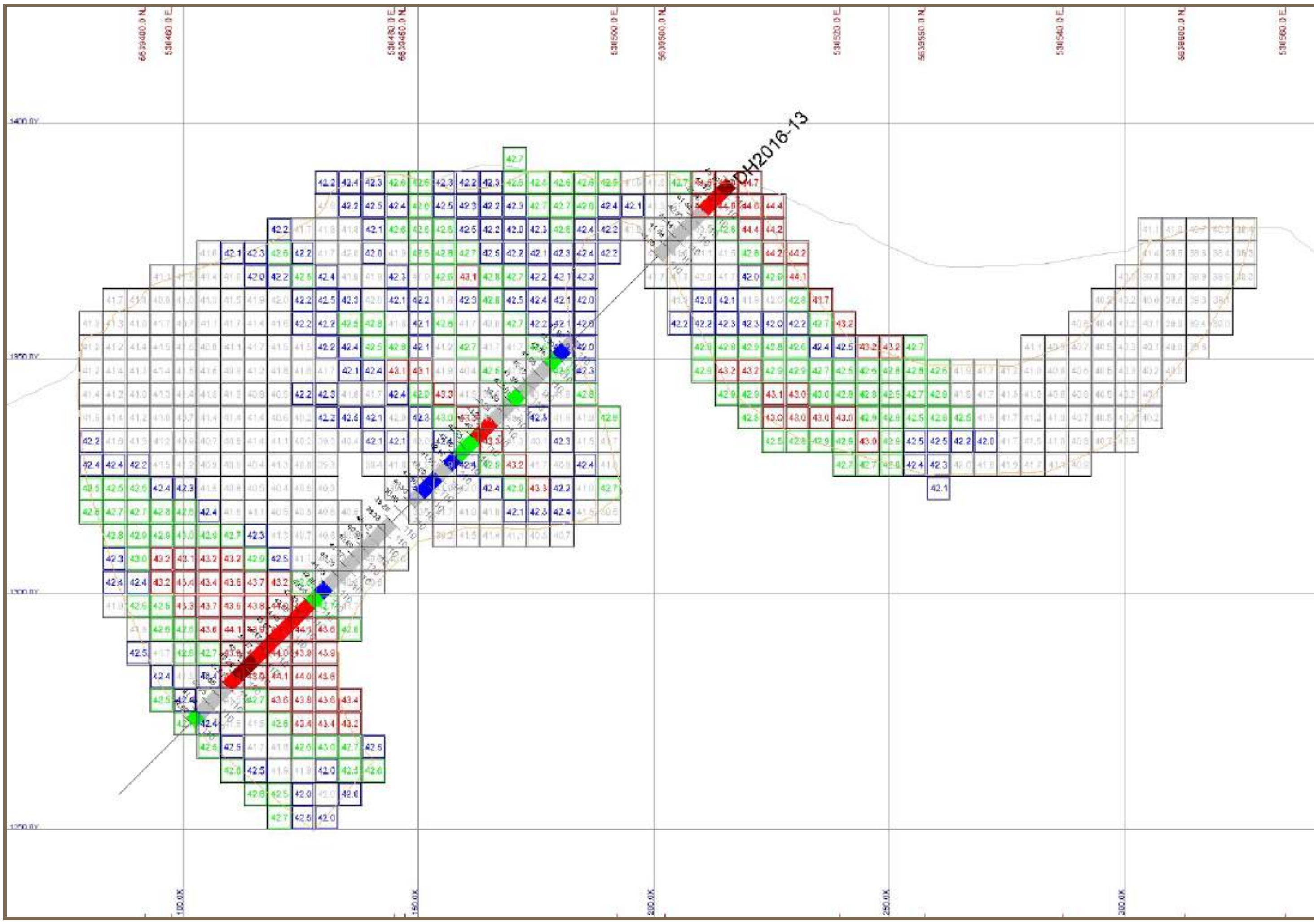


Figure 14-8: View along Section 1325E – West Zone MgO% Grades

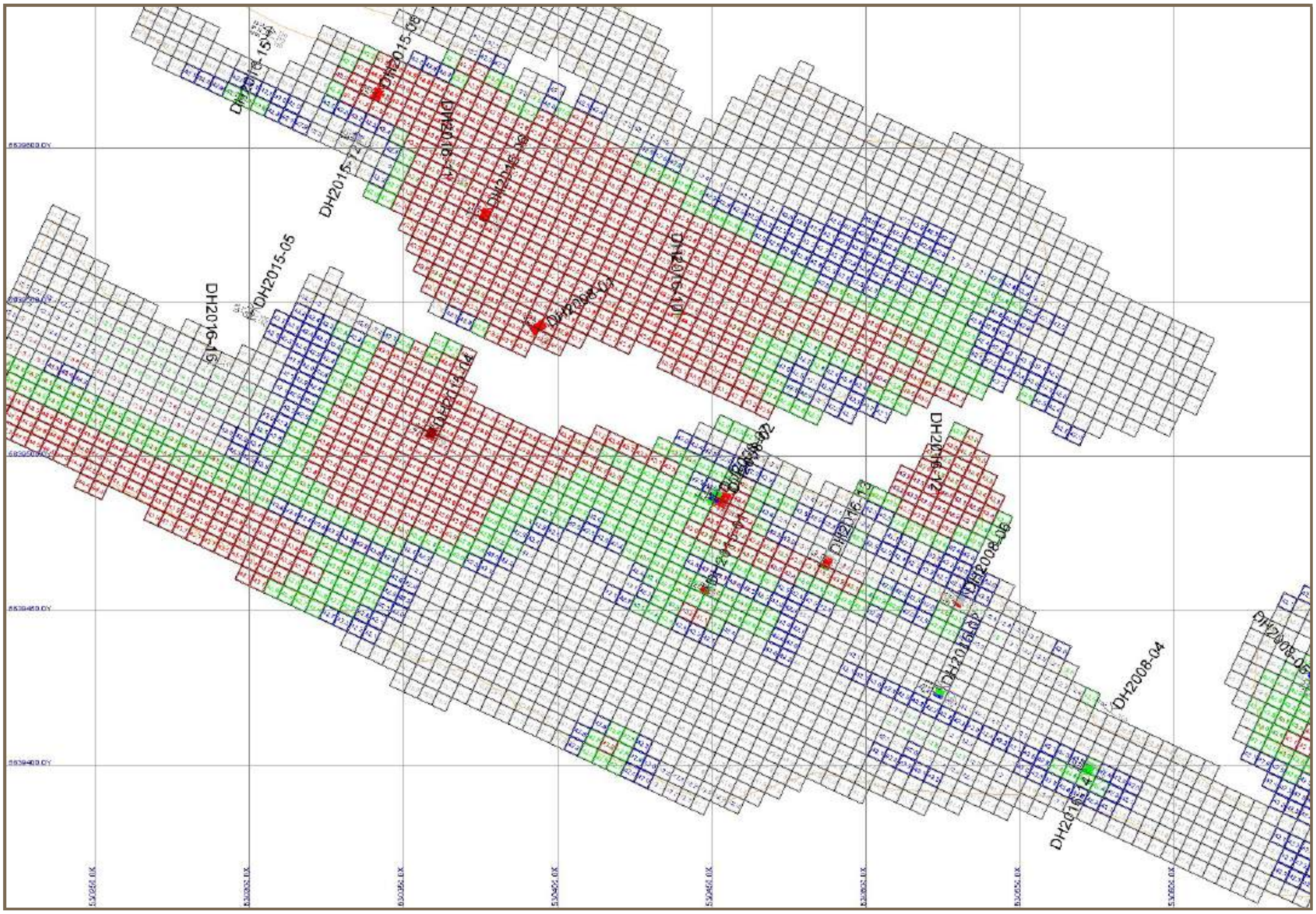


Figure 14-9: Plan View Level 1330 – West Zone MgO% Grades

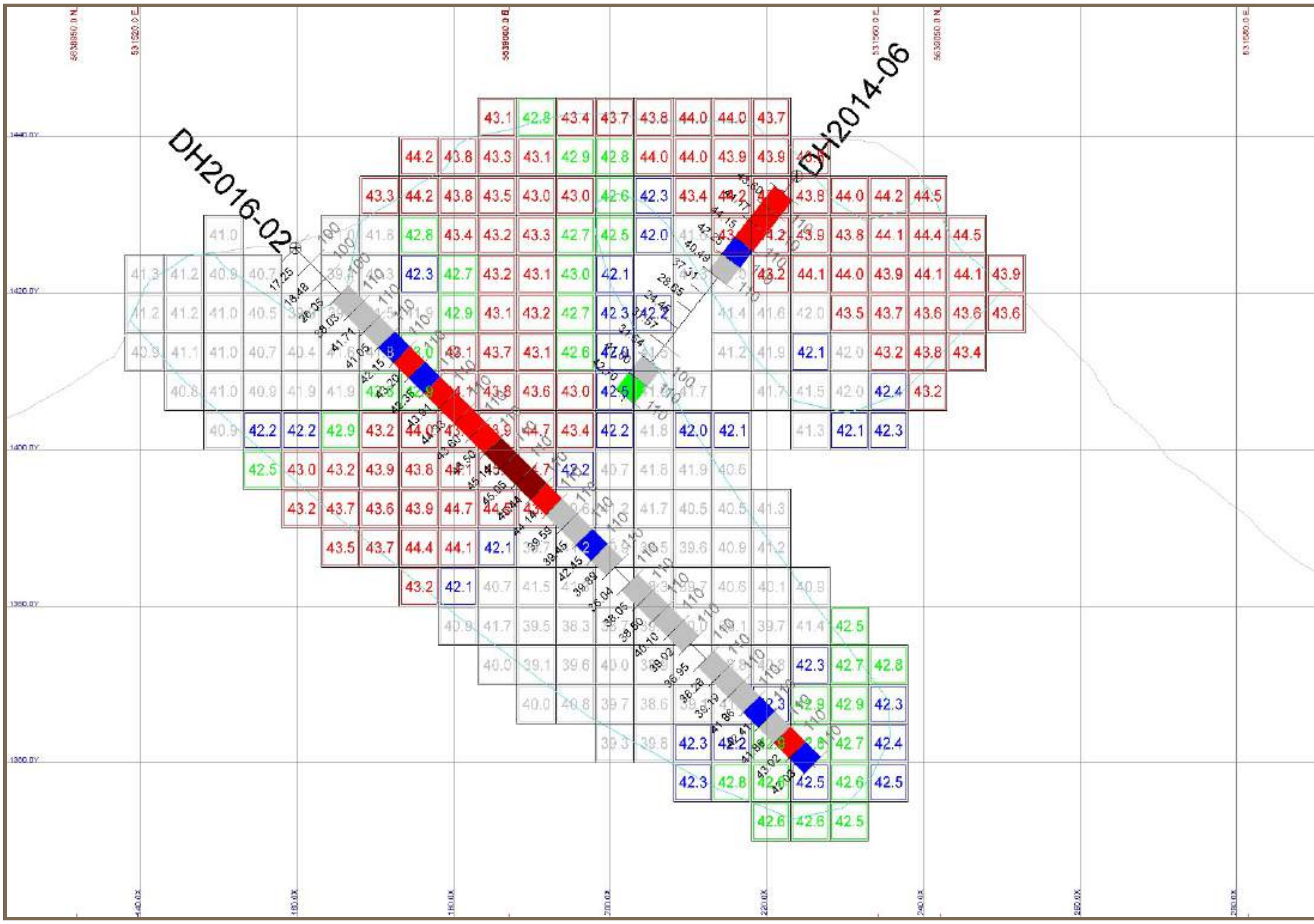


Figure 14-10: View along Section 2475E – East Zone MgO% Grades

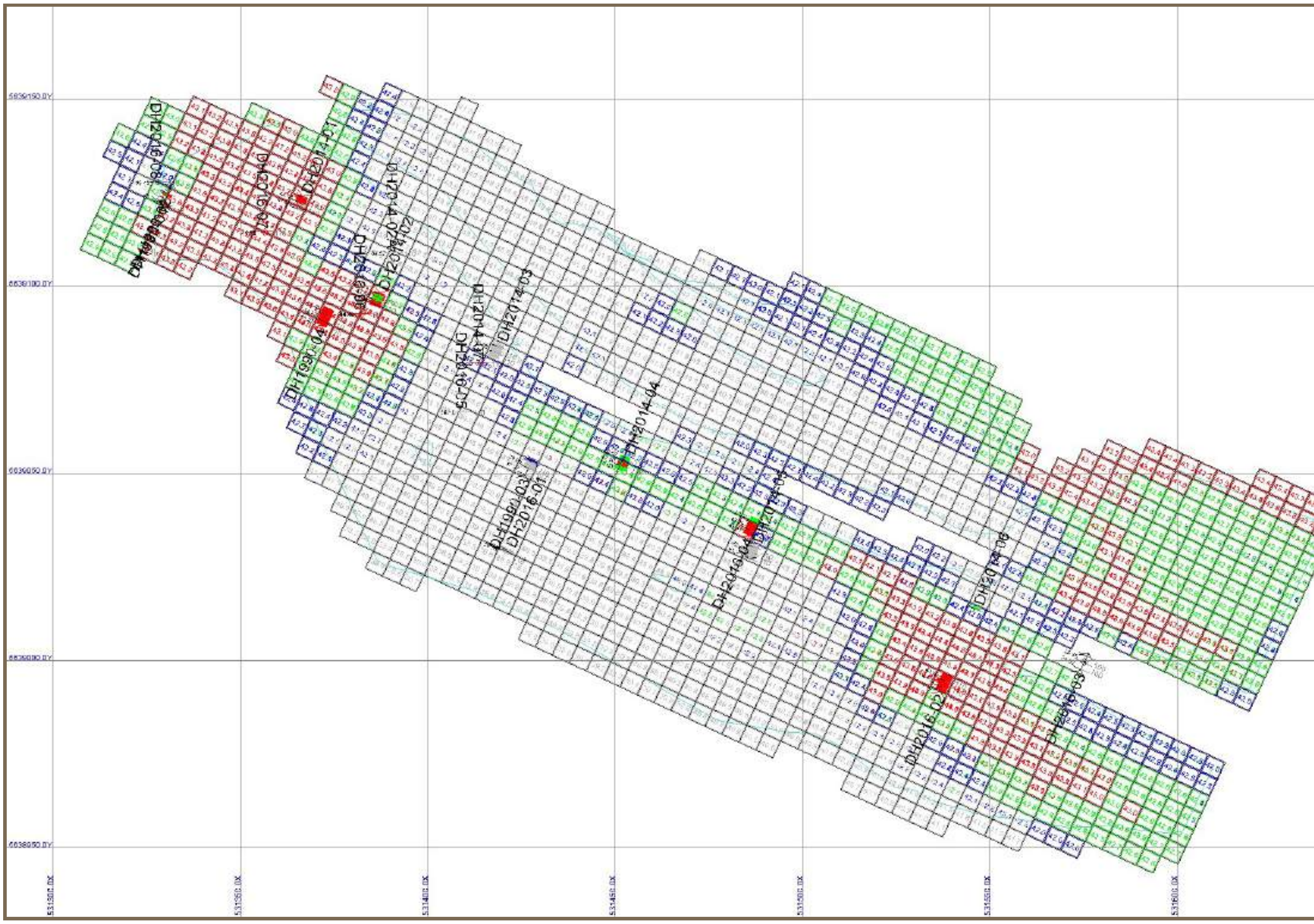


Figure 14-11: Plan View Level 1400 – East Zone MgO% Grades

As noted, NN and OK models were generated for comparison to the IDS model. Figure 14-2 shows the similarity of the three estimates, which is not unexpected given the consistency shown by the whole rock analyses.

The NN model represents an unbiased estimate, but visually on both plan and section views has some very high-grade zones. Conversely, the OK method had the reverse effect on visual inspection, with some zones of unusually low grades, probably caused by over-smoothing of the data.

Tuun believes that, overall, the IDS method was appropriate for the resource estimation.

Table 14-12: MgO Resource Estimate Comparisons

Method	Tonnage (Mt)	MgO %	Al_2O_3 %	SIO_2 %	Fe_2O_3 %	CaO %	LOI %
NN	19.30	41.46	1.00	7.21	1.32	1.18	46.51
IDS	19.30	41.47	1.00	7.13	1.32	1.13	46.55
OK	19.30	41.68	0.98	6.98	1.32	1.73	46.56

Notes: *% = percent; LOI = loss on ignition; Mt = million tonnes; NN = nearest neighbour; IDS = inverse distance square; OK = ordinary kriging*

Tuun also created quantile-quantile (Q-Q) plots of the OK model estimates versus the well-informed block composite grades as a cross-check. A well-informed block uses the arithmetic mean of all the samples within the block as the block estimate. They have not been weighted as they would be using the IDS or OK methods.

In the magnesite deposit (Figure 14-12); the block estimate by the mean of the composites is very similar to the block IDS estimate. While very close to a 1:1 correlation, overall the Q-Q plot shows that the estimate supports the visual inspection of the blocks presented in the previous section.

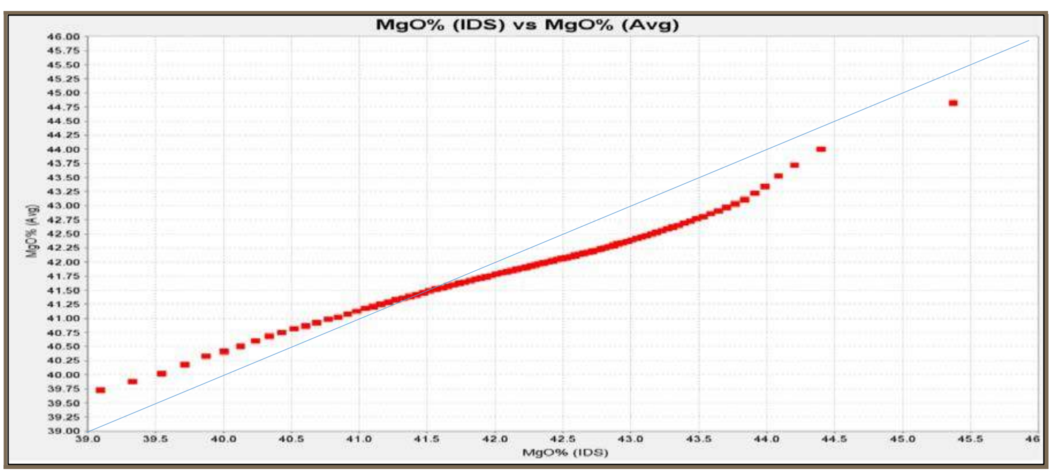


Figure 14-12: Q-Q Plot of MgO (IDS) vs. MgO (Mean) Grades

The plot shows that for the magnesite bed (40% to 45%) the estimation is a reasonable correlation.

Figure 14-13 through Figure 14-16 show the Q-Q plot comparisons of the potential impurities.

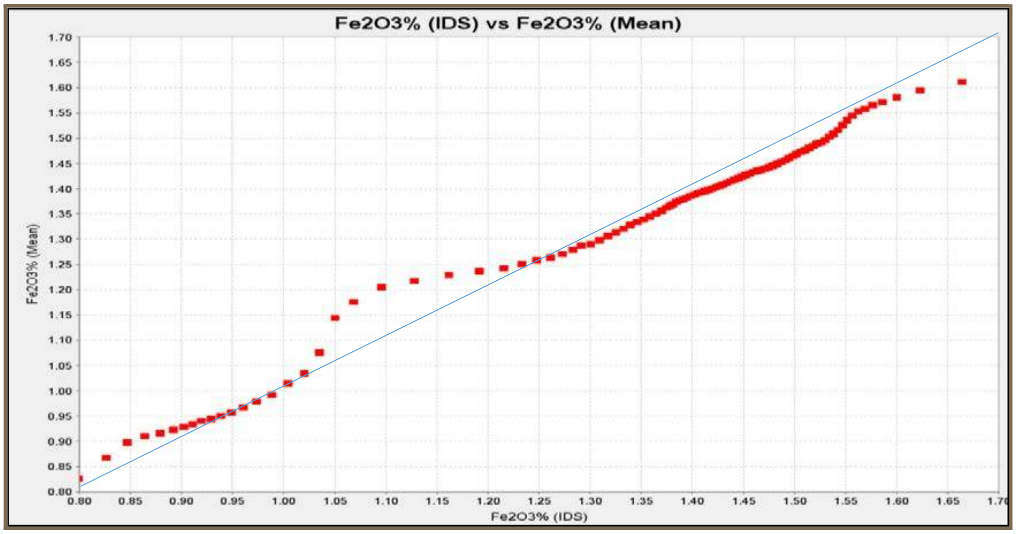


Figure 14-13: Q-Q Plot of Fe_2O_3 (IDS Estimate) vs. Fe_2O_3 (Mean) Grades

There is a minor bi-modality in the composites that may be related to alteration from the intrusive Cretaceous age granite (Kg). It is unclear if this impurity could be avoided by selective mining. Further work is warranted to determine the source.

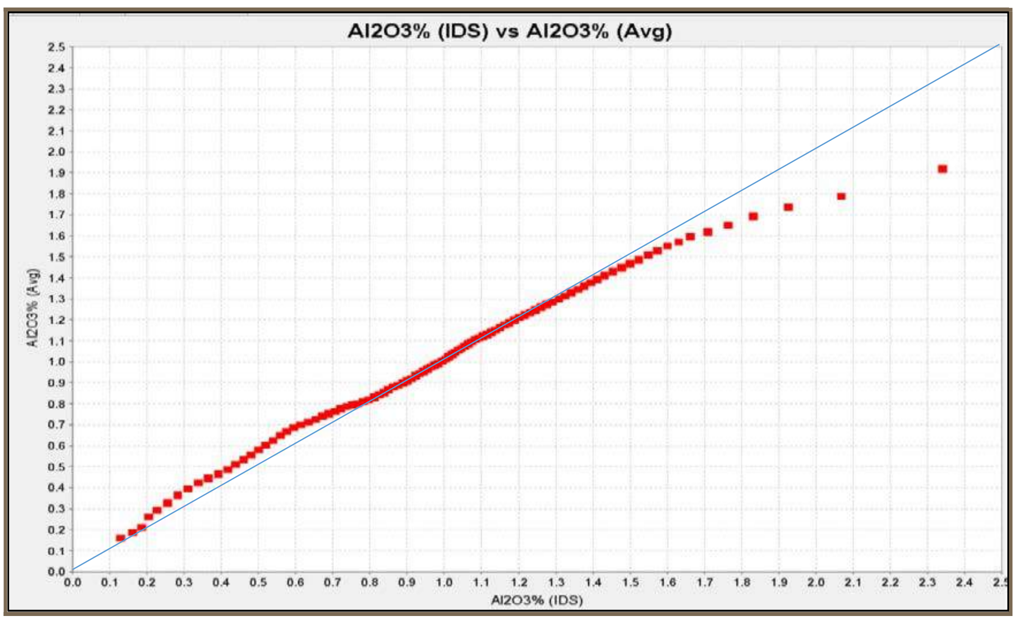
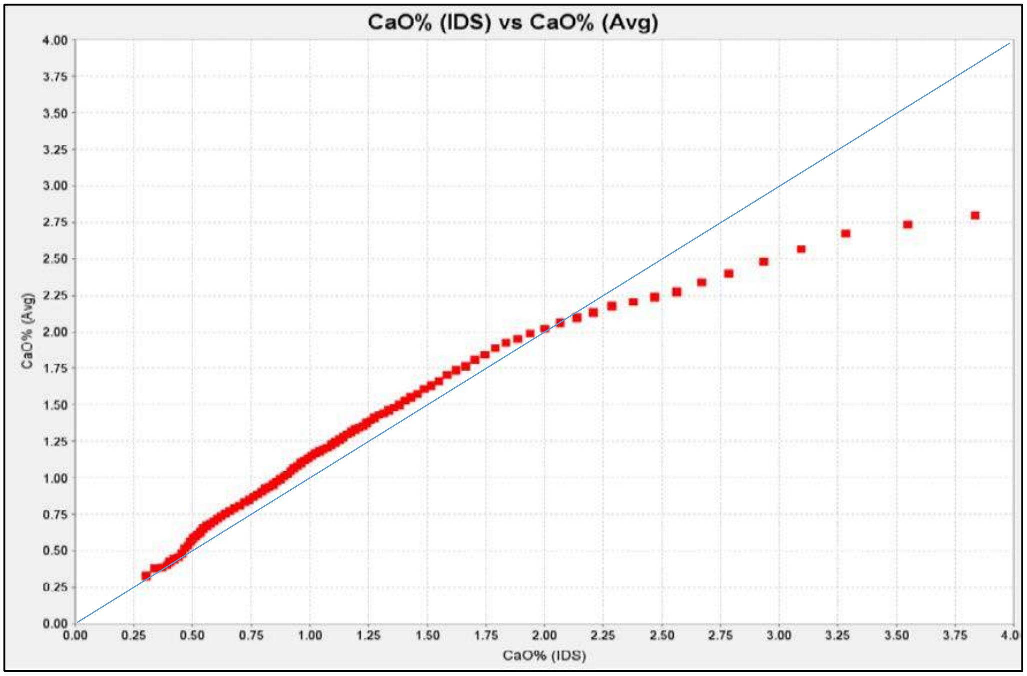


Figure 14-14: Q-Q Plot of Al_2O_3 (IDS Estimate) vs. Al_2O_3 (Mean) Grades

There is a very good correlation between the estimates.



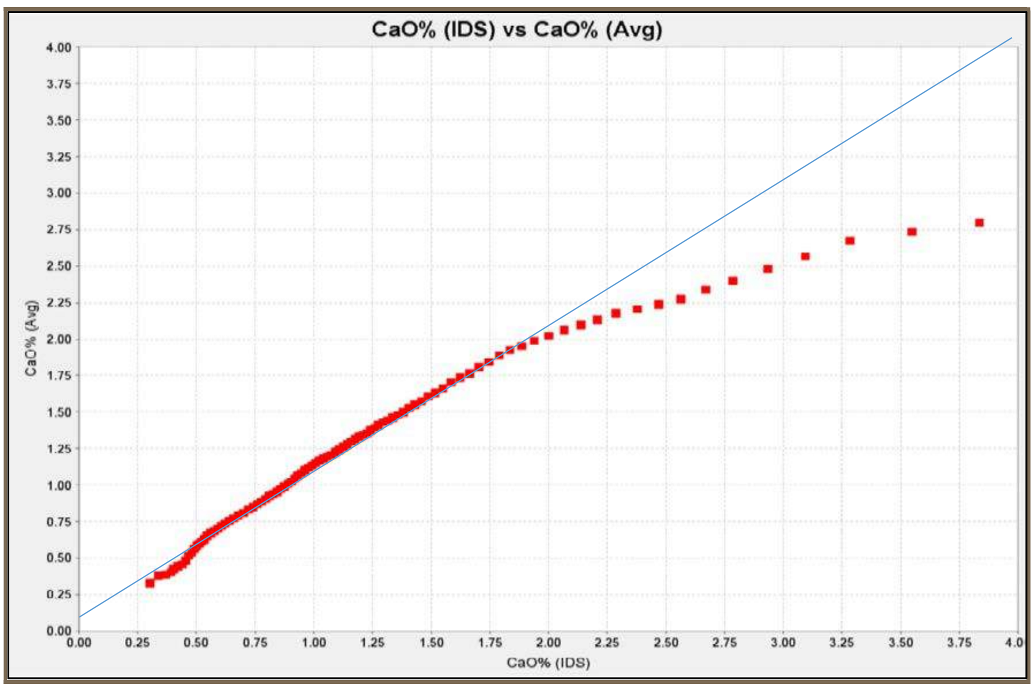


Figure 14-15: Q-Q Plot of CaO (IDS estimate) vs. CaO (Mean) Grades

At the higher end of the range, the IDS block estimate is lower than the composites, which suggests the presence of some miscoded calcic-dolomite core intervals. At the low end, the underestimation of CaO is considered insignificant relative to the MgO to be recovered.

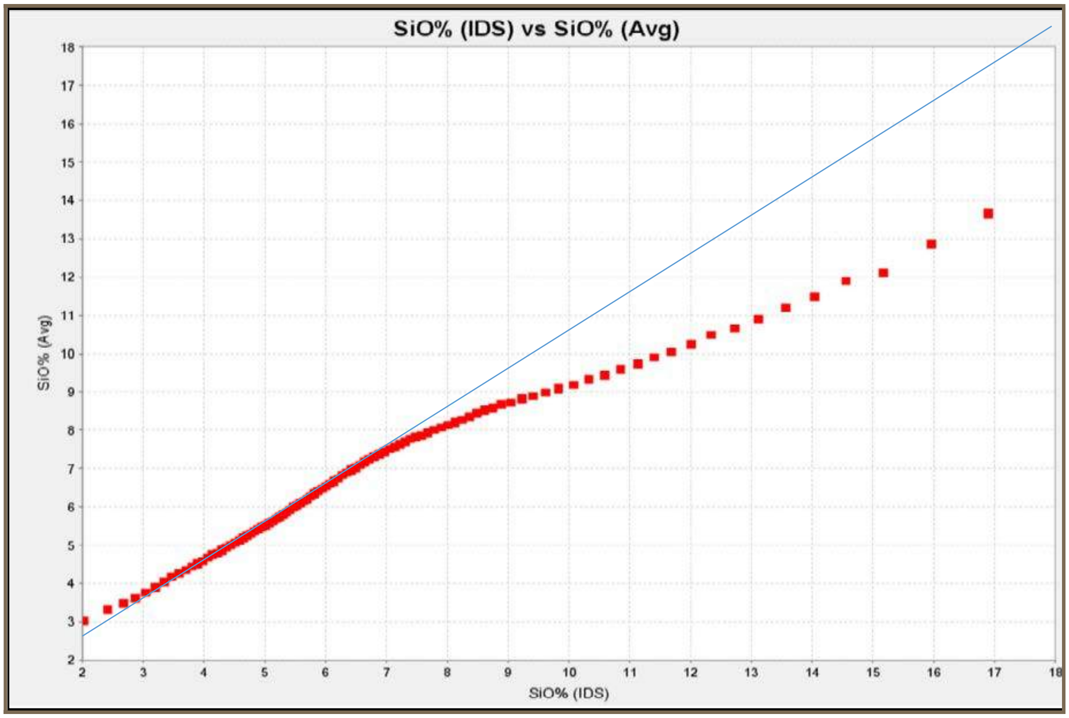


Figure 14-16: Q-Q Plot of SiO_2 (IDS estimate) vs. SiO_2 (Mean) Grades

Overall, silica is very well estimated within the magnesite bed. Tuun also created a grade-tonnage curve of MgO (Figure 14-17).

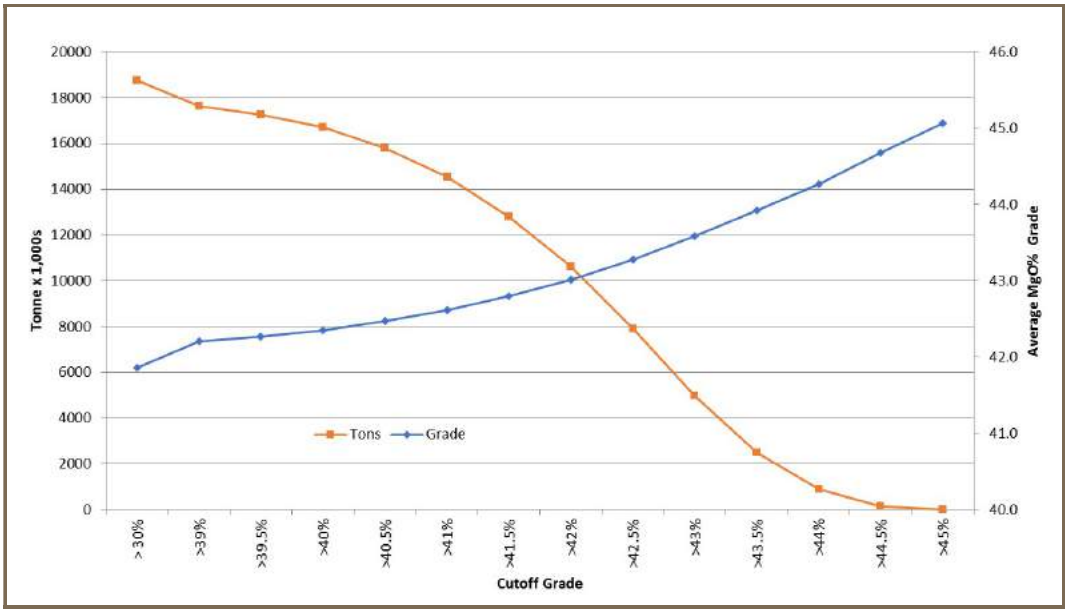


Figure 14-17: In Situ Grade-Tonnage Curve for Magnesium Oxide Resources

14.10 Mineral Resource Classification

The Driftwood Creek Magnesite Deposit block model quantities and grade estimates were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves. The grade estimation was done by Mr. Allan Reeves, P.Geo., of Tuun.

This Mineral Resource classification considered the geological continuity of the mineralized zones and the quality and quantity of exploration data supporting the estimates. The effective date of the Mineral Resource statement is December 31, 2016.

The estimate follows the guidelines of the generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines" (as adopted on November 23, 2003).

Tuun is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource estimation.

The mineralization generally exhibits good geological continuity, and has been investigated at an adequate spacing with reliable and accurately located sampling information. Tuun considers that blocks estimated during the first estimation pass by at least three drill holes can be classified in the Measured category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves.

Blocks that were estimated during the second pass were classified in the Indicated category, and those in the third pass as Inferred. Tuun believes that the level of confidence is sufficient to allow appropriate application of technical and economic parameters.

On November 28, 2015, CIM Council adopted a submittal by the Commodity Price Sub-Committee of the CIM Best Practices Committee: *"Guidance on Commodity Pricing used in Resource Estimation and Reporting."*

With respect to the CIM Definition of "reasonable prospects of eventual economic extraction," Tuun considered that the product is an industrial mineral and could possibly be quarried to the lithological contacts rather than to a defined cutoff grade to meet variable market requirements. It is the QP's judgement that such magnesium oxide quarrying would be a long-term prospect with upside in the use of steel alloys or environmental applications.

14.11 Magnesium Oxide Product Development and Current Pricing

Magnesite is magnesium carbonate ($MgCO_3$), plus other elements, that, upon calcination, releases CO_2 and leaves magnesium oxide (MgO). The molecular weight of $MgCO_3$ is 84.3, and for MgO it is 40.3. So that means on 100% basis one has to calcine 2.1 kg of $MgCO_3$ to produce 1 kg of MgO; the difference in weight is given off as CO_2.

Caustic calcining in the first stage creates CCM. From pure magnesite, the grades produced are normally around 93% MgO, and the rest are organics (that show up as LOI), silica (insoluble), lime (CaO), alumina, and iron oxides. It is important for marketing that lime content not exceed 5%, and preferably 3% to 4%. Other components are normally: no more than 5% LOI; 2% to 3% silica; no more than 1% to 2% aluminum oxide; and ideally, a maximum of 1% iron oxide.

In the second stage, one can calcine it to form DBM, which results in a product that is around 90% MgO (but can vary from 88% to 92%). Note that from synthetic routes (i.e., from brine) a higher purity product of up to 98% can be achieved, although most of the material coming in from China is at the 88% to 92% MgO level, as this is a practical level starting off with beneficiated magnesite. Typical composition of DBM from China is 90% MgO, 1% LOI, 4% to 5% silica, 2% to 4% lime, a maximum 2% iron oxide, and 2% aluminum oxide.

Practically this means that the magnesite has to be beneficiated to the point where it will yield material with this composition: the amount of each of the other components and what happens to them in calcination dictates how much one needs to process in order to make a tonne of MgO. Efficiency to calcine to CCM should be around 95%. For the second stage, going to DBM, efficiency will be lower due to briquetting and grinding, but if that is the final desired product one can use lower-content magnesite initially as one can lose 3% to 4% of LOI in the process, but MgO-content will also be lower.

Current market pricing (in US$) for CCM material is at the low end ≈$300/t and for typical large customers $325/t (currently the average price is $440/t in the US, and $350/t in Canada). For DBM material from China, it is available there for $250/t to $280/Mt, but that is as clinker, and is $280/t to

$300/t for ground material. For comparison purposes, one has to take into account taxes and transportation inland to a North American grinding facility. The material has to be ground and repackaged, and then shipped to market. When all those costs are added in, the final product sales prices could range from $500/t to $700/t delivered.

Note that the ability to accurately determine historical pricing averages and costs of production is impacted by confidentiality agreements that prevent access to proprietary information. For example, the only operating Canadian magnesite quarry is Baymag Inc.'s Mt. Brussilof Magnesite Mine near Radium, BC. The author is also aware of the Tami-Mosi Project in Nevada, which published an updated PEA in September 2011. Neither of these sources has current cost estimates applicable to the Project.

14.12 Mineral Resource Statement

The Mineral Resource statement has been prepared under the CIM Definition Standards for Mineral Resources and Mineral Reserves (adopted by CIM Council on May 10, 2014) which defines:

> *Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.*
>
> *A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.*
>
> *The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.*
>
> *Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.*
>
> *The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing*

method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity, and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

MEASURED MINERAL RESOURCE

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

REPORTING OF INDUSTRIAL MINERALS

When reporting Mineral Resource and Mineral Reserve estimates relating to an industrial mineral site, the Qualified Person(s) should be guided by the Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines for Industrial Minerals.

This Mineral Resource is based on drill data, BC Assessment Reports, and graphical cross-sections developed over many years. The information was reviewed and all work believed to have been executed in a professional manner based on the standards of care of the times.

In Tuun's opinion, the existing sample data is considered adequate for estimating the Mineral Resource. Tuun considers that the primary focus of the Driftwood Creek magnesite deposit will be amenable to magnesium oxide quarrying by a small excavator and truck fleet.

Given the variability of marketable MgO, it is the author's opinion that using a higher cutoff grade based on the selected annual production rate, and in line with the life of a furnace/kiln plant, is most appropriate for this updated Resource Estimate. Table 14-13 summarizes the Classified Resource by the selected cutoff grade of 42.5% MgO.

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all, or any part of the Mineral Resources estimated will be converted into Mineral Reserves.

Table 14-13: Preliminary Economic Assessment – % MgO Resource Estimate

Class	Pit	Tonnage (t x '000)	Grade					
			MgO %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	CaO %	LOI %
Measured	East	1,098	43.15	0.96	4.81	0.88	1.39	48.22
	West	3,605	43.36	1.03	5.13	1.42	0.82	47.71
	Total	**4,703**	**43.31**	**1.01**	**5.06**	**1.29**	**0.95**	**47.83**
Indicated	East	287	42.94	1.04	5.20	0.92	1.54	47.89
	West	2,858	43.24	0.99	4.61	1.47	1.00	48.00
	Total	**3,145**	**43.22**	**0.99**	**4.66**	**1.34**	**1.05**	**47.99**
M&I	East	1,385	43.11	0.98	4.89	0.89	1.42	48.15
	West	6,463	43.31	1.01	4.90	1.44	0.90	47.84
	Total	**7,848**	**43.27**	**1.00**	**4.90**	**1.34**	**0.99**	**47.89**
Inferred	East	-	-	-	-	-	-	-
	West	56	42.95	0.93	6.07	1.43	0.66	47.46
	Total	**56**	**42.95**	**0.93**	**6.07**	**1.43**	**0.66**	**47.46**
All Classes	East	1,385	43.11	0.98	4.89	0.89	1.42	48.15
	West	6,518	43.31	1.01	4.91	1.44	0.90	47.84
	Total	7,903	43.27	1.00	4.91	1.34	0.99	47.82

Notes: *__1.__ Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves. __2.__ The Lerchs-Grossman (LG) constrained shell economics used a mining cost of US\$8.82/t, processing+ g&a costs of US\$106/t, and a commodity price of US\$600.00/t 95%MgO DBM. __3.__ Mineral resources are reported within the constrained shell, using a cutoff grade of 42.5% MgO (based on a 20 year LOM) to determine "reasonable prospects for eventual economic extraction." __4.__ Mineral Resources are reported as undiluted. __5.__ Mineral Resources were developed in accordance with CIM (2014) guidelines. __6.__ Tonnages are reported to the nearest kilotonne (kt), and grades are rounded to the nearest two decimal places. __7.__ Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade, and contained metal.*
M&I = Measured and Indicated; t = tonnes% = percent; LOI = loss on ignition.

Table 14-14: Resource Estimate Summarized by Cutoff Grades

Cutoff Grade (%)	Tonnage (t x '000)	Grade					
		MgO %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %	CaO %	LOI %
>45	2	45.06	1.01	3.41	1.38	0.39	48.59
>44.5	133	44.68	0.86	3.12	1.35	0.67	48.88
>44	881	44.27	0.91	3.60	1.38	0.69	48.64
>43.5	2,498	43.92	0.99	4.14	1.37	0.74	48.29
>43	4,979	43.58	0.99	4.54	1.36	0.87	48.10
>42.5	7,903	43.27	1.00	4.91	1.35	0.99	47.90
>42	10,616	43.01	1.00	5.27	1.35	1.07	47.69
>41.5	12,807	42.80	1.01	5.59	1.35	1.13	47.51
>41	14,530	42.62	1.01	5.88	1.35	1.18	47.33
>40.5	15,784	42.47	1.02	6.13	1.34	1.22	47.16
>40	16,689	42.35	1.02	6.34	1.34	1.24	47.05
>39.5	17,254	42.27	1.02	6.48	1.33	1.27	46.91
>39	17,610	42.21	1.02	6.58	1.33	1.27	46.91
>30	18,737	41.86	1.01	7.01	1.33	1.37	46.65

Notes: ***1.*** *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.* ***2.*** *The Lerchs-Grossman (LG) constrained shell economics used a mining cost of US$8.82/t, processing+ g&a costs of US$106/t, and a commodity price of US$600.00/t 95%MgO DBM.* ***3.*** *Mineral resources are reported within the constrained shell, using a cutoff grade of 42.5% MgO (based on a 20 year LOM) to determine "reasonable prospects for eventual economic extraction."* ***4.*** *Mineral Resources are reported as undiluted.* ***4.*** *Mineral Resources were developed in accordance with CIM (2010) guidelines.* ***5.*** *Tonnages are reported to the nearest kilotonne (kt), and grades are rounded to the nearest two decimal places.* ***6.*** *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade, and contained metal.*
M&I = Measured and Indicated; t = tonnes; % = percent; LOI = loss on ignition

14.13 Preliminary Economic Assessment Mineral Resource Parameters

The LG pit optimization parameters, summarized in Table 14-15, resulted in the generation of two pits (East and West), separated by a barren zone considered to have resulted from being faulted away.

There are many different markets for the production of MgO, which helps stabilize the demand for the production of these compounds. Should demand in one industry decline, a plant can then produce a product for another industry. This holds true if the plant has the capability to do so, which MGX would since they are investigating both a MHF and a VSK.

Table 14-15: Pit Optimization Parameters

Parameter	Unit	2016 Resource Estimate
Process Output	t/a MgO Production	440,000
MgO Price	US$/MgO % DBM	600
US$:C$ Exchange Rate	$	0.77
Mining Cost	C$/t mined	8.82
Transport Mine to Plant	C$/t processed	43.95
Processing and G&A Costs	C$/t processed	62.06
Total Processing	**$**	**106.01**
Total Recovery	**%**	**90.0**
Mine Production Rate	t/a	440,000
Pit Slopes	degrees	50
Blocks		M+I+I

Notes: *M+I+I = Measured + Indicated + Inferred; t/a = tonnes per annum; % = percent C$ = Canadian dollar; US$ = United States dollar; G&A = general and administrative; DBM = dead-burned magnesia*

15 MINERAL RESERVE ESTIMATE

The Project has no declared Mineral Reserves as per CIM definitions.

16 MINING METHODS

Mine design and planning for the Project is based on the Tuun resource model, as detailed in Section 14 of this report. Mine planning and optimization results are based on Measured, Indicated, and Inferred resources for magnesium oxide (MgO).

This section outlines the parameters and procedures used to perform pit optimization and subsequent mine planning work for the Project.

16.1 Overview

The deposit will be a conventional, quarry pit, truck-and-excavator operation. A plant feed of approximately 1,200 t/d is planned over a 19-year life-of-mine (LOM). There will be pre-strip material in Year -1, with full production ramp-up in Year 1.

The mine design and planning, cutoff grade reporting, and optimization were completed using Maptek Vulcan^{TM} v9.1.1 software. Optimization was performed using the Lerchs-Grossman (LG) algorithm to determine an optimized shell. The ultimate pit was designed to develop the LOM plan.

Acid base accounting (ABA) testing was not available for rock material at the time of this study. The next level of study will require developing a rock management plan to categorize the material planned to be mined.

Table 16-1 shows the key results from the LOM plan. Non-resource material to be mined and the associated strip ratio include pre-stripping activities in Year -1.

Table 16-1: LOM Plan Key Results

Description	Unit	Value
Mineral Resource Material Mined	Mt	7.843
Average MgO Grade	%	43.27
Average CaO Grade	%	1.00
Average Al_2O_3 Grade	%	1.00
Average SiO_2 Grade	%	4.88
Average Fe_2O_3 Grade	%	1.34
Average LOI Grade	%	47.92
Non-Resource Material to be Mined	Mt	19.174
Strip Ratio	nr:r	2.4
Milling Rate	t/d	1,200
Project Life	years	19

Notes: *Mt = million tonnes; % = percent; t/d = tonnes per day; LOI = loss on ignition, nr:r = non-resource:resource*

16.2 Geotechnical

A geotechnical drilling program was underway at the time of the report. Assumptions of 50° overall slopes and 85° bench face angles were used for this study, pit optimization, and pit design. These assumptions were based on the surrounding rock outcrops.

16.3 Open Pit Optimization

16.3.1 2017 Optimization Parameters

The percent block model was provided by Tuun in an ASCII format with a 5 m (X) by 5 m (Y) by 5 m (Z) block size, and transferred into Maptek Vulcan™ software.

Parameters defined and outlined in Table 16-2 were estimated using the limited information available in early 2017. No capital costs were considered at the time of this study. Optimizations were run using Measured, Indicated, and Inferred Mineral Resources.

Table 16-2: 2017 Pit Optimization Parameters

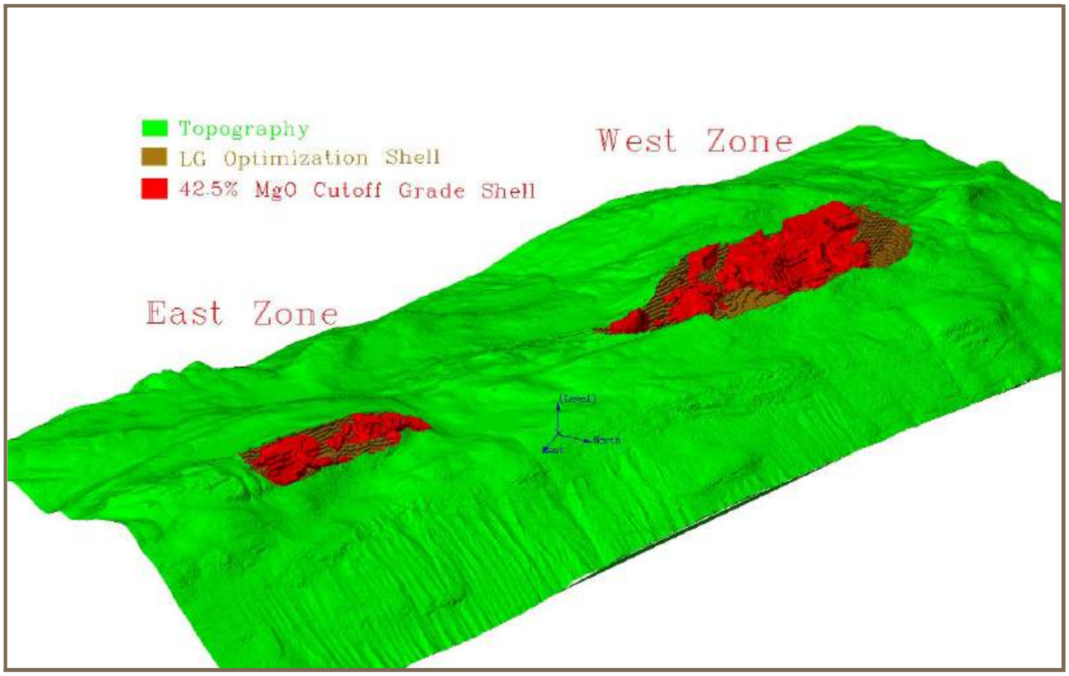
Parameter	Unit	Value
MgO Price	US$/t	400
Exchange Rate	US$:C$	0.84
Processing + G&A Costs	C$/t processed	106.91
Transportation Costs	C$/t processed	22.50
Mining Cost	C$/t mined	6.25
MgO Recovery	%	87
Pit Slopes Overall	degrees	50

Source: *AKF (2018)*

Notes: *G&A = general and administrative; US$/t = United States dollars per tonne; C$/t = Canadian dollars per tonne; % = percent*

16.3.2 2017 LG Pit Optimization Results

The LG optimization shell resulted in a resource of 7.916 Mt grading at 43.4% MgO using a 42.5% MgO cutoff. Non-resource material totalled 19.238 Mt with an overall strip ratio of 2.4:1 (non-resource:resource). Comparing the LG optimization shell with the pit design, the material difference in resource and non-resource tonnes was -1% and -0.3%, respectively. This is below the 'rule of thumb' of 5% for material differences between LG optimization shell and pit design. Figure 16-1 shows the LG optimization shell with a 42.5% MgO grade cutoff.

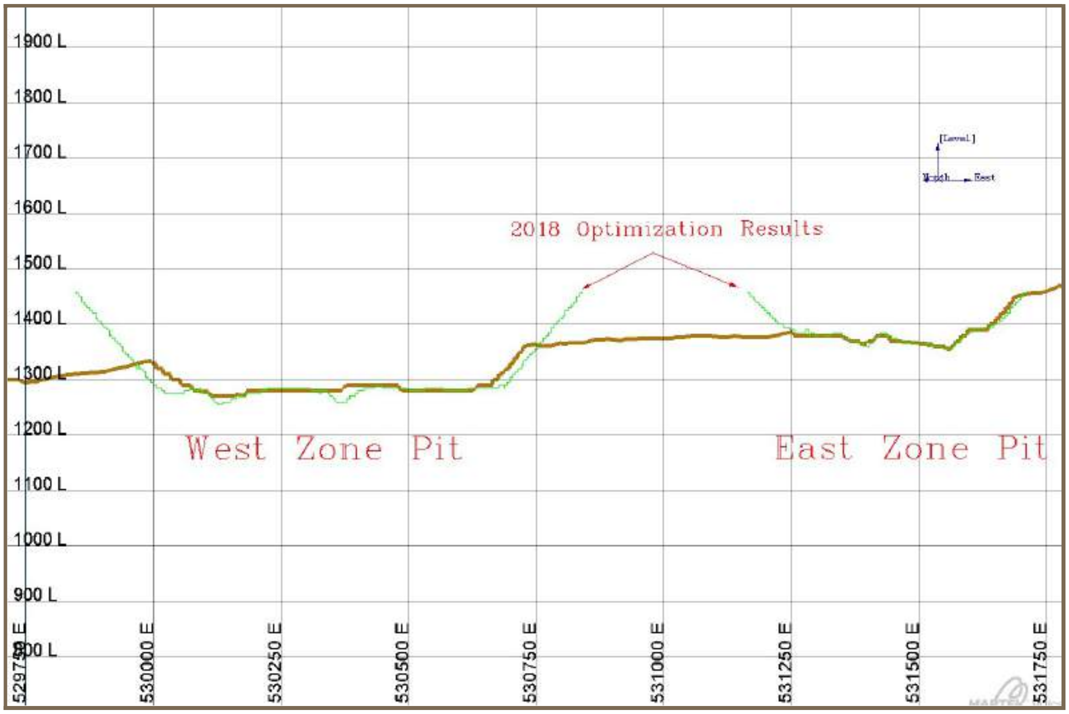


Source: *AKF (2018)*

Figure 16-1: Isometric View Looking Southwest showing LG Optimization Shell with 42.5% MgO Cutoff Grade Shell

16.3.3 2018 LG Optimization Update vs. 2017 Pit Design

The LG optimization shell used for the final pit designs in 2017 was based on the 2017 optimization parameters. Once the Project cost and prices were established for the cash flow model in early 2018, a comparison of the 2017 pit design with the 2018 updated values in Section 22, Table 22-2, shows the pit designs are well within the updated optimization shell, and validated the 2017 pit designs, as shown in Figure 16-2.



Source: *AKF (2018)*

Figure 16-2: 2018 Optimization Update vs. 2017 Pit Design

16.4 Mine Planning

16.4.1 Mine Design

The key focus of this PEA was to maximize the open pit resources and to show "reasonable prospects of eventual economic extraction." The final pit was designed based on the economic factors used in the 2017 pit optimization work, as shown in Table 16-3. The design for this study was completed following industry standards, and the design parameters are shown in Figure 16-4.

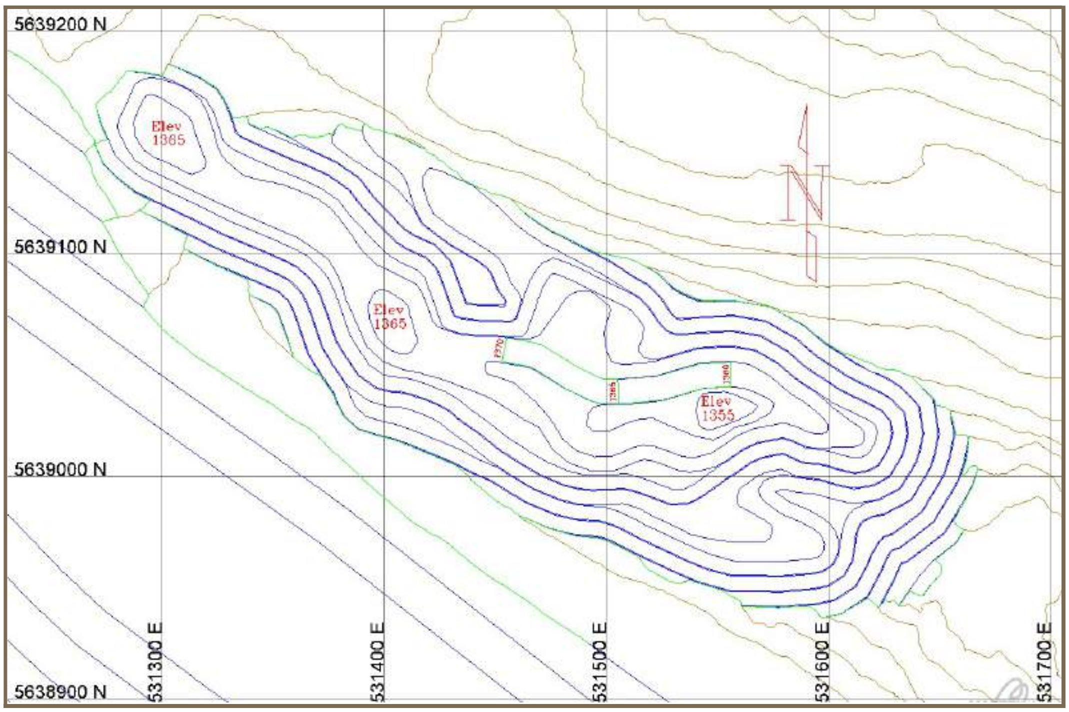
Mining will be performed on two 5 m "sub-benches," with an overall final bench height of 10 m. Two pits were designed for the Project, the East Zone and West Zone pits, shown in Figure 16-3 and Figure 16-4. No phase designs were completed for this Project, as the Mineral Resource is at or near surface. At the next level of study, a mine planning study should be conducted to optimize stripping requirements to further improve Project economics.

Table 16-3: Open Pit Design Parameters

Parameter	Value
Bench Height (Single/Double)	5 m / 10 m
Catchment Width	6 m
Ramp Gradient	10%
Ramp Width (Dual/Single)	14.0 m / 11.0 m
Bench Face Angle	85 degrees

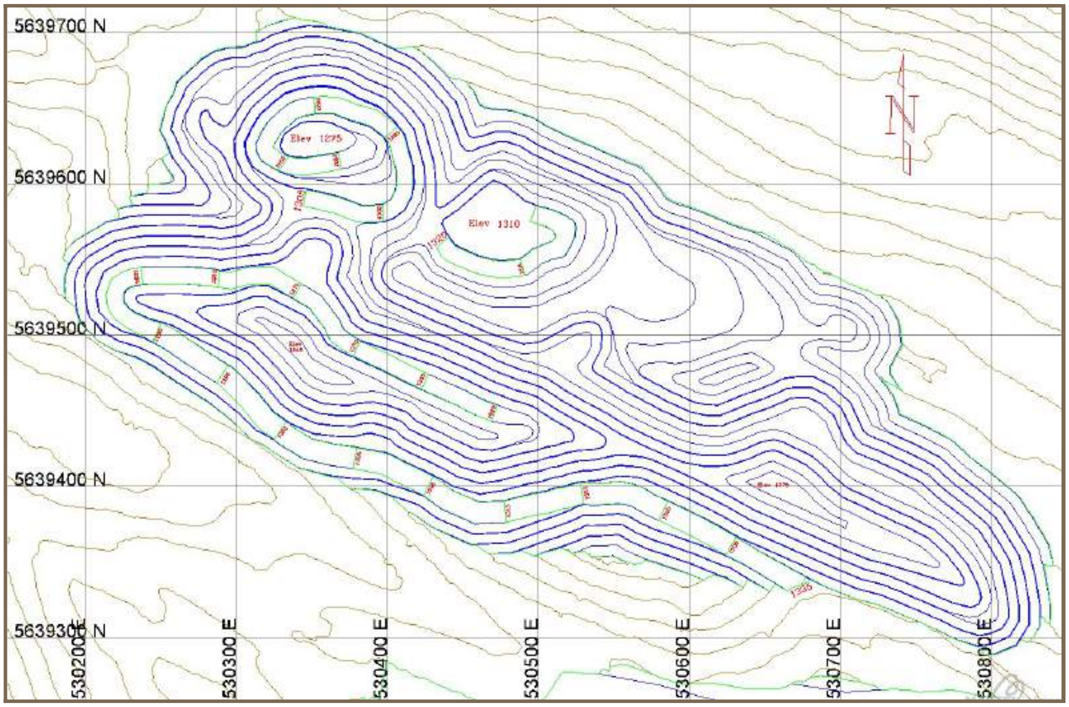
Source: *AKF (2018)*

Notes: *m = metres; % = percent*



Source: *AKF (2018)*

Figure 16-3: East Zone Pit



Source: *AKF (2018)*

Figure 16-4: West Zone Pit

16.4.2 Haul Ramp Design

The ramp design width follows the guidelines set out in the Health, Safety and Reclamation Codes for Mines in British Columbia, which calls for "a travel width where dual traffic exists of not less than three times, or where single lane traffic exists, of not less than two times the width of the widest haulage vehicle used on the road, and the shoulder barrier at least ¾ of the height of the largest tire on any vehicle hauling on the road." The current design uses a Cat D300 (30 tonne) articulated truck as the largest haul truck traveling on the ramp, with a truck width of 2.89 m. Tire size is based on the 23.5R25, with an overall height of 1.61 m. The calculated operating width for dual lane ramps is 14.0 m, and for single lanes is 11 m. These include a 1.0 m ditch for water runoff and snow containment.

At the time of the design, and with limited information available, the assumption was to use CAT D300 articulated trucks. As the Project progressed, a contractor was then facilitated to deliver a mining quote with equipment specifications. The contractor selection for haul trucks was to use the CAT 771. The haul road specifications for a CAT 771 haul truck are 15.0 m for double lanes and 11.0 m for single lanes. In comparing the current design with the contractor equipment, the design can facilitate single lane access based on the Health, Safety, and Reclamation Codes for Mines in British Columbia guidelines. It is recommended that the ramp design be updated in the next project iteration.

16.4.3 MgO% Cutoff Grade

A mine cutoff grade calculation was determined for the mine production schedule, based on the optimization parameters provided in Table 16-2:

$$\text{Mine Cutoff Grade} = (\text{Mine} + \text{Processing and G\&A} + \text{Transportation Costs}) / ((\text{Price/Exchange Rate}) * \text{Recovery})$$

$$\text{Mine Cutoff Grade} = (6.25 + 106.91 + 22.50) / ((400/0.84) * 87\%) = 32.7\%\ \text{MgO}$$

To maximize value during the LOM, a cutoff grade of 42.5% MgO was established, based on expected grade of concentrate to market. At the time of this study, material between 32.7% and 42.5% MgO was considered non-resource material; this should be reviewed in a later study to determine market potential for sales of lower-grade material.

16.4.4 Resource Loss and Dilution

At this level of study, to determine "reasonable prospects of eventual economic extraction," it was assumed that the current geological model incorporates some level of dilution. The in-pit mine resources tabulated in Section 16.4.3 therefore do not at this time include dilution or resource loss, but it will be recommended to include them at the next level of study.

16.4.5 In-Pit Non-Diluted Resources

The total in-pit, non-diluted, Measured, Indicated, and Inferred resources are tabulated below, with a 42.5% MgO cutoff by pit.

Table 16-4: In-Pit Non-Diluted Resources at 42.5% MgO Cutoff by Pit

Pit	In-Pit Resources							In-Pit Non-Resource Material (Mt)	Total Ex-Pit Material (Mt)	Strip Ratio (nr:r)
	Tonnage (Mt)	MgO (%)	CaO (%)	Al_2O_3 (%)	SiO_2 (%)	Fe_2O_3 (%)	LOI (%)			
East Zone	1.401	43.11	1.42	0.98	4.89	0.89	48.15	3.20	4,599	2.28
West Zone	6.44	43.31	0.90	1.00	4.88	1.44	47.87	15.98	22,418	2.48
Total	**7.843**	**43.27**	**1.00**	**1.00**	**4.88**	**1.34**	**47.92**	**19.174**	**27,017**	**2.44**

Source: *AKF (2018)*

Notes: ***1.*** *Tonnages are reported to the nearest million tonnes, and grades are rounded to the nearest two decimal places.*
2. *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade, and contained metal.*
Mt = million tonnes; % = percent; LOI = loss on ignition; nr:r = non-resource:resource

16.4.6 Mine Production Schedule

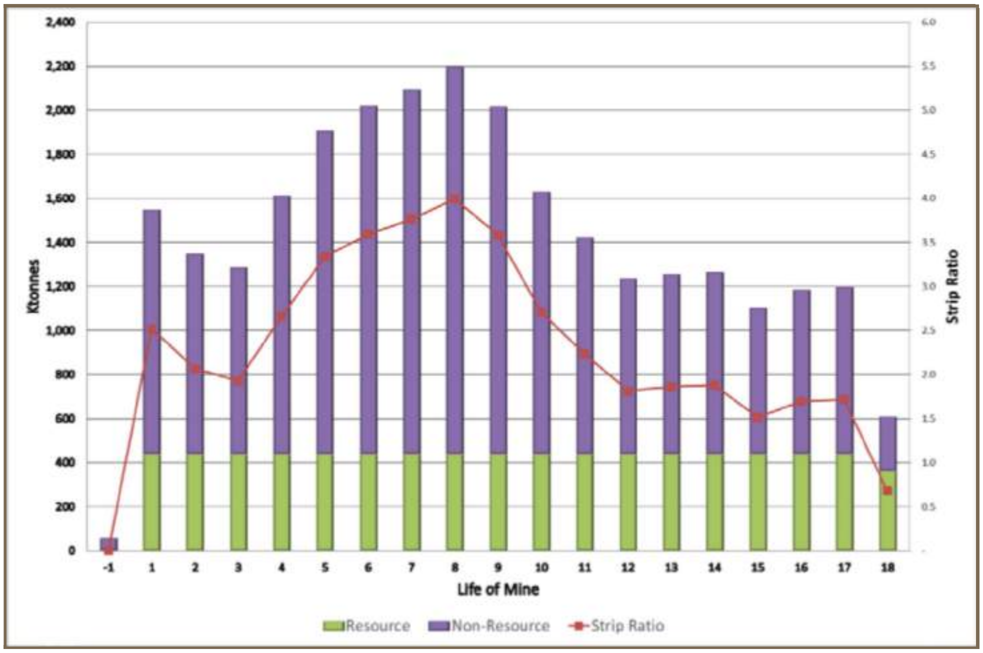
The mining production schedule was developed based on a maximum plant capacity of approximately 1,200 t/d (440 kt/a). The Project life is 19 years, with one year of pre-stripping followed by 18 years of operations. The throughput rate is assumed to achieve full capacity by Year 1 of operations. Table 16-5 and Figure 16-5 outline the mine production schedule by year, and Figure 16-6 outlines the plant production with MgO% grade by year.

Table 16-5: Mine Production Schedule

Year	Resources to the Plant (19 Year Schedule) Tonnes (kt)	MgO (%)	CaO (%)	Al_2O_3 (%)	SiO_2 (%)	Fe_2O (%)	LOI (%)	Non-Resource Material (kt)	Total Ex-Pit Material (kt)	Strip Ratio (nr:r)
-1	-	-	-	-	-	-	-	60	60	-
1	440	43.13	1.63	0.96	4.86	0.92	47.97	1,109	1,549	2.5
2	440	43.11	1.35	0.99	4.94	0.86	48.12	909	1,349	2.1
3	440	43.13	1.28	0.99	4.82	0.88	48.32	848	1,288	1.9
4	440	43.05	1.23	0.78	4.78	1.36	48.34	1,172	1,612	2.7
5	440	43.10	1.25	0.75	4.38	1.47	48.49	1,469	1,909	3.3
6	440	43.19	1.13	0.75	4.54	1.47	48.40	1,581	2,021	3.6
7	440	43.25	1.18	0.75	4.39	1.48	48.45	1,655	2,095	3.8
8	440	43.16	1.10	0.85	4.78	1.45	48.16	1,757	2,197	4.0
9	440	43.26	0.96	0.88	4.90	1.43	48.10	1,577	2,017	3.6
10	440	43.31	0.90	0.89	4.82	1.42	48.09	1,192	1,632	2.7
11	440	43.38	0.87	0.88	4.74	1.44	48.07	984	1,424	2.2
12	440	43.48	0.77	0.93	4.80	1.44	47.92	799	1,239	1.8
13	440	43.44	0.71	1.03	4.97	1.43	47.73	819	1,259	1.9
14	440	43.36	0.68	1.17	5.20	1.44	47.47	826	1,266	1.9
15	440	43.33	0.71	1.26	5.28	1.45	47.28	667	1,107	1.5
16	440	43.42	0.70	1.35	5.25	1.43	47.17	746	1,186	1.7
17	440	43.48	0.74	1.35	5.02	1.41	47.34	756	1,196	1.7
18	363	43.38	0.68	1.41	5.51	1.43	46.92	247	610	0.7
Total	**7,843**	**43.27**	**1.00**	**1.00**	**4.88**	**1.34**	**47.92**	**19,174**	**27,017**	**2.4**

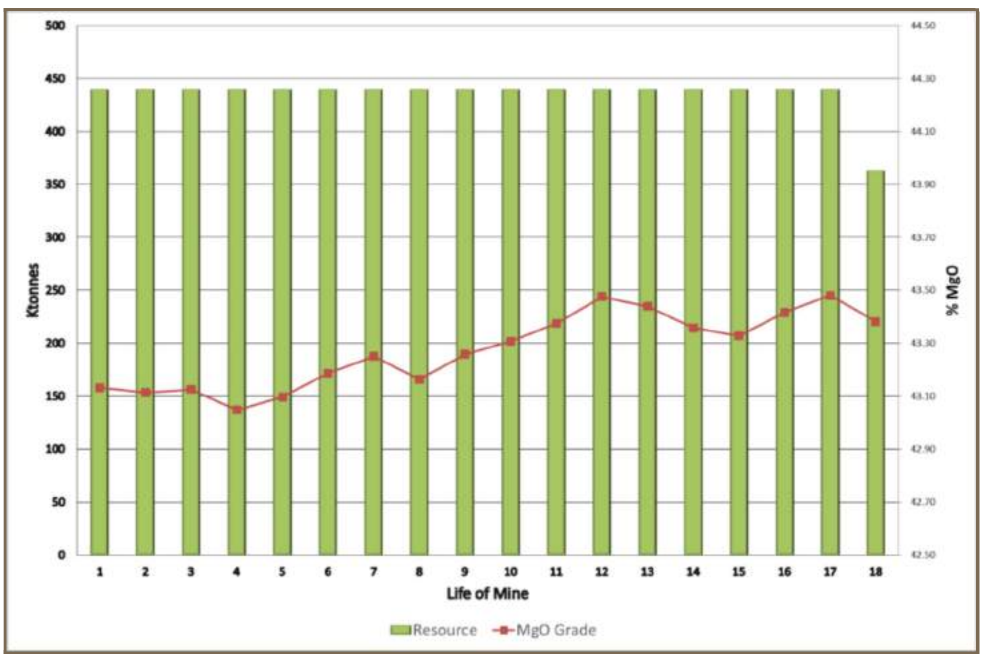
Source: *AKF (2018)*

Notes: ***1.*** *Tonnages are reported to the nearest kilotonne, and grades are rounded to the nearest two decimal places.* ***2.*** *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade, and contained metal.*
kt = thousand tonnes; % = percent; LOI = loss on ignition; nr:r = non-resource:resource.



Source: *AKF (2018)*

Figure 16-5: Mine Production Schedule and Strip Ratio



Source: *AKF (2018)*

Figure 16-6: Plant Production Schedule and MgO Grade

During the mine scheduling exercise, the East Zone was scheduled to be mined first and the West Zone last; this represented the lowest overall initial strip ratio, deferring the higher stripping requirements until later in the Project life, allowing for earlier payback and helping improve Project economics.

Only 60,000 tonnes of non-resource material will need to be moved during pre-stripping. The level of organics that will need to be moved is unknown at the time of this study. It is AKF's opinion that only a small percentage of the pre-stripping requirements are likely to be associated with the removal of organics, which will be stockpiled at the RMF.

All resource material to the plant will be mined and hauled downhill in 40-tonne mine haul trucks to the ready pile located at the bottom of the hillside, as shown on Figure 18-4. The resource material will then be loaded onto 40-tonne highway trucks and transported to the plant facility in Cranbrook, BC.

16.5 Mine Rock Management

Over the LOM, the open pit will produce approximately 19.174 Mt of non-resource material rock. At the time of this study, the mine rock ABA information was not available; therefore, all mine rock has been categorized at this time as non-potential acid-generating (NAG) rock, but it will be recommended to the update the ABA at the next level of study.

16.5.1 Rock Management Facility Design

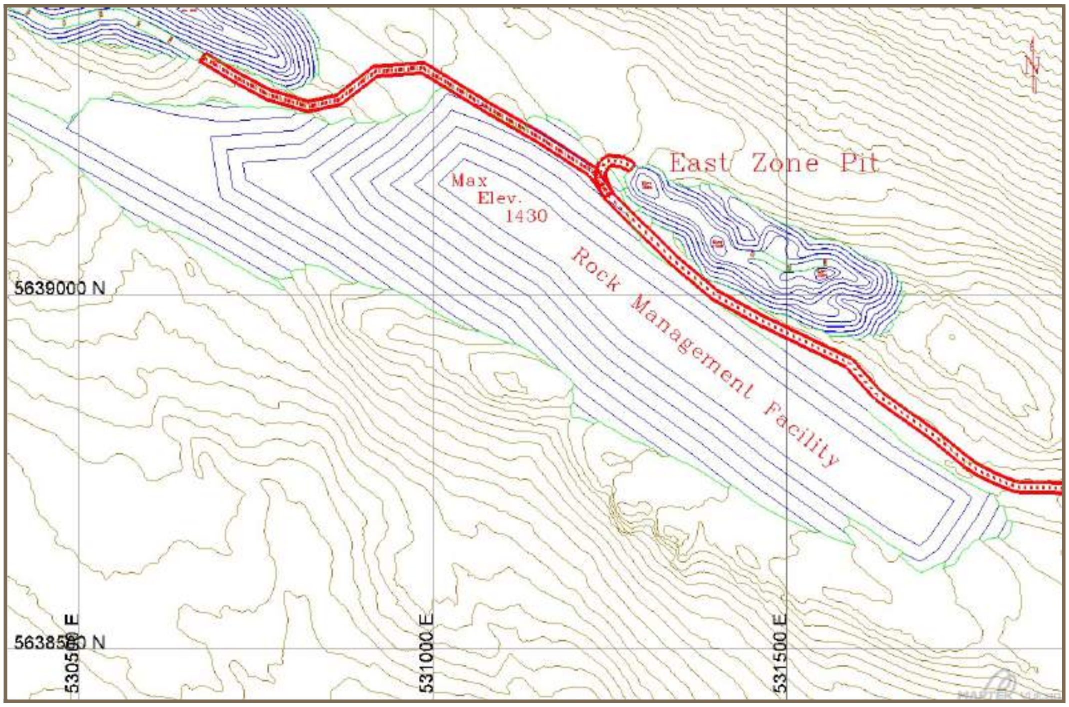
No geotechnical information was available at the time of the study. The RMF was designed following industry standard parameters.

Table 16-6: Rock Management Facility Design Parameters

Parameter	Value
Bench Lift	10 m
Swell Factor	30%
Overall Slope	26.6 degrees (2:1 Horizontal:Vertical)
Bench Face Angle	38 degrees (Angle of Repose)

Source: *AKF (2018)*
Notes: *m = metres; % = percent*

All mine rock material will be delivered to the RMF, proposed location is downhill from the East Zone Pit and southeast of the West Zone Pit. It will facilitate both free- and end-dumping on 10 m lifts with a catchment berm of 6 m to an overall design slope of 2(H):1(V). The maximum design elevation is 1,430 masl, and it has a footprint area of approximately 0.32 km^2. Figure 16-7 shows the RMF.



Source: *AKF (2018)*

Figure 16-7: Rock Management Facility Proposed Location

16.6 Contractor Mine Equipment

All mine and mine support equipment will be provided by contractors. The equipment description in this section provides general information on the size and/or capacity of the selected equipment.

This operation will be a conventional, quarry pit, truck-and-excavator operation. Track-mounted blasthole drills, either rotary drilling or down-the-hole (DTH), are planned for the Project. Due to the size of the operation, all equipment on site will be diesel powered.

16.6.1 Contractor Mine Equipment Parameters

The mine will operate 12 hour/day shifts, 360 days/year. The contractor's equipment is expected to have long-term mechanical availability of 85%. Utilization has been assumed to be 85%. This gives approximately 3,121 gross operating hours per year.

16.6.2 Contractor Mine Equipment Requirements

Table 16-7 lists major mine equipment to be provided by contractors, which was estimated based on the equipment parameters described above.

Table 16-7: Major Mine Equipment Requirements

Equipment Type	Initial	Ultimate
Crawler-mounted Ranger drills, 100 mm to 150 mm dia.	1	2
Diesel 2.4 m^3 hydraulic excavators	1	2
Diesel 3.1 m^3 hydraulic excavator	0	1
40-tonne class haul trucks	2	6
D8-class track dozer	1	1
14H-class grader	1	1

Source: *AKF (2018)*
Notes: *m^3 = cubic metres; mm = millimetre; dia. = diameter*

The contract mine support equipment will consist of:

- Pickup ambulance (cab-over);
- Trailer-mounted fire pump;
- Mechanics' tool truck;
- Pickups; and
- Trailer-mounted lighting towers.

16.7 Contractor Explosives

Explosives will be supplied by a single service contract, using packaged emulsion, and delivered by an onsite explosives truck to the blasthole. No explosives magazine storage facilities will be set up on site.

Blast design is based on 5 m (for resource material selectivity) and 10 m benches (for non-resource material), using powder factors of approximately 0.3 kg/t. Over the Project life, approximately 8 million kilograms (Mkg) of packaged emulsion will be used, with an average use of 0.5 Mkg/a.

The Project will use conventional blasting products: non-electric detonating cords, delays, and boosters.

The contractor will be responsible for blasting pattern design, loading holes, and tie-ins.

Pre-shearing explosives products should be evaluated at the next study stage to determine whether higher blasting costs to steepen the overall wall angle will reduce overall mining of non-resource material and costs. In addition, a blast fragmentation study should be conducted for the run-of-mine (ROM) resources, which will be transported to the plant facility in Cranbrook, BC.

16.8 Contractor ROM Haul for Resource Material

The contract ROM haul will operate 24 h/d, 365 d/a, on 8 hour shifts and a 5 days on/3 days off rotation. The contractor ROM haul will consist of 40-tonne highway trucks, with a tri-axle dump, and a quad-axle trailer. They will be loaded with a 2.4 m^3 bucket hydraulic excavator.

The contract ROM haul will travel from the mine site near Brisco, BC, to the plant site at Cranbrook, BC, with an approximate cycle time of 7 hours. There will be approximately three trips per day, per truck, based on a 20-hour operating day. The contract ROM hauler will operate about 10 highway trucks in each 24-hour period (approximately 30 trips). The return haul from the plant site will carry the dry-stack tailings to the mine site.

The contract ROM haul operators, equipment maintenance, and fueling station will be based at the plant site at Cranbrook, BC. All highway trucks will be scaled at the plant site to determine weights hauled per day.

16.9 Mine Personnel

The management staff and technical personnel will operate on a single 10-hour day shift, on a 4 days on/3 days off rotation. This will require two mining crews, working 12 hours per shift, on a standard rotation of 4 days on/4 days off. Personnel requirements are estimated based on the peak number of equipment units operating. Peak mine personnel requirements are estimated and summarized in Table 16-8 through Table 16-10.

Table 16-8: Mine Supervision Personnel Summary – Owner

Position	Quantity	Hourly/Salary
Mine Manager	1	Salary
Mine Engineer	1	Salary or Contract
Mine Geologist	1	Salary
Mine Surveyor	1	Salary or Contract
Mine Clerk	1	Salary
Mine Supervision Personnel Total	5	

Source: *AKF (2018)*

Table 16-9: Mine Operations Personnel Summary – Contractors

Position	Quantity	Hourly/Salary
Driller	1	Hourly (day shift only)
Blaster	1	Hourly (day shift only)
Blasting Helper	1	Hourly (day shift only)
Excavator Operator	6	Hourly (day & night shifts)
Truck Driver	12	Hourly (day & night shifts)
Support Equipment Operators	1	Hourly (day shift only)
Mine Services	1	Hourly (day shift only)
Mine Maintenance	2	Hourly (day shift only)
Mine Operations Total	25	

Source: *AKF (2018)*

Table 16-10: Total Mine Personnel Summary

Team	Personnel
Supervision – Owners	5
Operations and Maintenance – Contractors	25
Total Mine Personnel	**30**

Source: *AKF (2018)*

16.10 Important Caution Regarding Mine Planning

The PEA is preliminary in nature, in that 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.

17 RECOVERY METHODS

17.1 Introduction

The Driftwood Creek Magnesite property is located approximately 53 km southwest of Golden, BC. Mineralized material will be mined on site and then transported 210 km via truck to the plant located in Cranbrook, BC. Here the mineralized material will undergo crushing, grinding, flotation upgrading, calcination, and sintering to produce a saleable DBM product. The plant will also have the ability to produce CCM as a separate product.

The crushing plant operates one 12-hour shift, 365 days per year. The availability for the crushing plant is 50%. The grinding and processing plant will operate 24 hours a day using two 12-hour shifts. The grinding and processing plant will also operate 365 days a year with a 90% availability.

Figure 17-1 was used to represent the main process steps for this report. The flowsheet is partially based on the development work completed by SGS Lakefield and a proposal from Industrial Furnace Company Inc. for the calcination and sintering operations. The process operations can be divided into a three-stage approach, where the mineralized material will initially be sized and screened, then upgraded, and finally calcined into CCM and then sintered into DBM. It should be noted that limited metallurgical testwork has been completed to date, and the flowsheet prepared for this report follows industry-standard practices for the production of CCM and DBM.

Process design criteria were developed to support the preliminary Process Flow Diagram (PFD), plant design, equipment sizing, and capital cost estimate. The design criteria are presented below.

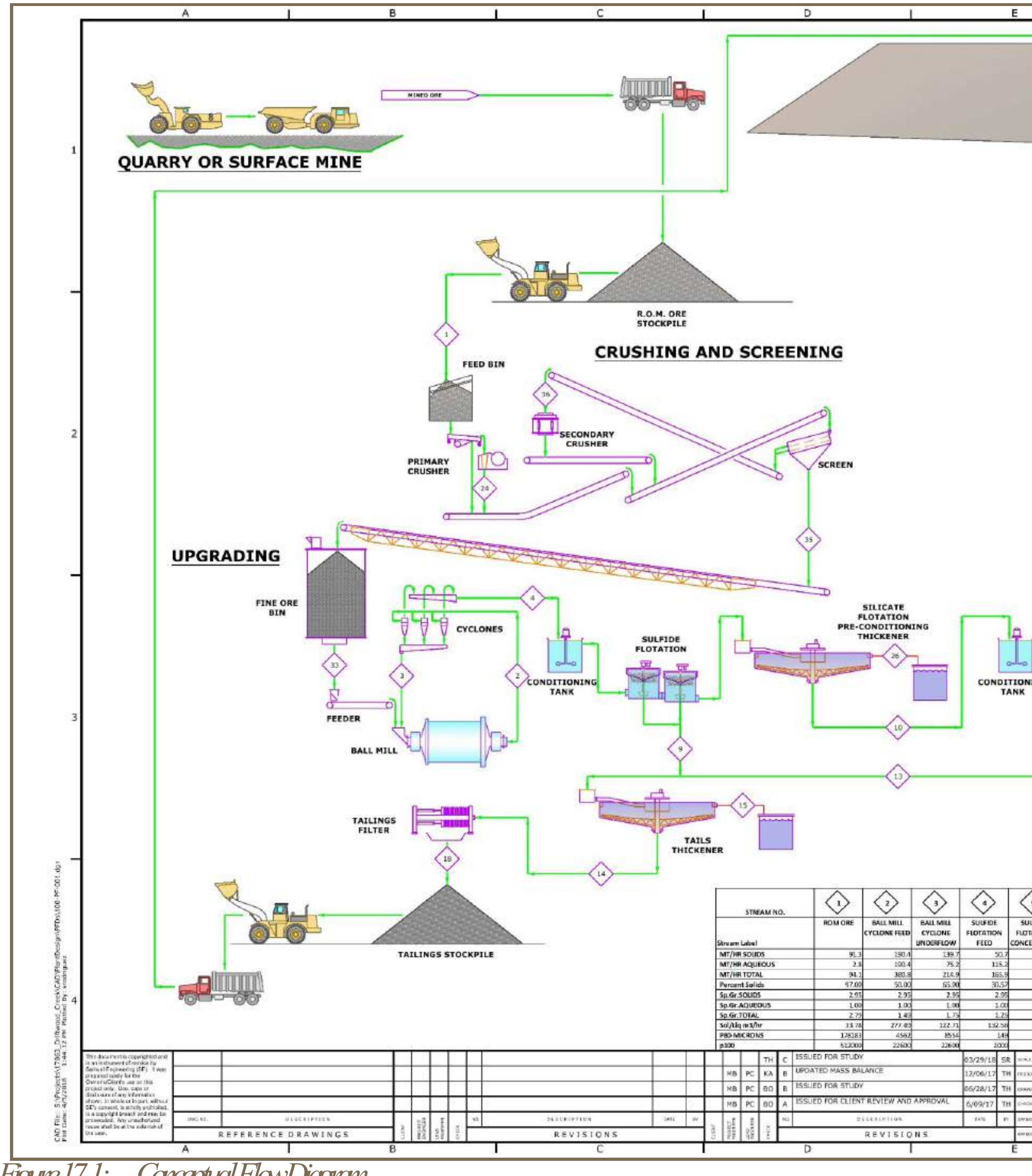


Figure 17-1: Conceptual Flow Diagram

Table 17-1: Process Design Criteria

	Unit	Nominal	Design	Source
General				
Location Coordinates	N	50 54' 12" N		Client
	W	116 33' 16"		Client
Elevation	m	949		SE
Climatic Conditions				
Wet Season		May–Sep.		SE
Dry Season		Dec.–Feb.		SE
Ambient Temperature Range				
High	˚C	32		Client
Low	˚C	-20		Client
Precipitation				
Annual Average	mm	405		Client
Highest Monthly (Nov.)	mm	50		Client
Lowest Monthly (Jul.)	mm	20		Client
Evaporation				
Annual Average	mm	457		SE
Production Rates				
Design Factor			1.10	SE
Mine Life	years	18.8	18.8	Client
Mill Annual Throughput	dmt/a	400,000	440,000	Client
Calendar Days per Year	d/a	365	365	Client
Shifts per Day	shifts/d	2	2	Client
Hours per Shift	h/shift	12	12	Client
Hours per Day	h/d	24	24	Client
Availability	%	90.0%	90.0%	Client
Daily Throughput	mt/d	1,200	1,300	Calculated
Hourly Throughput	dmt/h	50.7	55.8	Calculated
DBM Produced	t/a	169,700	186,700	Client
	t/d	498	547	Calculated
	t/h	20.7	22.8	Metsim
DBM Purity, % MgO	%	94.6	94.6	Client
CCM Produced	t/a	169,700	186,700	Calculated
	t/d	498	547	Calculated
	t/h	20.7	22.8	Metsim
CCM Purity, % MgO	%	94.6	94.6	Client

	Unit	Nominal	Design	Source
Material Characteristics				
Head Grade				
Magnesite, $MgCO_3$	%	93.4	86.3	Testwork
Silicate, SiO_2	%	4.88	5.91	Testwork
Ore Particle Size, P_{100}	mm	308	308	Client
Ore % Moisture	%	3.0	3.0	Client
Ore SG		2.99	2.99	Testwork
Ore Bulk Density	t/m^3	0.64	0.64	SE
Crusher Work Index	kWh/t	9		SE
Ball Mill Bond Work Index	kWh/t	11.1		SE
Abrasion Index		0.079		SE

Notes: *DBM = dead-burned magnesia; CCM = caustic-calcined magnesia; SG = specific gravity; SE = Samuel Engineering; kWh/t = kilowatt hour per tonne; % = percent; mm = millimetre; t/m³ = tonnes per cubic metre; h/d = hours per day; d/a = days per annum; dmt/h = dry metric tonne per hour; ˚C = degrees Celsius; d/a = days per annum; d = days; h = hours*

17.2 Comminution

Magnesite material with a top size of 308 mm will be transported from the mine site to the processing facility via over-the-road haul trucks. The magnesite material will be stockpiled at the plant site, where it will be reclaimed and fed into the primary crusher feed bin via a front-end-loader (FEL).

17.2.1 Crushing

A two-stage crushing operation has been designed, where the primary crushing stage will receive the material from the mine for a first pass sizing and screening. Primary crushing will be performed by a jaw crusher fed at a nominal rate of 91.3 dry metric tonne per hour (dmt/h). Crushed magnesite will be discharged to the crushing screen, where undersize material will be conveyed to the fine mineral storage bin, and oversize conveyed to secondary crushing.

The secondary crushing operation will consist of a cone crusher with a nominal feed rate of 80.2 t/h. Discharge from the secondary crusher will be conveyed to the crushing screen for particle size separation. Again, oversize material will be routed to the secondary crusher, and sized material will be transferred to the fine mineral storage bin.

The crushing and screening plant will be equipped with dust collection at all conveyor transfer points, and at the crushing and screening equipment, for dust control. The crushing and screening plant will be operated in one 12-hour shift to mitigate noise in the surrounding area. The availability for the crushing plant is assumed to be 50%.

17.2.2 Grinding

The fine mineral bin will store one day's worth (24 hours) of crushed magnesite to allow for continuous feed to the grinding plant. The fine mineral bin will feed a 2.8 m by 5.3 m ball mill at a nominal rate of 50.7 t/h. Ball discharge will be pumped to a cyclone, where a P_{80} split of 149-µm slurry will go to the overflow, with the underflow returned to the ball mill for additional grinding. It is assumed that the grinding circuit has a 275% recirculating rate. The ball mill cyclone will be fed a slurry of 50% solids, and will split an underflow of 65% solids and an overflow of 31% solids. Cyclone overflow will be transferred into the upgrading flotation circuit.

The grinding plant will operate 24 hours a day, on two 12-hour shifts, 365 days a year, with a 90% availability.

17.3 Upgrading

The ground magnesite material will initially be preconditioned in the rougher flotation conditioning tank with reagents DF250 and potassium amyl xanthate (PAX). The conditioned slurry will be pumped into the two rougher flotation cells for removal of any sulphides present in the ground magnesite slurry. The floated gangue sulphide material will be discharged to the tailings thickener, while the sulphide-free magnesite slurry will be transferred to the silicate flotation pre-conditioning thickener. Flocculant will be added to the magnesite slurry to aid in dewatering. The 31% solids slurry will be partially dewatered in the thickener, producing a 50% solids slurry in the underflow. Water in the overflow will be collected and pumped to the process water tank for reuse in upstream operations.

The thickened slurry will be pumped from the thickener underflow into a conditioning tank where Armac 1225 will be added at 250 g/t to aid in silicate removal. The slurry will be fed to a series of five conventional flotation cells where Armac 1225 will be stage added to assist in floating the silicate minerals from the magnesite stream. The floated silicate slurry will be discharged into the tailings thickener. The upgraded magnesite slurry will be collected in the magnesite filter feed tank.

The upgrading plant will operate 24 hours a day, on two 12-hour shifts, 365 days a year, with a 90% availability.

17.4 Dewatering

Magnesite slurry from the silicate flotation circuit will be pumped through a plate-and-frame pressure filter to further dewater the 62% solids slurry to a filter cake that will be approximately 75% solids. Filtrate from the filter will be collected in the process water tank for reuse in the plant.

17.5 CCM and DBM Production

The magnesite filter cake will be fed to a multiple hearth furnace where excess moisture will be driven off and the magnesite partially calcined to produce a CCM powder product. The multiple hearth furnace will be operated at temperatures ranging from 760°C to 900°C to decompose the magnesite into MgO and CO_2 according to the following reaction:

$$MgCO_3(s) + Heat \rightarrow MgO(s) + CO_2(g)$$

The CCM powder will be fed to a briquetting machine where the powder will be pressed to form cylindrical briquettes. These briquettes will either be fed to the vertical shaft kiln for further processing into DBM or bagged and sold as CCM briquette products. The CCM briquettes will be transferred to a vertical shaft furnace where they will be sintered into DBM product at temperatures ranging from 1,900°C to 2,200°C producing a product of 94.6% magnesium oxide (MgO) purity. The DBM briquettes will then be fed to a dry grinding mill and air classification to generate a final DBM powder product.

The off-gases from both the multiple hearth furnace and vertical shaft kiln will be collected and treated prior to release to the atmosphere

17.6 Tailings Preparation

Flocculant will be added to the tailings slurry where the solids are dewatered to produce a 60% solids in the thickener underflow. Tailings thickener underflow will be pumped through a vertical pressure filter where it will be further dewatered to approximately 92% solids. The dewatered solids will be trucked back to the mine site quarry for dry stacking in a tailings storage facility. Water recovered from both the thickener overflow and the pressure filter will be collected and pumped to the process water tank for reuse in the plant.

17.7 Recoveries

Recoveries for the flotation equipment, and the overall recovery, were based on the SGS Lakefield report titled *Geochemical & Mineral Beneficiation Report Driftwood Creek Magnesite Property*, dated August 2008. No new testwork has been performed to date. Nominal design was based on the higher recoveries for the East pit, which will be used as the first feed source. Design was based on the lower recovery to ensure that the flotation equipment was sized to handle larger rejected volumes of material. No losses are accounted for around the multiple hearth furnace or the vertical shaft furnace. Overall magnesite recovery is estimated to be 90%.

18 PROJECT INFRASTRUCTURE

The following section discusses Project infrastructure, including the mine site access and infrastructure, process plant, dry-stack tailings management facility (DS-TMF), and rock management facility (RMF).

18.1 Mine Site Access

The Project property is located approximately 210 km northwest of Cranbrook, BC, via Highway 95 and three Forestry Service Roads. From Cranbrook to Brisco, the route along Highway 95 is approximately 172 km. From Brisco to the mine site is approximately 38 km through FSR routes, Brisco road, Bugaboo Creek Road, and the Driftwood Creek Road. Good infrastructure currently exists in the form of paved highways, a CPR spur line (at Brisco, BC), and a major power line within 15 km of the property.

An alternate route is proposed for the Project from the Brisco to the mine site, as shown in Figure 18-1. This proposed access route will be approximately 10 km shorter for ROM haul trucks. This will require a road access study to be performed at the next level of study.

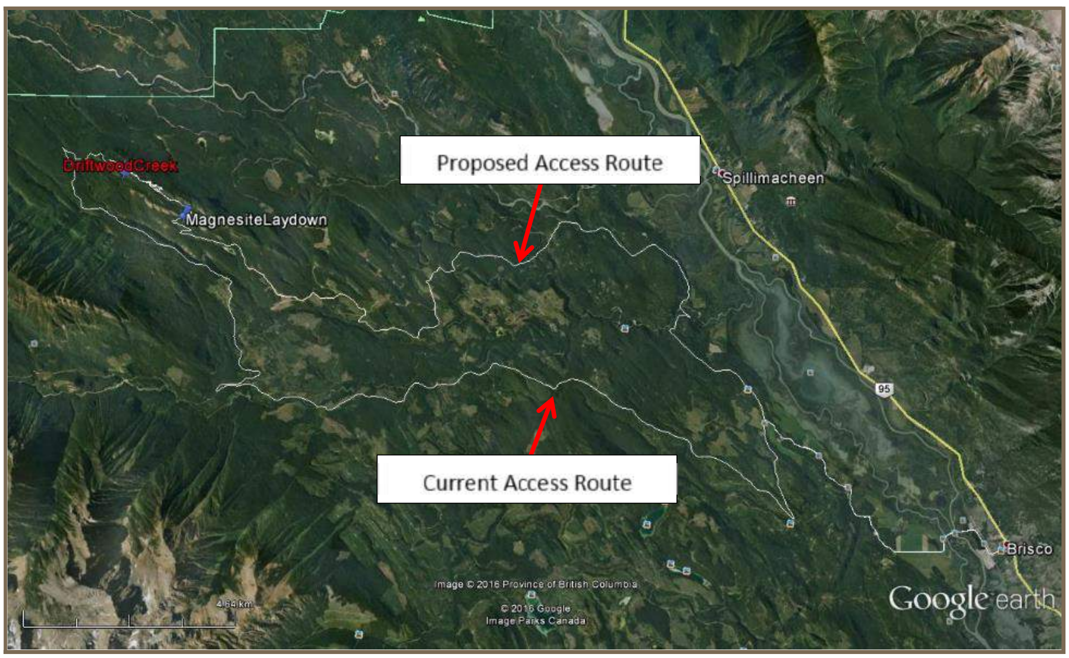


Figure 18-1: Proposed Mine Access Route

18.2 Mine Site Infrastructure

18.2.1 Administration Office

Proposed portable Atco trailers (approximately two or three trailers) will be set up on site to support office staff, a lunchroom, a medical room, and washrooms. No living accommodations will be provided on site.

18.2.2 Power Supply

The Project is proposing using a trailer-mounted, 40 kW genset to supply the portable Atco trailers and maintenance shop. No line power will be required to site, but a major power line is available at 15 km across the valley. Diesel-powered trailer-mounted light towers will be available at the excavation site and the ROM loading, ready-pile area. Since this a daytime operation, limited lighting will be required during operational hours.

18.2.3 Water Supply

The mine site will not require a main water supply. Due to the low water quantities required, all potable water will be delivered, and waste water removed off site. Note that the area has abundant access to a freshwater supply, and water rights are governed by the *Water Act*, which is administered by the Water Stewardship Division of the Ministry of Environment.

18.2.4 Maintenance Truck Shop

Contract mining will maintain a portable maintenance truck shop for minor repairs. Any major repairs will be completed off site.

18.2.5 Fuel Tanks

The Project is proposing a 20,000 L capacity fuel tank farm with dispensing systems. The fuel tank farm will follow the Health, Safety, and Reclamation Codes for Mines in British Columbia.

18.2.6 Mine Water Containment Facility

All mine runoff water will be managed by the proposed sediment pond containment facility near the DS-TMF. A geotechnical investigation will be required at the next level of study to determine the containment facility requirements.

18.2.7 ROM Ready-Pile / Stockpile

The Project is proposing to have a ready-pile for ROM resources. The ready-pile will contain approximately two to three days of ROM resource material available for loading onto highway trucks. No long-term stockpile will be managed on site.

18.2.8 Rail Spur at Brisco

A 40 tonne haul trucks is proposed for the Project to deliver ROM resources from the mine site to the plant site, located 210 km away, at Cranbrook. An existing rail spur located at Brisco could provide an alternative and cost-reducing method for hauling ROM resources to the plant site. It is recommended that a trade-off study on using a combination of truck and rail for the delivery of ROM mineralized material be completed at the next level of study.

18.3 Process Plant

The proposed plant facility will be located in the town of Cranbrook in the industrial park of the decommissioned Timbec Lumber site (1400 Industrial Road #1), as shown in Figure 18-5. This site meets the major requirement of having a natural gas supply line located on the premises. Other infrastructure is also available on site, including town water supply, sewage connections, power supply, and a rail sidings. The property is currently under a purchase option agreement with MGX. The property will have to undergo an environmental review at the next level of study.

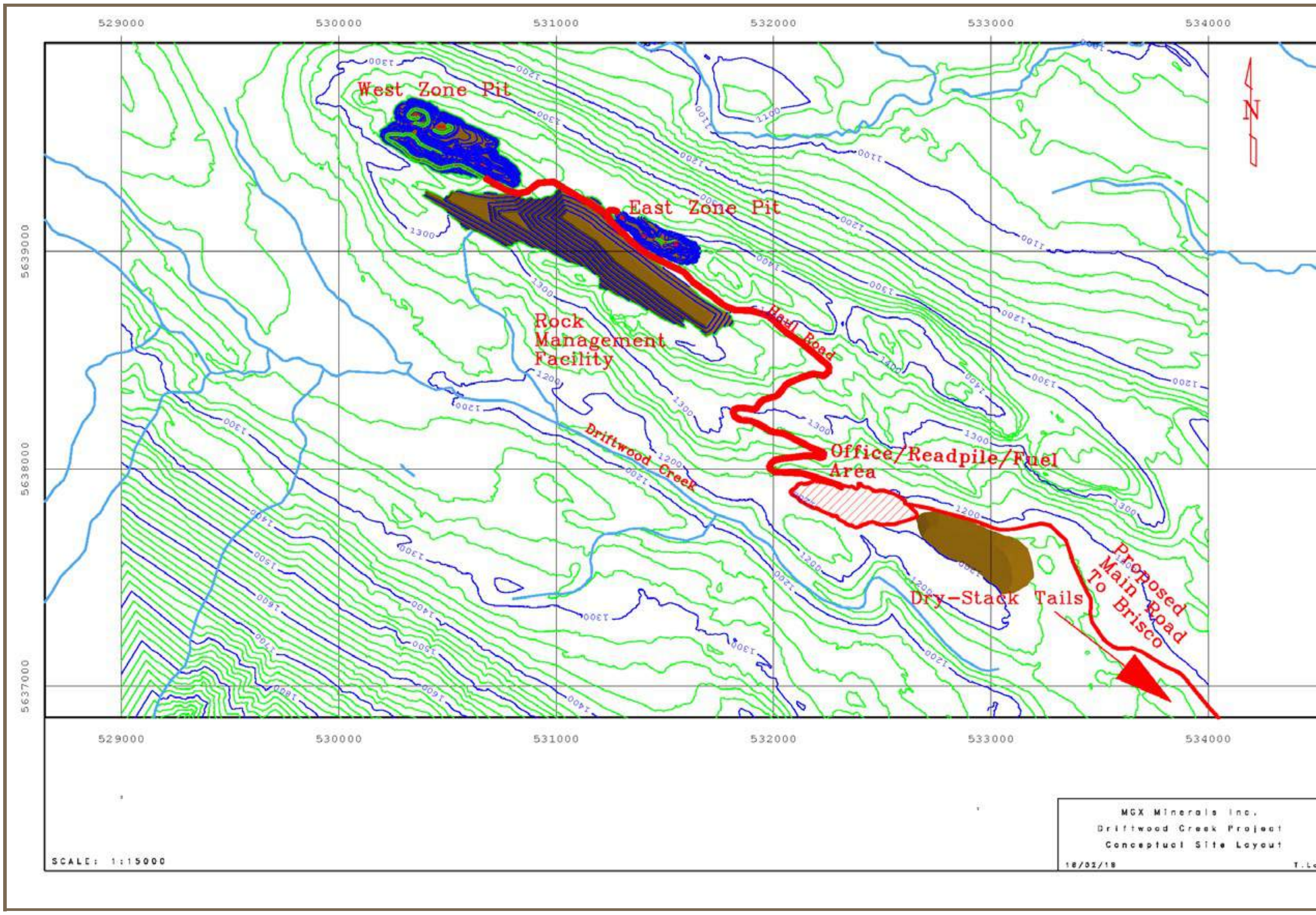
The plant processes comprise crushing, grinding, flotation upgrading, calcination, and sintering to produce a saleable DBM product. The plant will also have the ability to produce CCM as a separate product. The tailings thickener area and filter presses will be used to produce dry-stack tailings to be delivered on the return haul trip to the mine site near Brisco. Adequate warehouse and office space have been provided, as shown in the conceptual perspective layout and plan layout for the plant site in Figure 18-4 and Figure 18-5.

18.4 Dry-Stack Tailings Management Facility (DS-TMF)

The Project opted to incorporate a DS-TMF and eliminate the requirement for any subaqueous tailings. The DS-TMF will be located at the mine site, in close proximity to the ROM loading, ready-pile area, as shown in Figure 18-2. This will allow for simple access for the returning haul trucks to end dump, then proceed to ROM loading, reducing truck turnaround time.

The DS-TMF was design using industry standards with a 4(H):1(V) slope ratio using an area of approximately 0.11 km^2. Over the LOM, the DS-TMF will contain approximately 1.4 Mt of dry-stack material with about 8% to 12% moisture. All water runoff will be captured and managed by the mine water containment facility.

At the time of this study, the ABA information was not available; therefore, all dry-stack tailings material has been categorized as NAG material. A metal leaching / acid rock drainage (ML/ARD) test, along with a geotechnical investigation, will be required at the next level of study to determine the containment facility requirements.



Source: *AKF (2018)*

Figure 18-2: Conceptual Mine Site Layout

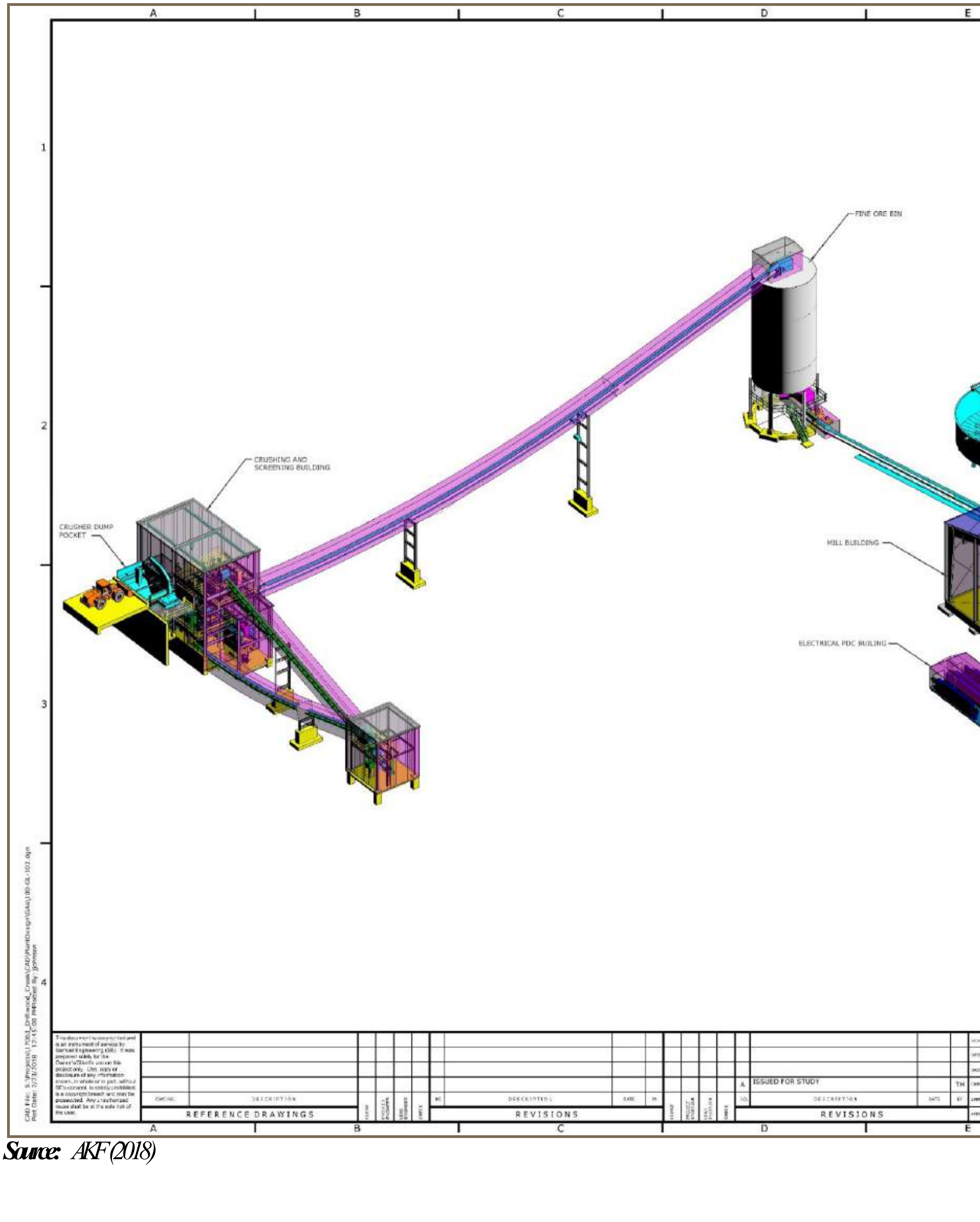
18.5 Rock Management Facility (RMF)

A single rock management facility (RMF) has been located and sized for approximately 19.174 Mt of mine rock, based on a 19-year LOM for this PEA study. The RMF proposed location will be downhill from the East Zone Pit and southeast of the West Zone Pit, and will facilitate end-dumping on 10 m lifts with a catchment berm of 6 m, to an overall design slope ratio of 2(H):1(V). The maximum design elevation for the RMF is 1,430 masl and it has a footprint area of approximately 0.32 km^2. All water runoff will be captured and managed by the mine water containment facility. The RMF design is shown in Section 16, Figure 16-3, and is also illustrated in Figure 18-3, conceptual mine site layout.

At the time of this study, the mine rock ABA information was not available; therefore, all mine rock has been categorized as NAG material. An ML/ARD test and geotechnical investigation will be required at the next level of study to determine the containment facility requirements.

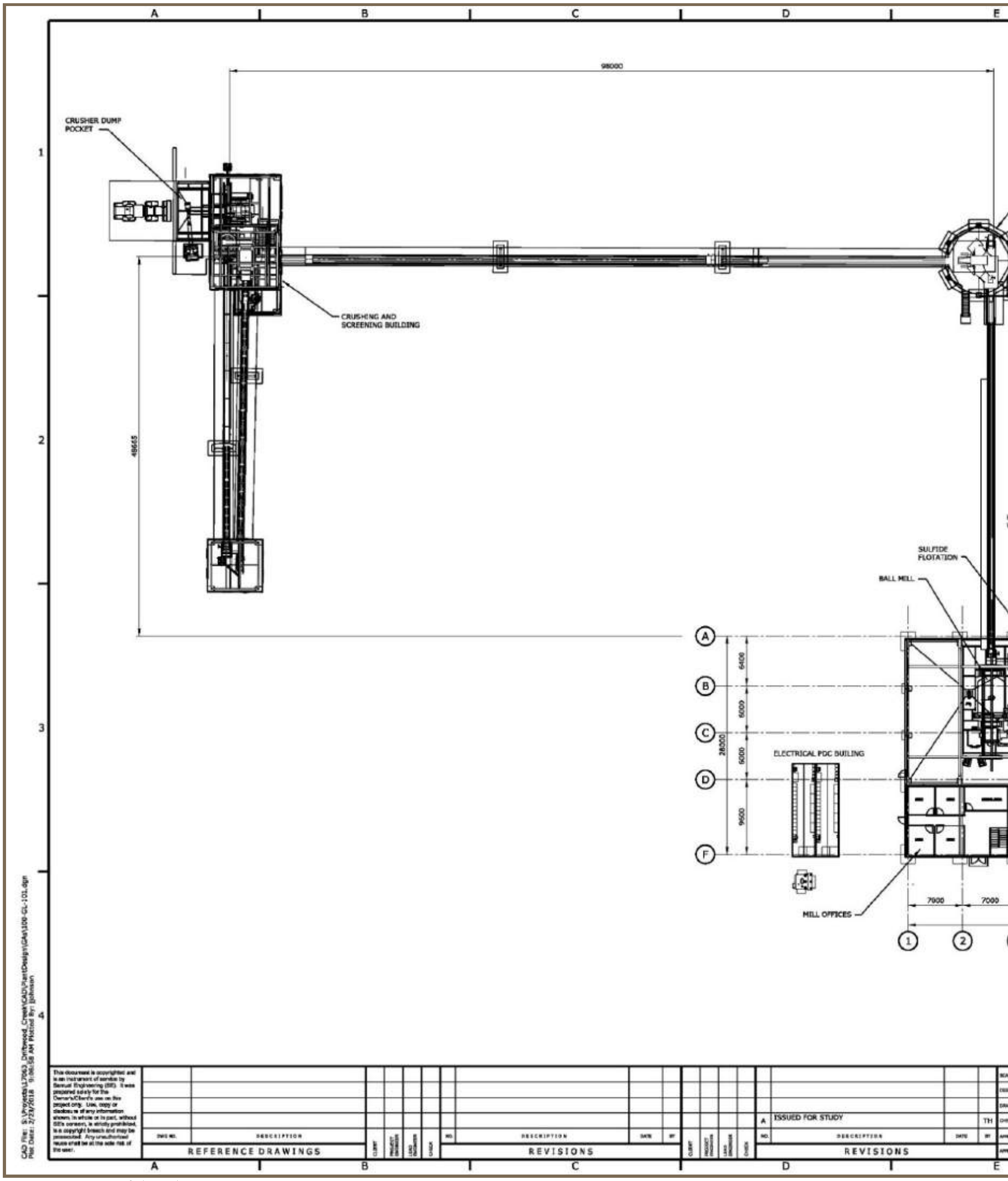
Source: *Timbec (unknown date)*

Figure 18-3: Proposed Plant Site Location in Cranbrook, BC



Source: AKF (2018)

Figure 18-4: Conceptual Site Layout



Source: Samuel (2018)

Figure 18-5: Conceptual Plan Layout

19 MARKET STUDIES AND CONTRACTS

The Project has the ability to produce both caustic calcined magnesia (CCM) and dead-burned magnesia (DBM) magnesium oxide (MgO). The process will start by taking magnesite concentrate after the flotation process and feeding a multiple-hearth furnace where excess moisture will be driven off, and the magnesite partially calcined to produce a CCM powder product. The multiple-hearth furnaces will be operated at temperatures ranging from 760°C to 900°C to decompose the magnesite into MgO and CO_2 according to the following reaction:

$$MgCO_3(s) + \text{Heat} \rightarrow MgO(s) + CO_2(g) \text{ (to atmosphere)}$$

The CCM (<5% CO_2) powder will be fed to a briquetting machine where the powder will be pressed to form cylindrical briquettes. These briquettes will either be fed to the vertical shaft kiln for further processing into DBM or bagged and sold as CCM briquette products. The CCM briquettes will be transferred to the vertical shaft furnace where they will be sintered into DBM product at temperatures ranging from 1,900°C to 2,200°C, producing a product of 94.6% MgO purity. The DBM (<0.5% CO_2) product will be bagged and transported to market for sale.

19.1 Summary of Information

The Project will potentially sell CCM and DBM, both of which are industrial mineral products without a known spot price. Only DBM pricing is evaluated in this PEA.

The nature of the product and the fact there are only two producers of DBM in North America, this creates a duopoly market and a challenge in tabulating information to determine DBM pricing for this Project. To validate pricing, for example, the website Alibaba.com indicated a DBM price range from $200/t to $1,200/t. Alternatively, data was tabulated from Refractories Window.

Based on AKF's evaluation, the suggested pricing for DBM is US$600/t; the DBM will be the basis for the Mineral Resource and economic analysis. Pricing is assumed to be freight-on-board (FOB) Driftwood Creek Project processing plant in Cranbrook, BC, Canada.

19.2 Market Studies

In 2017, world global capacity for MgO (all grades) was approximately 17 Mt. Global markets for MgO in 2016 were predominantly refractories, at 80% of demand, with the other 20% split between magnesium metal and chemicals (8%), agricultural (6%), manufacturing/pharma and food (3%), environmental (2%), and construction products (1%). The refractory market is expected to remain stable or slightly decrease (based on economic cycles), and the fastest growing markets are environmental and construction products expected to grow at >5% compound annual growth rate (CAGR) over the next five years.

Global magnesite production in 2016 was led by China, with 70% of world capacity, followed by Turkey at 10%, Russia at 5%, all other sources, including the US, were below these, in the range of 0.5% to 2%. Regarding reserves, China only has 20% of the world reserves, while Russia has 27%, and Greece 18%; thus, China is consuming its reserves faster than any other country. As a result, China is undergoing consolidation of its business, implementing greater regulations, and putting a greater emphasis on quality. This trend has prompted a further capacity expansion in other countries with new investments being made in Russia, Turkey, Australia, and more recently Canada. A number of less efficient, more polluting manufacturing facilities are being rationalized, and it is estimated that China's overall production decreased by 10% to 15% in 2017. This trend is expected to continue for the next few years.

In refractories, the steel industry used approximately 6.8 Mt to 7.0 Mt in 2016 with China accounting for half and the rest of Asia accounting for 25%. Due to a slowdown in China's economy, this consumption is expected to decrease in coming years. The same holds true for cement manufacturing that totals 2.1 Mt to 2.3 Mt globally. However, due to tougher air regulations in Western countries (including the US and Canada), more cement production is being moved to China and other Asian countries where regulations are significantly less stringent, resulting in an increased demand in China the in coming years. In North America in 2015-2016, refractories constituted 48% to 50% of the market for MgO, with agriculture accounting for 27% to 30%, and environmental uses (excluding oilfield demand) for 10% to 12%; consumption for construction product applications was minimal (<0.2%).

A significant market that is shifting overall magnesium ion supply and demand is that of globally-produced magnesium metal. In 2015, global consumption of the metal increased from 0.9 Mt to 1.0 Mt. While there are no direct routes from MgO to metal, Mg is being diverted to production of the metal, with the largest portion derived from magnesite by the Pidgeon process and a smaller portion obtained by electrolysis of magnesium chloride from brine/seawater sources. The largest producer of metal is China, with 78% to 80% of world production. Market for the metal is significantly above GDP growth, driven by environmental and economic pressures to increase the mileage of vehicles and aircraft. Magnesium metal is alloyed with aluminum to provide significantly lower weight, replacing steel and other aluminum alloys that weigh more. It is estimated that each automobile currently produced in North America contains approximately 12 lb of magnesium metal and consumption is increasing dramatically to meet the new goals of automobile fuel consumption per mile. It has been estimated that by 2020, approximately 250 lb of magnesium will replace 500 lb of steel, and 90 lb of magnesium will replace 120 lb of aluminum in each vehicle, resulting in a 15% weight reduction per vehicle, and significantly increasing demand. This application is becoming of greater interest to MgO manufacturers and will drive the balance of supply for this product as well as prevent any significant price drops in MgO as demand for this material has decreased in Canadian oilfield usage.

A large majority of the MgO produced worldwide is used in the same region it is produced due to the relative price of material vs. transportation costs. China is the largest exporter of MgO having exported 0.3 Mt of fused silica, 0.4 Mt of DBM, and 0.3 Mt CCM with approximately 70% going to North America, and the bulk of the remainder going to the European Union (EU). In North America, approximately 90% of the MgO produced was utilized in North America with approximately 10% going to export mainly to the EU. Recently, Brazil experienced lower demand in South and Central

America and entered the North American market by offering lower prices than all other producers. The impact on the North American market was 3% to 5% in 2015 and is not expected to exceed 6% over the next few years. EU producers (TIMAB) have also established a foothold in the US due to lowered domestic demand, but only account for 3% to 5% of the demand due to higher pricing driven by transportation costs. They are mainly competitive in the higher purity/value products where they compete with brine-derived material from known producers, or with product imported from Brazil and China. Indian demand has been growing. It significantly exceeds local production and is expected to continue growing at about 5% CAGR. It is difficult to establish actual consumption due to significant volumes of internal trading in the country, but the closest total MgO consumption estimates are 0.4 Mt to 0.5 Mt with the bulk going into refractories. Most supply to India comes from the EU, Russia, and Turkey, but a portion is a product actually exported from China through EU traders. Growing demand in India has justified investments in Turkey and Russia.

19.3 Magnesium Oxide Pricing

Since there are only two producers of DBM in North America, it is difficult to obtain data on pricing in North America and there is no published data available. For pricing, the must rely on global market pricing and import/export data that is not reliable in assessing prices and feedback from customers in the marketplace. In general, pricing for DBM in North America tends to be $50/Mt to $100/Mt higher compared to other countries due to higher pricing by domestic producers and significant freight costs for imported material to key markets (mid-west of the continent).

Prices for magnesium oxide in North America started a downward trend in the last decade, beginning during the 2007–2009 recession and hitting a low in 2011. They started seeing a recovery in 2012 with a spike in prices of about 15% to 20% and the increase remained steady at approximately GDP-equivalent growth through 2017. Several factors combined to cause a drop in pricing starting towards Q3 of 2015 and carrying on through to the present. Production from magnesite rock increased over the period while production from brines/seawater decreased due to more favourable economics.

In general, pricing were highest for fused magnesia and DBM and lower for CCM, although skewed by exports from China, the largest global producer of magnesia. It accounts for 65% to 70% of world magnesite production, compared with North America’s overall 3% contribution to global DBM production and 5% to 6% contribution to the CCM market. DBM production in North America is restricted mainly to Martin Marietta, with some production coming from Baymag; thus, the North American DBM market is short of local production, resulting in significant import volume from China and Brazil. This trend has been reversed in Q4 2017 with Chinese imports dropping prices increasing.

In 2015, several factors influenced pricing and instigated downward pressures reflected in Q3 2015 that continue to the present:

- Fall in oil prices that lowered production costs (energy costs of production are approximately 35% to 40%), as well as freight to destination.

- BayMag in Canada lost significant sales to the oil sands market due to an energy crisis in the market (oil production from oil sands now higher than market price). It is estimated that BayMag sales in this market (used mainly to precipitate silica from water effluents, allowing reuse) were around 40,000 Mt prior to the oil crisis. This followed Baymag's 70,000 Mt expansion three years earlier, driven by this market and freeing up volume for other applications.
- Decreasing demand for DBM and fused magnesia in key markets—refractories, steel production with only a 10% to 15% increase in CCM demand for environmental and animal feed applications.
- Investments in production in Australia, Turkey, Russia, Brazil, and more recently Canada.
- Changing economic and socio/political factors in China:
  - Economic slowdown is affecting internal demands for DBM and fused silica, mostly—CCM is less affected as environmental and building products demand is increasing;
  - Overcapacity;
  - Rationalization of magnesia industry—shutting down less efficient production;
  - Mixed results in exports—18% decreased for fused magnesia and DBM, 10% increase in CCM; and
  - Container freight cost experience significant drop (50% to 60%).

Net results are that pricing appears to have dropped 5% to 10% for CCM in 2015 and will continue a slow decline for the next 12 months due to increased imports from China, Brazil, and Europe, and because of the decrease in market demand from oil sands in Canada. However, recovery in the markets and decreased imports from China due to lower availability resulted in an upward pricing trend in 2017 that is expected to continue over the next few years driven by growth in cement markets (construction products, concrete admixtures). Environmental, animal feed, agriculture, and waste treatment will continue to expand along with the economy and a recovery of the Canadian oilfield market. In fact, pricing from China significantly increased in Q4 17 resulting in DBM pricing no longer being competitive in the North American market and losing significant market share.

Downstream product magnesium sulphates (Epsom salts), produced from CCM are also increasing steadily at a rate of more than 25% CAGR. Local North American producers will fight to maintain market share/volume with Premier being in the best cost position to do so.

19.3.1 Basis for Pricing

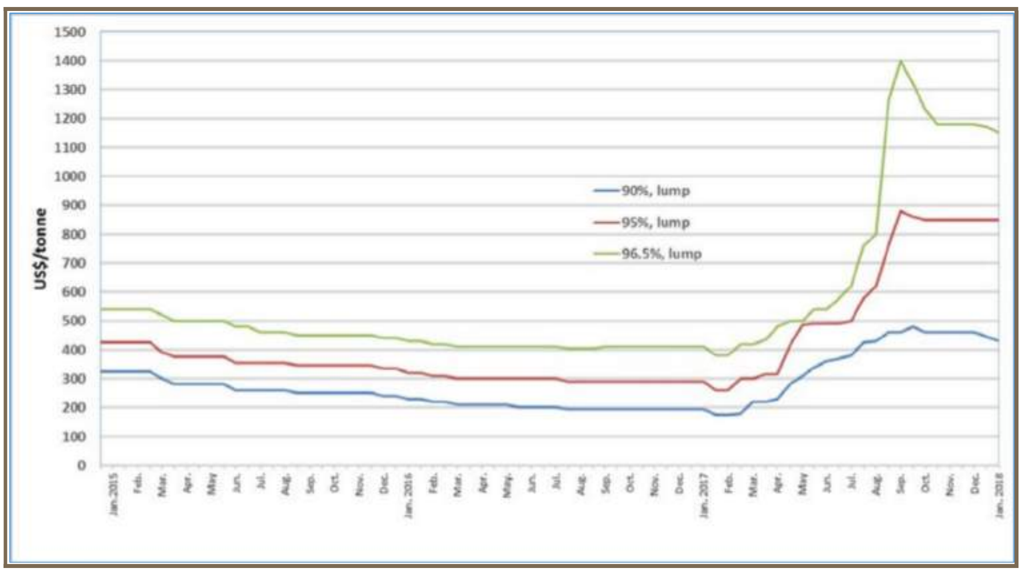
To determine "reasonable prospects of eventual economic extraction." The basis used for pricing, taken from the "2012 Mining Report" prepared by the British Columbia Securities Commission and dated January 2013, is noted as, "the lesser of the three-year moving average and current spot price." Since there is no known market spot price and due to the nature of a duopoly market for DBM, data collected from Refractories Window was used in the evaluation of the three-year moving average for the 95% lump data.

The volatility over the past year has been significant as shown in the data collected from Refractories Window. If using the three-year moving option and current price against the Refractories Windows data as Illustrated in Figure 19-1, the DBM prices would be US$420/t, versus their current price of US$850/t.

The data used from Refractory Window is considered FOB China, and additional shipping and rail costs will be applied to the consumer. These could range from $80/t to $160/t depending on the part of the world to which the product is being shipped. Since this Project can provide magnesia in North America, that premium can be applied back to the Refractories Window data. Using $80 as a conservative number, the DBM price will be adjusted up to $500 from $420.

In addition to the DBM price, making a DBM powder instead of salable DBM briquettes adds a premium to the DBM product. In discussion with local buyers and furnace manufactures, the premiums on a DBM powder can range from $100/t to $200/t. Applying the powered premium of $100/t as a conservative number, the DBM price to be used for this Project's Mineral Resource and economic analysis is US$600/t.

For the next level of study, it is recommended that an independent research group complete market research and evaluate demands, pricing, and supply sources to develop a pricing report to be used for the Project.



Source: *Refractories Window (2018)*

Figure 19-1: DBM Lump Price FOB China from Jan 2015 to Jan 2018

19.4 Contracts

The Project does not have any material contracts relating to development, including mining, concentrating, smelting, refining, transportation, handling, sales and hedging, or forward sales contracts or arrangements. Due to the nature of a PEA and the extended permitting and construction times the Project is likely to face, AKF cannot advise on what material contracts will be signed in the future.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Related Information

The property is located approximately 53 km southeast of Golden, BC, and approximately 210 km northwest of Cranbrook, BC. Access is by FSR from either Brisco or Spillimacheen. Local infrastructure is the paved Highway 95, with a CPR spur nearby. The property consists of seven contiguous mineral tenures, with a total area of 835.44 ha (2,064.42 acres).

The magnesite deposit is described as white-buff to cream-coloured, very fine-grained to very coarse-grained (coarser grained near faults/conduits), containing irregular concentrations of siliceous veinlets, laminae, or blebs of up to 2 cm thick.

The Project magnesite occurrence is classified as a sparry magnesite deposit (E09) by the BC MEM (Simandl and Hancock, 1998). This deposit type is characterized by stratabound (and typically stratiform), lens-shaped zones of coarse-grained magnesite, mainly occurring in carbonates, but also observed in sandstones or other clastic sediments.

The BC Government required that notifications for the 2017 geotechnical drilling be provided to referral officers at the Shuswap First Nation, Ktunaxa First Nation, Neskonlith Indian Band, and Adams Lake Band (3/14/2017 email communication with R. Fraser, BC Ministry of Forests, Lands, Natural Resource Operations, & Rural Development (FLNRORD)). These four groups either include the Driftwood Project area within their traditional areas or are near the Project area. Notifications were provided to referral officers for these organizations, along with personal correspondence with the Shuswap First Nation band chief. No concerns were noted during the consultation period, and the Mineral Exploration Permit was approved on July 6, 2017.

20.2 Environmental Studies

MGX has not conducted environmental baseline studies as yet; however, in 2017, eleven geotechnical holes were established in the area, with monitoring wells for water quality monitoring, and several (DD-BGC17-07, -17-08, and -17-10) fitted with groundwater vibrating wire piezometer (VWP) data loggers for assessment of water levels and water quality, to improve understanding of the hydrogeology in the Project area. In addition, samples of core from each hole were stored, and will be used to conduct the necessary acid rock drainage/metal leaching (ARD/ML) assessments.

There are presently no known environmental issues that could materially impact the Project's ability to extract the Mineral Resources and process material. The only known environmental liabilities are associated with the exploration site activities and access roads, and are covered under a reclamation security with the BC Government. Reclamation of surface disturbances and any resultant contamination is required as part of the exploration permit.

20.3 Exploration Permitting Requirements

Activities to date have focused on mineral exploration and bulk sampling under BC Ministry of Energy, Mines, and Petroleum Resources (BC MEMPR) Permit MX-5-644. In 2016, a bulk sample was taken from a zone of 120 m^2, and 25 drill holes were drilled and sampled, to obtain approximately 100 tonnes of magnesite as a bulk sample for detailed metallurgical testwork. SG testing was also undertaken.

As noted above, MGX conducted an infill-drilling program aligned with recommendations from the maiden resource report.

In 2017, 11 diamond drill holes (DDH) were drilled and the rock cored for the purpose of a geotechnical assessment by BGC Geotechnical Engineering, which will perform analysis of the materials. During the program, several holes were set up as monitoring wells, and some had VWP data loggers installed.

20.4 Development Permitting Requirements

The development and permitting process for industrial minerals is well established in British Columbia and in Canada. In each Canadian jurisdiction, the process consists of a two-tiered system, whereby the proposed Project undergoes a screening process to determine if an Environmental Assessment (EA) is necessary. The EA process provides a mechanism for reviewing major projects to assess their potential impacts. Although exceptions do occur, the EA phase typically involves departments of both the federal and provincial governments. Following a successful EA, the operation undergoes a construction and operating licensing/permitting phase. Both federal and provincial Ministries regulate projects through all phases (construction, operation, closure, and post-closure).

20.4.1 Environmental Assessment Process

Provincial Environmental Assessment Process

Major mines and expansions in BC (including large-scale industrial mineral and aggregate mines such as Driftwood) typically require an EA certificate. In BC, the Environmental Assessment Office (EAO) manages the assessment of proposed major projects under the BC *Environmental Assessment Act* (BC EAA). The assessment process examines projects for potentially adverse environmental, economic, social, heritage, or health effects that may occur during the life cycle of a project.

There are three ways an industrial mineral project may require review by Government under the BC EAA:

1. BC *Reviewable Projects Regulation* (RPR, 2002) provides for an industrial mining project to be automatically reviewable if it meets the following thresholds:

   1. A new quarry facility or other operation that:

      a. involves the removal of construction stone or industrial minerals or both;

      b. is regulated as a mine under the *Mines Act*; and

c. during operations will have a production capacity of >250,000 t/a of quarried product.

2. Ministerial Designation by the Minister of Environment, who has the authority to order the review of a project that is not immediately reviewable under the Regulation.
3. Proponent opt-in, whereby a proponent may request that the EAO consider designating its project as a reviewable project, and the EAO concurs and orders such a designation.

Federal Environmental Assessment Process

In the spring of 2012, the *Canadian Environmental Assessment Act* (CEAA, 2012) was amended. Two significant results of these amendments were the redefinition of what triggers a federal EA, and the introduction of legislated time periods within a federal EA if it is required.

With respect to the Project, there are two main methods of triggering a federal EA under CEAA 2012:

4. A proposed project will require an EA if the project is described in the *Regulations Designating Physical Activities* (2012); or
5. A project may require an EA if, in the opinion of the federal Minister of Environment, carrying out the project may cause adverse environmental effects, or that public concerns related to those effects warrant further review.

With respect to number 1 above, the *Regulations Designating Physical Activities* (2012) states:

> *15. The construction, operation, decommissioning and abandonment of a) a metal mine, other than a gold mine, with an ore production capacity of 3,000 t/d or more.*

The industrial mineral category does not exist in CEAA; therefore, the metal mine criteria would apply.

Once a federal assessment is triggered, the Agency then determines what type of EA the project will required. There are two types of EAs conducted under CEAA 2012: an EA by responsible authority (Standard EA), and an EA by a review panel. Both types of assessments can be conducted by the federal government alone, or in conjunction with another jurisdiction. The responsible authority in the case of base and precious metals mining is the Agency.

20.4.2 Environmental Assessment Requirements of the Project

In BC, agreements are in place with the federal government that require the EA to be a single cooperative process. The BC EAO and the Agency have signed a Memorandum of Understanding (MoU) that establishes expectations, roles, and procedures for implementing the substitution of EAs in BC. This is a new tool enabled by CEAA 2012. Under substitution, where both federal and provincial EAs are required, there can be a single review process (the provincial one) and two decisions (federal and provincial).

Provincial Requirements

The Project would automatically trigger an EA under RPR 2002, as it is a new industrial mineral mine with 440,000 tonnes of magnesite material production per year.

There are three stages in an EA: pre-application, application review, and the decision stage. The general steps required are illustrated in Figure 20-1. As outlined below, the process to obtain a certificate may take 30 months to complete, but it may take more or less time depending on a variety of circumstances, including the technical complexity of the project and consultation requirements.

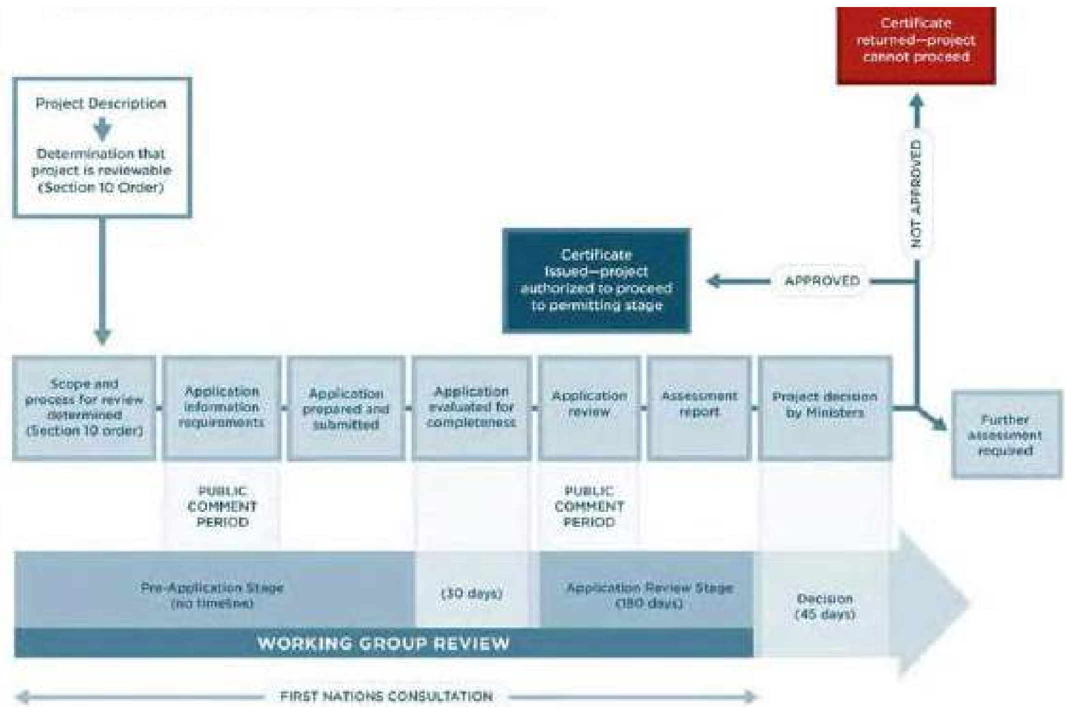


Figure 20-1: Environmental Assessment Process

Pre-application Stage: it is assumed it will take between 18 and 24 months to gather the required environmental baseline information needed at the pre-application stage, evaluate the data, and prepare the EA application for the Project.

Application Review Stage: the application review stage is governed by legislated timelines, and may take up to one month in screening the application to ensure it contains the required information, and then six months in reviewing the application once it has been accepted by the EAO. Additional time will be required for the proponent to address any deficiencies in the application and to respond to comments and information requests from reviewers.

Decision Stage: the *Prescribed Time Limits Regulation* sets down a time limit of 45 days from the date of referral to the Minister for them to make a decision on whether or not to certify the project. If the Minister decides that more time is needed, an order may be issued to extend the time limit. The Minister may also decide that further assessment is required.

Federal Requirements

The Project as it is currently defined will likely require a federal EA in accordance with CEAA 2012. It is anticipated that a Standard EA will be required, with the possibility of this EA being escalated to a panel review. Once initiated, completion of a Standard EA will require approximately 24 to 30 months, unless substitution is enabled, upon which the process would then follow the BC timelines.

After the application review stage, the Minister may issue an EA certificate allowing the proposed project to proceed with obtaining permits, licenses, authorizations, and approvals.

20.4.3 Provincial Environmental Permitting Process

Following a successful EA, the Project will be required to obtain a number of provincial licences and permits. The process can be initiated at the commencement of the EA process by requesting concurrent permitting, or would be initiated during the final stages of the EA. Typically permitting processes with BC's Major Mine Permitting Office (MMPO) begin after the EA certificate is issued.

In BC, proposed major mines require approval under the *Mines Act* as per part 10.1.2 of the *Health, Safety, and Reclamation Code for Mines in British Columbia*. The MMPO coordinates major mine authorizations across governments and brings clear accountability to both industry and government to ensure timely, high-quality applications and enduring decisions are made with respect to permitting major mine projects.

The *Mines Act* permitting process is closely integrated with the *Environmental Management Act* (EMA) permitting process for major mines, and includes geotechnical design and reclamation and closure plans. The Project will follow the MEMPR and Ministry of Environment guidance document for joint application information requirements for *Mines Act* and EMA permits. In addition to these permits, various other authorizations are required for major mining projects. Depending on the complexity of the proposal, applications are reviewed by either the relevant regional Mine Development Review Committee (MDRC) led by MEMPR, or project-specific Mine Review Committees (MRCs) coordinated by MMPO.

The main provincial permits that would be required for the construction and operation of the Project include a *Mines Act* permit for the mine plan, tailings storage facility, and reclamation program, and EMA permit(s) authorizing any liquid effluent discharges, air emissions, sewage, or solid refuse disposal. In addition, ancillary licences may be required from FLNRORD, such as water licenses for water diversions, or a Licence to Cut to clear land prior to construction.

Federal authorizations may be required for the Project, such as granting authority to manufacture or store explosives under the *Explosives Act*, or for any work that has the potential to impact waters defined as fish habitat under the *Fisheries Act*.

20.5 Social and Community

Local indigenous consultation was initiated in the 2017 Notice of Work process with notifications to four bands: the Shuswap First Nation, Ktunaxa First Nation, Neskonlith Indian Band, and Adams Lake Band. An introductory letter regarding Driftwood was also sent to the Shuswap First Nation chief and council.

Public consultation with these local bands, along with sport/environmental organizations and communities near the Project, will be conducted through the EA process to consider their concerns and incorporate them into Project operations and design. All discussions, and responses by MGX, will be recorded and provided in the final consultation report to the EAO.

20.6 Operating and Post-Closure Requirements and Plans

20.6.1 Environment Management Plans

Given the proposed development plans outlined in the document, the following key environment management plans (EMPs) for operations will be developed:

- Water Management Plan;
- Sediment and Erosion Control Management Plan;
- Tailings Management Plan;
- Dust Management Plan;
- Wildlife Management Plan; and
- Chance Find Procedure.

Other plans will be developed based on the EA Certificate conditions and/or BC permit requirements.

20.6.2 Conceptual Decommissioning and Reclamation Plan

Conceptually, the plan for the closure of the facility will consist of the following main components:

- Reclamation objectives, including closure design criteria;
- The progressive reclamation of the site during the life of the operation;
- The removal or stabilization of any structures or workings;
- The design of tailings and waste rock disposal areas;
- The reclamation and revegetation of surface disturbances wherever practicable;
- Methods for protection of water resources;
- A temporary closure plan in the case of work stoppage;

- A conceptual end of mine plan closure program and cost estimate of the work required to close and reclaim the mine; and
- A plan for ongoing and post-closure monitoring and reporting at the site.

The mine facilities will be decommissioned, salvaged, and removed from the site, with any hazardous wastes disposed of at approved facilities. The concrete slabs will be covered with material, all roads recontoured, and the area revegetated.

Waste rock dumps will be resloped to angles appropriate for overburden placement and subsequent seeding/fertilizing to establish sustainable vegetation. The dewatered tailings dry-stack will be covered with overburden and revegetated. The waste material is not expected to be acid-generating due to the basic host rock, and therefore capping with compactable materials to reduce water ingress will not be necessary. In all cases, sediment control will be considered and included in the design of mining facilities and in the reclamation plan. Dependent upon the approved reclamation objectives, planting of native shrub and tree species would also be conducted.

The water management plan will ensure ditching of surface water will be directed such that erosion will not occur on reclaimed surfaces, and likely directed overall to the open pit for containment. A closure spillway will be designed and constructed from the open pit to an appropriate watercourse.

A post-closure monitoring plan will be designed and established to monitor the facility discharges and reclamation success until closure objectives have been met and the property is eligible to be returned to the Crown.

20.7 Environmental and Social Issues

Development of the Project will be subject to an assessment of environmental and socio-economic impacts, including cumulative effects. The Project will entail the development of an open pit, waste rock storage facility, and dry-stack tailings storage facility. Milling will take place off site. The complexity of the EA and permitting of the facilities will be dependent on the siting of facilities, waste characterization, and the engagement of the regulators, local community, and First Nations. A typical schedule for this type of EA is presented in Figure 20-2.

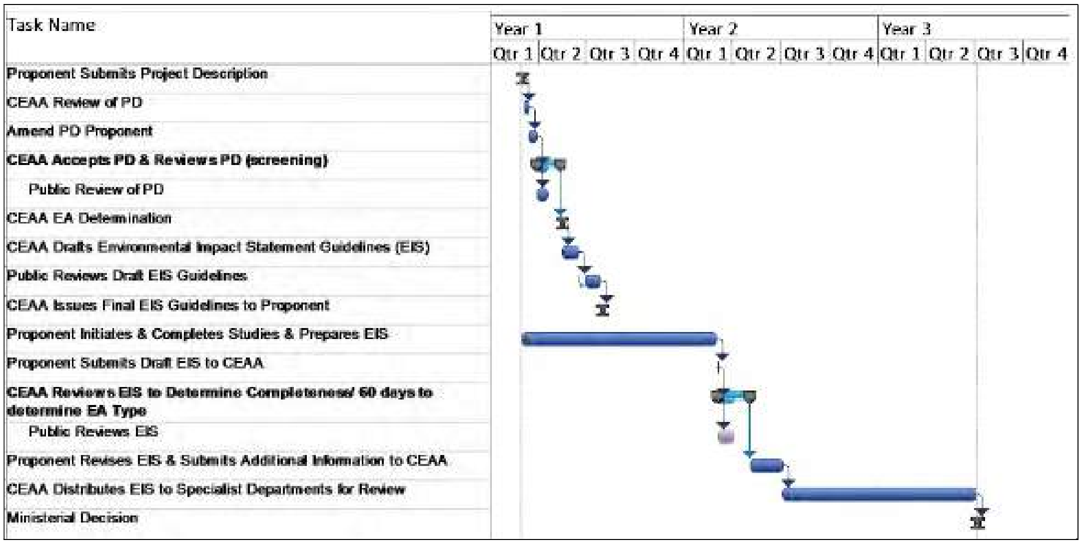


Figure 20-2: Schedule

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs Summary Estimate

The capital expenditure (CAPEX) cost estimate for the Project is based on a combination of first principles build-up, experience, references from relatively similar projects, budgetary quotes, and factors appropriate with a PEA level of study.

The CAPEX estimate includes the costs required to develop, sustain, and close the operation for a planned 19 year LOM. The construction schedule is based on an approximate two-year build period. The intended accuracy of this estimate is ±20% for the mine, whereas ±25% is used for the process plant, which is suitable for project evaluation, but not for engineering, procurement, and construction management (EPCM) or financing.

All capital and operating cost estimates are reported in Canadian dollars (C$) unless stated otherwise.

The CAPEX estimate summary is shown in Table 21-1.

Table 21-1: Capital Costs Estimate

Description	Pre-Production ($M)	Sustaining/Closure ($M)	LOM ($M)
EA, Permitting, Basic Engineering	6.8	0	6.8
Capitalized Stripping – Rock	0.5	0	0.5
Capitalized Stripping – Organics	0.3	0	0.3
Mine Site and Development	1.5	0	1.5
Plant Site (Timbec Site in Cranbrook, BC)	3.8	0	3.8
Process Plant	37.7	0.4	38.1
MgO Calcination	108.7	0.4	109.1
EPCM	14.4	0	14.4
Indirects	15.9	0	15.9
Reclamation/Closure	0.0	2.5	2.5
Owner's Costs	7.1	0	7.1
Subtotal	196.6	*3.3*	199.9
Contingency (20%)	39.3	0.7	40.0
Total Capital Costs	235.9	3.9	239.8

Source: *AKF (2018)*

Notes: *Rounding as required by reporting guidelines may result in apparent summation differences between dollar values. EA = Environmental Assessment; EPCM = Engineering, Procurement, Construction Management; $M = million dollars; LOM = life-of-mine; % = percent*

21.2 Mine Capital Costs Estimate

The mining capital expenditures used in this evaluation are estimated at $1.5 million without contingency applied. All mining (drilling, blasting, loading, and hauling) and ore loading is assumed to be contracted, which also includes a portable maintenance shop, which will carry no capital cost in the estimate. The basis of the estimate only includes portable trailer offices, emergency equipment, fuel tank farm, portable genset, haul road construction, main access road upgrade to the mine, and stripping organics for the mine rock management facility and the dry-stack tailings management facility (DS-TMF).

21.2.1 Environmental Assessment and Basic Engineering Capital Costs Estimate

The environmental assessment (EA) and basic engineering cost is estimated at $6.8 million without contingency. The basis of the estimate is from previous estimates of similar quarries in BC.

21.2.2 Open Pit Stripping Capital Costs Estimate

Estimated non-resource material stripping cost is approximately $530,000, based on 60,000 tonnes of pre-strip material at $8.82/t mined. It was assumed that only a small amount would be required for organics removal for the open pit, estimated at approximately $250,000. Both estimates do not include contingency costs.

21.2.3 Reclamation and Closure Costs Estimate

The reclamation and closure costs estimated for the plant and mine are based on similar quarries' costs in British Columbia, and are estimated at $2.5 million without contingency.

21.3 Process Plant Capital Costs Estimate

The MgO process plant and capital costs estimate follows the American Association of Cost Engineers (AACE) International Recommended Practice No. 47R-11, *Cost Estimate Classification Matrix for Mining and Mineral Processing Industries, Rev. July 6, 2012*. The maturity of Project definition ranges from 0% to 2% of complete Project definition. The capital cost addresses the development, construction, and startup of a DBM plant to be located at an industrial park site in Cranbrook, BC, Canada. Mineralized material will be transported approximately 210 km from the Driftwood Creek mine site by ROM-haul highway trucks to the plant in Cranbrook.

The process plant uses a combination of conventional crush, grind, and flotation upgrading, in conjunction with specialized process equipment, to calcine and sinter the material, producing either or both a caustic calcined magnesia (CCM) and/or dead-burned magnesia (DBM) from magnesite ($MgCO_3$). The total estimated cost to design, procure, construct, and start up the process plant and related infrastructure facilities described in this report is $176.7 million, including 5% contingency. The plant contingency was split out, with 5% applied during the estimation, and 20% applied in the cash flow model, for a total of 25% for the process plant capital cost estimation.

Table 21-2: Process Plant Capital Costs Summary Estimate

	Cost ($M)
Site Preparation and Development	2.2
Process Plant	128.5
Filtered Tailings	2.4
Site Infrastructure	6.8
Subtotal Directs	139.8
Engineering/Startup	27.3
Consultants	0.3
Freight/Insurance	1.2
Subtotal Indirects	28.8
Contingency*	8.0
Total	176.7

Source: *Samuel (2018)*

Notes: **Costs for the DS-TMF at the mine are not included. *Excludes the 20% cash flow contingency*
Rounding as required by reporting guidelines may result in apparent summation differences between dollar values. $M = million dollars.

21.3.1 Process Plant Capital Sustaining Costs Estimate

Sustaining costs for the process plant include periodic brick relining for the multiple hearth furnaces ($355,000) and occasional replacement of plant mobile equipment ($468,000). Total LOM sustaining cost is $823,000, which includes contingency.

21.3.2 Owner's Costs

Owner's costs of approximately $7.066 million without contingency are estimated at 4% of the total process plant capital cost of $176.7 million.

21.3.3 Exchange Rate

The exchange rate for Driftwood is US$0.77:C$1, and is based on a three year trailing average from the Bank of Canada as of January 2018.

21.4 Operating Costs Summary Estimate

The operating cost estimates are based on a combination of first principles build-up, project references, quotes, and applied factors appropriate for this PEA study.

These costs include mining by a contractor, transportation of magnesite ore from the mine to the plant in Cranbrook, processing, and general and administrative (G&A), as shown in Table 21-3.

Table 21-3: Operating Costs Summary

	Cost		
	$/t Processed	LOM $M	$M/a
Mining	30.30	237.7	13.2
Transport from Mine to Plant	43.95	344.7	19.2
Processing + G&A	62.06	486.8	27.0
Total	**136.31**	**1,069.1**	**59.4**

Source: *AKF (2018)*

Notes: *Mining cost is based on $8.82/t mined. All operating costs have been adjusted for fuel costs using $0.98/L. G&A = general and administrative; LOM = life-of-mine; $M/a = million dollars per annum; $/t = dollars per tonne.*

21.4.1 Mining Operating Costs Estimate

All mining (drilling, blasting, loading, and hauling) and ROM loading is assumed to be contracted. The mining and ROM-loading equipment are only used to estimate fuel quantities. Mobilization charges and diesel costs are included in the mine operating costs.

Operating costs were based on the following criteria:

- Mining operating costs have been estimated based on a contractor quote during 18 years of operation at $8.37/t mined. This also includes mobilization and demobilization of contractor equipment.
- Fuel costs are estimated at $0.98/L or $0.11/t mined.
- Supervision and technical support is included at $0.34/t mined.
- Mine operations will be conducted on one 12-hour shift per day, 360 days per year, on a work schedule of four days on/four days off.
- Costs exclude pre-stripping, which has been capitalized as described in Section 21.1 of this report.

21.4.2 Transporting Resource from the Mine to the Plant Operating Costs Estimate

The transportation of the resource material from the mine site to the plant will be completed by a ROM-haul contractor using 40-tonne highway trucks. The basis of the estimate uses 40-tonne highway trucks at a cost of $190/h all-in, and a seven-hour cycle time. This also includes the contractor loading time of $295/h for a 2.4 m^3 excavator, operating 24 hours per day, on two 12-hour shifts per day.

21.4.3 Processing Operating Costs Estimate

The processing facilities will consist of a comminution circuit, flotation, DBM production, product handling, and tailings thickening/filtration. The comminution circuit comprises primary and secondary crushing and a closed-circuit ball mill. The comminution circuit is followed by reverse magnesite flotation, thickening, and filtration circuits, multiple hearth furnace, vertical shaft furnace, tailings thickening and filtration, and product bagging for shipment. Filtered tailings will be returned to the mine location via haul truck and disposed of in the DS-TMF.

Operating costs were based on the following criteria:

- Plant to process approximately 1,200 t/d of resource material;
- Plant operations will run continuously, with two 12-hour shifts, seven days per week, 365 days per year;
- Crushing and product load out will be conducted during the 12-hour day shift only;
- Power cost, demand charge, and daily rate charge were provided by BC Hydro. The power cost used is $0.088/kWh; and
- Natural gas cost was provided by Fortis BC.

Table 21-4: Processing Operating Costs Estimate

	Annual Cost ($)	Cost ($/t)
Salaried Labour	1,345,000	3.36
Operating Labour	4,926,000	12.32
Technicians and Assayers	665,000	1.66
Maintenance Labour	664,000	1.66
Site Plant Electrical Power	3,270,746	8.18
Fuel (Natural Gas)	10,269,480	25.67
Reagents	468,900	1.17
Grinding Media	420,652	1.05
Product Loadout (Super Sacks)	1,813,700	4.53
Maintenance Supplies (5% of Installed Equipment Cost)	228,622	0.57
Miscellaneous Operating Expenditure (1% of Process Operating Cost)	240,721	0.60
Processing Total	**24,312,840**	**60.78**
Diesel Fuel Costs	**556,986**	**1.28**
Total Processing Cost	**24,869,826**	**62.06**

Source: *Samuel (2018)*

Note: *Cost estimate for the DS-TMF at the mine is not included. $/t = dollars per tonne; % = percent*

22 ECONOMIC ANALYSIS

This economic analysis is based on work performed by Samuel and AKF, and meets the guidelines for a PEA study, with a ±25% level of accuracy.

An economic model was developed to estimate the cash flow and sensitivities of the entire Project life. Pre-tax and post-tax results were evaluated as part of the economics, and are only an estimate.

The net present value (NPV) and internal rate of return (IRR) are measured from construction in Year -2, adding the EA and permitting cost from Year -3. Corporate sunk costs, including costs for exploration and technical studies, are not included in the cash flow model. IRR is assumed 100% equity financing.

Sensitivity analyses were also run over the entire Project life, varying metal prices, exchange rates, operating costs, and capital costs to determine Project value drivers.

22.1 Summary of Economic Analysis

Table 22-1 summarizes the results of the economic analysis for the 19-year project, with both pre- and post-tax results shown.

Table 22-1: Summary of Pre- and Post-Tax Results

Description	Value	Unit
MgO DBM Price	600	US$/t
Resources to the Plant	7,843	Kt
MgO Grade	43.27	%
MgO Produced	3,055	Kt
Non-Resource Material Mined	19,174	Kt
Strip Ratio	2.44	nr:r
Net Operating Income (EBITDA)	1,311,163	**($ x ‘000)**
Cash Costs (AISC)	351	**$/MgO t**
Capital Costs		
Initial Capital Incl. Contingency	235,885	($ x ‘000)
Sustaining Capital Incl. Contingency	3,935	($ x ‘000)
Total Capital Costs	**239,820**	**($ x ‘000)**
Working Capital	20,101	($ x ‘000)
Net Pre-Tax Cash Flow	1,051,242	($ x ‘000)

Description	Value	Unit
Pre-Tax		
NPV at 5%	$529,810	($ x '000)
IRR	24.5%	%
Payback	3.47	Years
Post-Tax		
NPV at 5%	$316,750	($ x '000)
IRR	19.3%	%
Payback	3.95	Years

Source: *AKF (2018)*

Notes: *NPV = net present value; % = percent; IRR = internal rate of return; EBITDA = earnings before interest, tax, depreciation and amortization; AISC = all-in sustaining costs, nr:r = non-resource:resource*

The Project is economically viable with a post-tax IRR of 19%, and an NPV, using a five percent discount rate ($NPV_{5\%}$), of $316.8 million using the Base Case metal prices.

22.2 Principal Assumptions

The principal assumptions used are shown in Table 22-2. The MgO DBM metal price scenario was used to prepare the economic analysis, and was developed using Section 19, Marketing. However, a sensitivity analysis on the metal prices was completed, and is outlined in Section 22-6.

All costs, metal prices, and economic results are reported in Canadian dollars (C$) unless stated otherwise.

Table 22-2: Principal Assumptions

Parameters	Value	Unit
MgO Price[1]	600	US$/t
Exchange Rate[2]	0.77	US$:C$
MgO Recovery	90.0	%
Mining Costs[3]	30.30	$/t processed
Transportation: Mine to Plant	43.95	$/t processed
Processing + G&A	62.06	$/t processed
Discount Rate	5	%

Source: *AKF (2018)*

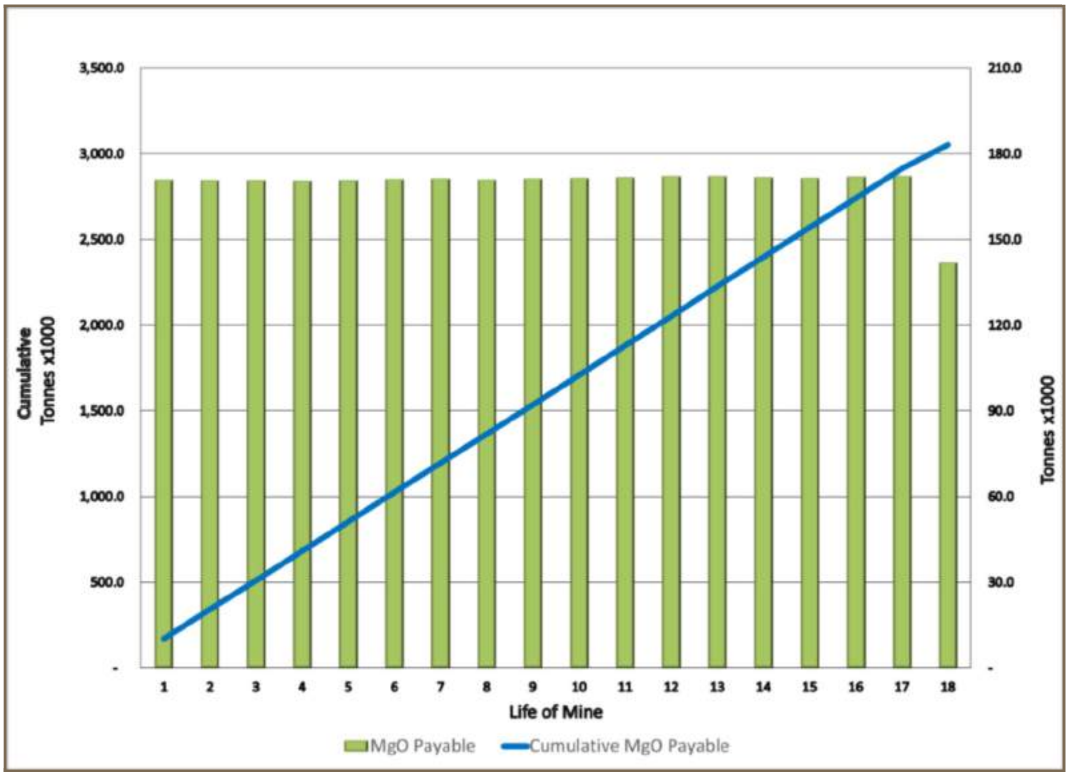
Notes: ***1.*** *MgO DBM price is FOB Cranbrook, BC.* ***2.*** *Exchange rate three-year trailing average from the Bank of Canada as of January 2018.* ***3.*** *Mining Cost is based on $8.82/t mined.*
US$/t = United States dollars per tonne; G&A = general and administration; % = percent

The reader is cautioned that the MgO DBM prices used in this study are only estimates, and there is absolutely no guarantee that they will be realized if the Project is taken into production.

22.3 Cash Flow and Annual Production Forecasts

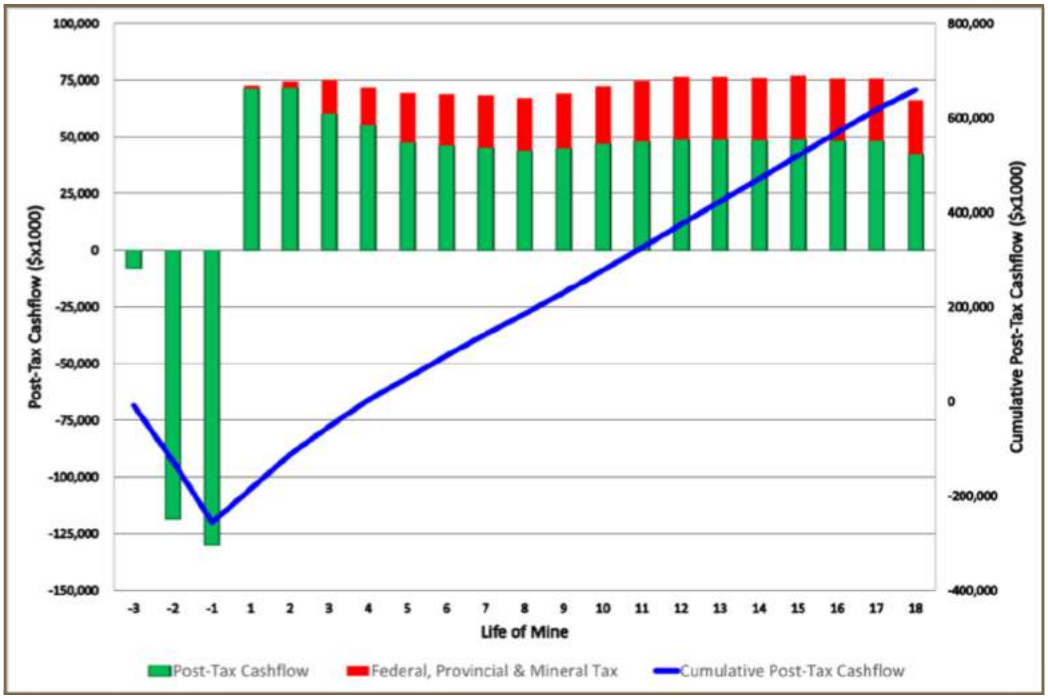
The cash flow is determined by the capital and operating costs. The pre-production capital is estimated to be $235.9 million, plus an additional $3.935 million in sustaining capital. Capital contingencies of 20% for mining and 25% for processing are included. The working capital is estimated at $20.1 million, four months of operating cost for the first year of operation. The average operating costs for the life of mine are estimated at $59.4 million, with an average net operating income (EBITDA) of $72.8 million. The cash cost is estimated at $350/MgO tonne.

The annual MgO production schedule, post-tax undiscounted annual cash flow, and tax schedule for the Project are illustrated in Figure 22-1 and Figure 22-2.



Source: *AKF (2018)*

Figure 22-1: Annual MgO Production



Source: *AKF (2018)*

Figure 22-2: Post-Tax Undiscounted Cash Flow and Tax Schedule

Economic factors include the following:

- Discount rate of 5%;
- Closure cost of $2.5 million;
- Nominal 2018 dollars;
- Revenues, costs, and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment
- Working capital calculated as four months of operating costs (mining, transportation of mineralized material from mine to plant, processing, and G&A) in Year 1 based on a 365 day operation;
- Results are assumed 100% ownership;
- No management fees or financing costs (equity fundraising was assumed); and
- All pre-development and sunk costs up to the start of detailed engineering were excluded (i.e., costs for exploration and resource definition, engineering fieldwork and studies, environmental baseline studies, etc.).

22.4 Taxes, Royalties, and Other Interests

This Project has been evaluated on a post-tax basis in order to provide a more indicative, but still approximate, value of the potential Project economics. A tax model was prepared by PricewaterhouseCoopers (PwC) located in Vancouver, Canada. The tax model contains the following assumptions:

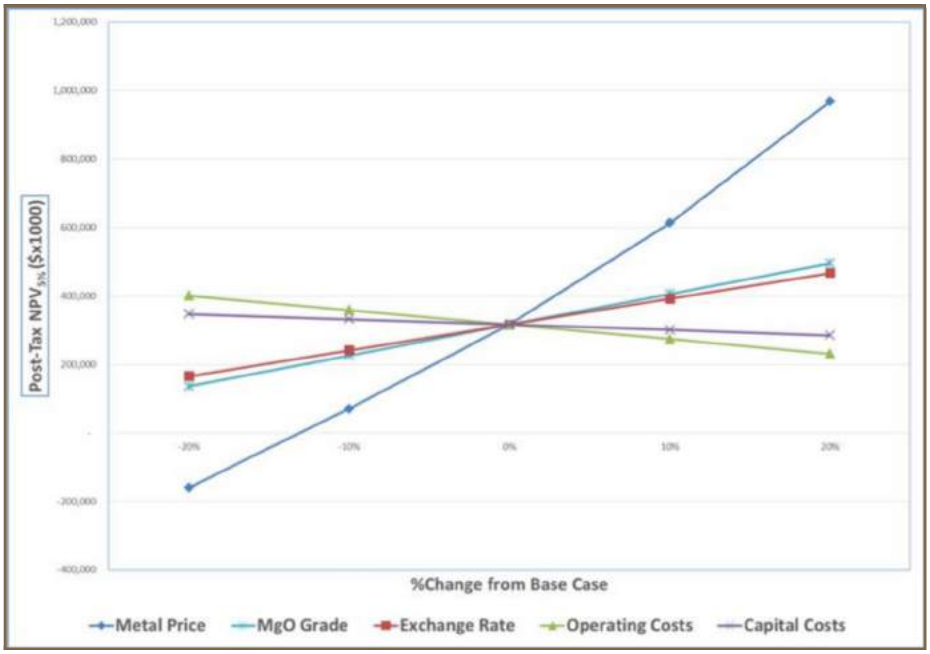
- No royalties;
- Federal/provincial tax applied at 27% rate;
- Provincial mining tax at 2% of net current proceeds and 13% of net revenue (not assessed until all pre-production capital expenditures have been amortized); and
- Canadian capital cost allowance.

Total taxes for the Project amount to $391.8 million.

22.5 Sensitivity

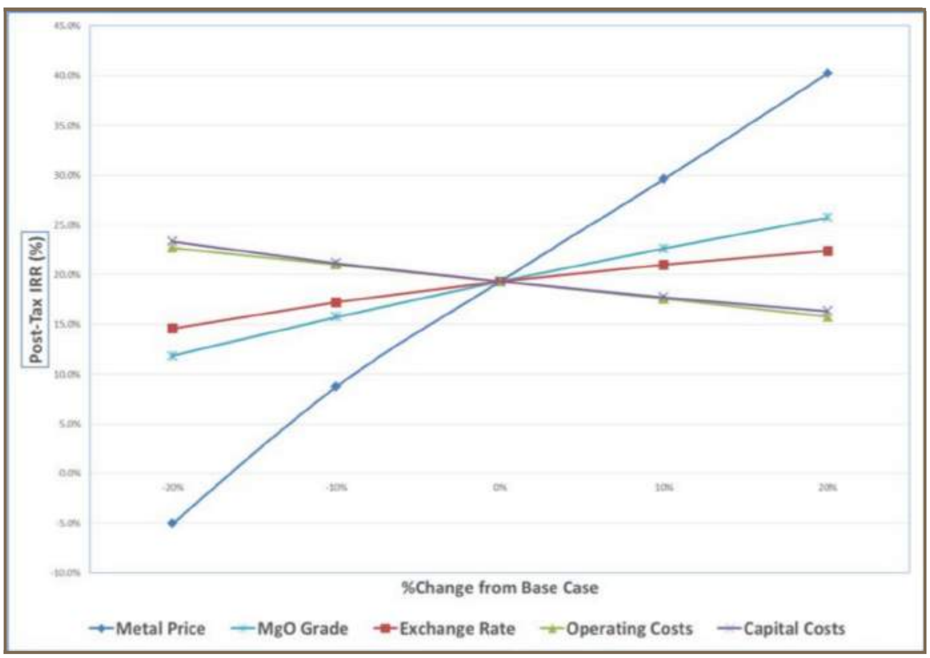
Sensitivity analysis for NPV and IRR were carried out on the following parameters, as shown in Figure 22-3, Figure 22-4, and Table 22-3:

- MgO price;
- MgO grade;
- Exchange rate;
- Operating costs;
- Capital costs; and
- Discount rates.



Source: *AKF (2018)*

Figure 22-3: Post-Tax NPV at 5% Sensitivity Analysis



Source: *AKF (2018)*

Figure 22-4: Post-Tax IRR Sensitivity Analysis

A sensitivity analysis was performed on the Base Case metal pricing cost scenarios to determine which factors most affected the Project post-tax economics for both NPV and IRR graphs. The analysis revealed that the Project is most sensitive to metal prices, followed by MgO grade and exchange rate. The Project showed the least sensitivity to operating and capital costs.

Table 22-3: Discount Rate Post-Tax Sensitivity

Discount Rate	Post-Tax NPV $M
0%	659,426
5%	316,750
8%	199,512
10%	142,854
12%	98,387

Source: *AKF (2018)*

Notes: *NPV = net present value; $M = million dollars; % = percent*

22.6 Important Caution Regarding the Economic Analysis

The PEA is preliminary in nature, in that 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.

23 ADJACENT PROPERTIES

There are no nearby properties.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Pilot Plant Milling

The Company is in the process of transporting and assembling a pilot plant at the current stockpile location of the recently completed 100-tonne bulk sample (refer to press release dated June 9, 2016). Milling equipment includes a jaw crusher, ball mill, flotation cells, cyclone dewatering equipment, and a tailings filtration and thickener system. Previously the Plant was utilized to process polymetallic concentrate. MGX intends to process mineralized bulk sample material through the Plant using reverse flotation to produce two products: a high purity magnesite tailings concentrate, and a silica sand float byproduct. The magnesite ($MgCO_3$) material will then be shipped off site to undergo calcination optimization testing to produce magnesium oxide (MgO), as well as thermal and electrolytic analysis to produce magnesium metal (Mg).

24.2 Metallurgy

Extensive metallurgical and process design work was previously completed on mineralized material from Driftwood Creek by SGS. The process design developed by SGS closely follows the current flowsheet plans for the pilot plant, inclusive of the reverse flotation and tailings dewatering system, to produce high-grade magnesite concentrate. Pilot plant testing will allow the Company to further optimize grinding, milling and flotation elements to develop a finalized process flow.

25 INTERPRETATION AND CONCLUSIONS

It is the conclusion of the Qualified Persons' preparing this technical report that the information contained within adequately supports the positive economic results obtained for the Project. The Project contains 7.843 Mt, grading at 43.27% MgO, using a 42.5% MgO cutoff grade that can be mined by open pit methods, and recovered using processing methods consisting of crushing, grinding, flotation upgrading, calcination, and sintering to produce saleable DBM and CCM products.

As demonstrated by the information contained in this report, the Project is economically viable, with a positive post-tax $NPV_{5\%}$ of \$316.7 million, with an IRR of 19.3%, and payback of four years. This Project should proceed to the next level of evaluation, either a prefeasibility or feasibility study stage.

The preliminary metallurgical testwork by SGS Lakefield indicates that iron is tied up in the magnesite crystal structure. The work did show that an acceptable magnesite concentrate could be produced with conventional flotation techniques. For this reason, MGX embarked on collecting a 100-tonne bulk sample for more detailed testwork. This testwork was in progress at the time of writing this report.

25.1 Risks and Uncertainties

As with any mining project, there are risks that could affect the economic viability of the Project. Many of these risks are based on lack of detailed knowledge, and can be managed as more sampling, testing, design, and engineering are conducted at the next study stages.

The most significant potential risks associated with the Project are lower than unexpected grades and recoveries than those projected, unanticipated mining dilution, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing, and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and proactive management.

The Project does not appear to have any environmental risks, as it is in non-potentially acid-generating dolomitic host rocks.

The Project also does not appear to have any mining or infrastructure risks, especially for labour, since the towns of Golden and Cranbrook are both within a two hour drive. Major infrastructure within approximately 15 km includes a paved highway, a CPR spur line, and a power line.

Currently, the proposed processing plant site is the decommissioned Timbec Lumber site (1400 Industrial Road #1) in Cranbrook, BC. The property is currently under a purchase option agreement with MGX and will need to be purchased should the Project progress to construction or operations stage. The site will have to undergo environmental and social impact reviews. All required infrastructure is available on site.

Additional diamond infill drilling will be needed to categorize a proven probable reserve for this Project.

Advancing the process testwork is a critical part of this Project, and, along with the pending bulk sample testing, will provide valuable information as to the quality of the product and development of a long-term marketable magnesite product.

26 RECOMMENDATIONS

The Driftwood Creek deposit has been known for 40 years, and the Project should advance to a PFS in alignment with MGX's desire to develop the resource. It is also recommended that environmental baseline studies and a socioeconomic study be initiated as soon as practical. The proposed environmental characterization studies are included as part of the proposed budget. However, other responsibilities more typically associated with environmental permitting and developing sustainable community relations are not included within the PFS budget, as they are normally considered and executed separately.

Estimated costs for a PFS-level study specific to the Project totals $8.68 million, and is itemized in Table 26-1.

Table 26-1: PFS Estimated Costs

Item	Description	Costs ($ x '000)
Processing and Calcination PFS Test work	Testing to select optimum process and recovery circuit design; determining concentrate content	250
In-fill and Condemnation Drilling Program	Inferred resources to proven and probable; drilling to ensure no resources are present under RMF and DS-TMF	300
Open Pit Geotechnical and Mine Planning Study	Mine geotechnical study to determine slope angles, and mine planning study to optimize strip ratio	200
RMF and DS-TMF	Geotechnical engineering for management facility	250
Access Road Option Preliminary Engineering	Engineering design for access road option to reduce truck turn-around time	150
Baseline Studies, Environmental Assessments, and Permitting	ML/ARD testing, water sampling, public and Aboriginal consultation, etc.	6,800
Trade-off Study for ROM haul	ROM haul directly to Cranbrook vs. rail and truck haul	80
Project Management and Engineering	PFS-level mine, infrastructure, and process designs. Cost estimation, scheduling, and economic modelling.	600
Market Studies	Independent reports to support pricing and economic model	50
Total	**Excludes owners costs, permitting activities, and contingency**	**8,680**

Source: *AKF (2018)*

Notes: *PFS = prefeasibility study; RMF = Rock Management Facility; DS-TMF = dry-stack tailings management facility; ML/ARD = metal leaching / acid rock drainage; ROM = run-of-mine*

27 REFERENCES

Bray, E.L., 2016. Magnesium, U.S. Geological Survey 2014 Minerals Yearbook – Magnesium (Advance Release).

Bray, E.L., 2016. Magnesium Compounds, US Geological Survey Annual Update.

Bray, E.L., 2016. Magnesium Metals, US Geological Survey Annual Update.

Katay, F., 2014. Exploration and mining in the Kootenay-Boundary Region, British Columbia. BC Ministry of Energy and Mines, BC Geological Survey IC 2015-2.

Kikauka, A., 2000. Geological and Geochemical report on the MG 1-7 claims, Driftwood Creek, Brisco, BC, Golden Mining Division. BC Ministry of Energy, Mines and Petroleum Resources, Assessment Report 26,345.

Kikauka, A., Klewchuk, P., and Rodgers, G.M., 2001. Driftwood Creek Magnesite, in 37th Forum on the Geology of Industrial Minerals with emphasis on Western North America, pg 81.

Klewchuk, P., 2002. Prospectors Assistance Grant Report on MG 1-7 mineral claims, Driftwood Creek area, Golden Mining Division.

Klewchuk, P. 2010. Assessment Report on Access Trail Construction and Diamond Drilling, Driftwood Creek Magnesite property; BC Ministry of Energy and Mines Assessment Report 31353, 43 pp.

Klewchuk, P. 2010a. Report on the 2008 diamond-drilling program, Driftwood Creek magnesite deposit, unpublished internal company report.

MacIntyre, D.G., 2015. Driftwood Creek Magnesite Property NI 43-101 Technical Report (Amended).

McCammon, J.W., 1965. The Brisco Magnesite area; in B.C. Minister of Mines and Petroleum Resources, Annual report for 1964. Golden Mining Division. BC Ministry of Energy, Mines and Petroleum Resources, Assessment Report 17538.

McCosh, F.D. and Schultz, B.G., 1987. Mt. Brussilof Magnesite Project, Baymag Mines Co. Ltd., Report on 1987 Exploration Program.

Morris, R.J., 1978. Fish Magnesite Deposit; BC Ministry of Energy, Mines and Petroleum Resources, Assessment Report 8760.

Rodgers, G.M., 1989. Geological Report on the Tam 1-8 mineral claims; B.C. Ministry of Energy, Mines and Petroleum Resources, Assessment Report 19,416.

Rodgers, G.M., 1991. Geological Report Covering Diamond Drill Program on Tam 1-8 Mineral Claims (Eastern Magnesite), Canadian Occidental Petroleum Report.

Rodgers, G.M., 2008. Aghamirian, M. and Imeson, D., 2008. An Investigation into the Beneficiation of Magnesite from Driftwood Magnesite property, B.C. Report prepared for Tusk Exploration Ltd. by SGS Lakefield Research Ltd.; B.C. Ministry of Energy, Mines and Petroleum Resources, Assessment Report 30,243.

Reesor, J.E., 1973. Geology of the Lardeau Map Area, east-half, B.C., GSC Memoir 369.

Simandl, G.J., and Hancock, K.D., 1991. Geology of the Mount Brussilof Magnesite Deposit, Southeastern British Columbia, Geological Fieldwork, 1991: B.C. Ministry of Energy, Mines and Petroleum Resources, Paper 1992-1, p. 269-278.

Simandl, G.J., and Hancock, K.D., 1991. Geology of dolomite-hosted magnesite deposits of the Brisco and Driftwood Creek areas, Geological Fieldwork, 1991: B.C. Ministry of Energy, Mines and Petroleum Resources, Paper 1992-1, p. 461-477.

Simandl, G.J. and Hancock, K.D., 1998. Sparry Magnesite, in Geological Fieldwork 1997, British Columbia Ministry of Employment and Investment, Paper 1998-1, p. 24E-1 to 24E-3.

Simandl, G.J. and Irvine, M., 2007. Primary Magnesium Industry at the Crossroads? Light Metal Age, April 2007 p. 32-35.

Simandl, G.J, and Schultes, H. 2001. Classification of Magnesite Deposits with Emphasis on the Mount Brussilof and Kunwarra Types, in 37th Forum on the Geology of Industrial Minerals with emphasis on Western North America, p. 83.

Simandl, G.J, Schultes, H, Simandl, J. and Paradis, S., 2007. Magnesium – Raw Materials, Metal Extraction, and Economics – Global Picture, Proceedings of the Ninth Biennial SGA Meeting, Dublin.

SRK Consulting Inc. 2013. Revised NI 43-101 Technical Report, Preliminary Economic Assessment, Record Ridge Project (West High Yield Resources Ltd.), BC, Canada.

Wardrop – A Tetra Tech Company, 2011. Preliminary Economic Assessment and Technical Report of the Tami-Mosi Project, Nevada.

28 CERTIFICATE OF AUTHOR

28.1 Allan Reeves, P.Geo.

I, Allan Reeves, P.Geo., an author of this technical report titled "National Instrument (NI) 43-101 Preliminary Economic Assessment Technical Report on the Driftwood Magnesite Deposit Project, Brisco, BC, Canada," with an effective date of December 31, 2016, prepared for MGX Minerals Inc. (MGX), hereby certify that:

1. I am President of Tuun Consulting Inc., with a business office c/o 539 – 23 Avenue NW, Calgary, AB, T2M 1S7, Canada.
2. I graduated from the University of Waterloo with a BSc. in 1989. I have practiced my profession continuously since 1991. My experience includes over 25 years in mine geology, engineering, and mine operations. I have worked as a consultant for the past nine years on exploration and mining projects. The work has included due diligence reviews, grade estimation, and project management.
3. I am a Registered Professional Geologist (#58763) with the Association of Professional Engineers and Geoscientists of Alberta and the Association of Professional Engineers and Geoscientists of BC (#18738), and a Project Management Professional (PMI #943540). All registrations are currently in good standing.
4. I have read the definition of "qualified person" set out in NI 43-101, and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies, applying all of the tests in Section 1.5 of the NI 43-101.
5. I visited the Project on June 9 to 11, 2016.
6. I am responsible for, Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, and 14 (except for sub-section 14.13).
7. I am independent of the Issuer and related companies, applying all of the tests in Section 1.5 of the National Instrument 43-101.
8. I have not had any prior involvement with the property.
9. I have read NI 43-101, and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the effective date of this Technical Report and the date of this certificate, to the best of my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Effective Date: December 31, 2016 Signature Date: April 16, 2018

"Original Signed and Sealed"

Allan Reeves, P.Geo., PMP
AB Permit to Practice #P12576

28.2 Antonio Loschiavo, P.Eng.

I, Antonio Loschiavo, P.Eng., an author of this technical report titled "National Instrument (NI) 43-101 Preliminary Economic Assessment Technical Report on the Driftwood Magnesite Deposit Project, Brisco, BC, Canada," with an effective date of December 31, 2016, prepared for MGX Minerals Inc. (MGX), hereby certify that:

1. I am currently the President of AKF Mining Services Inc. in Vancouver, BC, Canada; c/o 520-6362 Fraser Street, Vancouver, BC, V2W 0A1, Canada.
2. I am a graduate of University of British Columbia with a B.A.Sc in Mining & Mineral Process Engineering, 1998. I have practiced my profession continuously since 1998.
3. My relevant experience includes more than 19 years' experience in mining operations, project studies, and engineering. I have held senior mine planning and technical mine positions in Canada. I have worked as a consultant for over nine years and have performed resource evaluations, optimizations, mine planning, scheduling, cost estimation and economic analysis work for a significant number of projects throughout Canada, USA, Europe, Russia, and Central and South America.
4. I am a Professional Engineer (License# 39436) with the Association of Professional Engineers and Geoscientists of British Columbia.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I have visited the Driftwood Creek Project site on multiple occasions, with my last site visit on June 9 to 11, 2016.
7. I am responsible for, Sections 1, 2, 3, 14.13, 15, 16, 18, 19, 20, 21 (except for 21.3, but include sub-sections 21.3.2 and 21.3.3), 22, 23, 24, 25, and 26.
8. I am independent of the Issuer and related companies, applying all of the tests in Section 1.5 of the National Instrument 43-101.
9. I have had prior involvement with the property, providing engineering support since 2015.
10. I have read NI 43-101, and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11. As of the effective date of this Technical Report and the date of this certificate, to the best of my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Effective Date: December 31, 2016 Signature Date: April 16, 2018

"Original Signed and Sealed"

Antonio Loschiavo, P.Eng.

8450 East Crescent Parkway, Suite 200 Phone: 303.714.4840
Greenwood Village, CO 80111 FAX: 303.714.4800

28.3 Matthew R. Bender, P.E.

I, Matthew R. Bender, P.E., an author of this technical report titled "National Instrument (NI) 43-101 Preliminary Economic Assessment Technical Report on the Driftwood Magnesite Deposit Project, Brisco, BC, Canada," with an effective date of December 31, 2016, prepared for MGX Minerals Inc. (MGX), hereby certify that:

1. I am currently employed as Director of Metallurgy with Samuel Engineering, Inc., with an office at 8450 E. Crescent Parkway, Suite 200, Greenwood Village, CO, 80111, USA.
2. I am a graduate of the Colorado School of Mines with a BSc. in Metallurgical Engineering, 1987. I have practiced my profession continuously since 1987. My relevant experience includes more than 30 years in processing operations, project studies, technical and engineering, and management positions. I have been an independent consultant for over 20 years, and have performed metallurgical testwork programs, process design, process engineering, start-ups, plant audits, process technology and equipment sales, cost estimation, technical due diligence reviews, and report writing for mining projects worldwide.
3. I am a Registered Professional Metallurgical Engineer in the states of Nevada, USA (#11594), and Colorado, USA (#31471). I am a registered Qualified Person (QP) member with the Mining and Metallurgical Society of America (MMSA-01095QP). I am a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME).
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101), and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101. I am independent of the Issuer and related companies, applying all of the tests in Section 1.5 of the NI 43-101.
5. I have not visited the Driftwood Creek Project site.
6. I am responsible for, Sections 13, 17, 21.3 (except for sub-sections 21.3.2 and 21.3.3), and 21.4.3.
7. I am independent of the Issuer and related companies, applying all of the tests in Section 1.5 of the National Instrument 43-101.
8. I have had no prior involvement with the property that is the subject of this Technical Report.
9. I have read NI 43-101, and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the effective date of this Technical Report and the date of this certificate, to the best of my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading.

Effective Date: December 31, 2016 Signing Date: April 16, 2018

"Original Signed and Sealed"

Matt R. Bender, P.E.

APPENDIX A – Assay Method and Drill Holes Used in Estimate

Figure A-1: ALS Minerals Whole Rock Geochemistry

ALS Minerals

WHOLE ROCK GEOCHEMISTRY

ME- XRF06

SAMPLE DECOMPOSITION

50% - 50% $Li_2B_4O_7$ - $LiBO_2$ (WEI- GRA06)

ANALYTICAL METHOD

X-Ray Fluorescence Spectroscopy (XRF)

A calcined or ignited sample (0.9 g) is added to 9.0g of Lithium Borate Flux (50 % - 50 % $Li_2B_4O_7$ – $LiBO_2$), mixed well and fused in an auto fluxer between 1050 - 1100°C. A flat molten glass disc is prepared from the resulting melt. This disc is then analysed by X-ray fluorescence spectrometry.

ELEMENT	SYMBOL	UNITS	LOWER LIMIT	UPPER LIMIT
Aluminum Oxide	Al_2O_3	%	0.01	100
Barium Oxide	BaO	%	0.01	100
Calcium Oxide	CaO	%	0.01	100
Chromium Oxide	Cr_2O_3	%	0.01	100
Ferric Oxide	Fe_2O_3	%	0.01	100
Potassium Oxide	K_2O	%	0.01	100
Magnesium Oxide	MgO	%	0.01	100
Manganese Oxide	MgO	%	0.01	100
Sodium Oxide	Na_2O	%	0.01	100
Phosphorus Oxide	P_2O_5	%	0.01	100
Silicon Oxide	SiO_2	%	0.01	100
Strontium Oxide	SrO_2	%	0.01	100
Titanium Oxide	TiO_2	%	0.01	100
Loss On Ignition	LOI	%	0.01	100
	Total	%	0.01	101

NOTE: Since samples that are high in sulphides or base metals can damage Platinum crucibles, a ME- ICP06 finish method can be selected as an alternative method.

Figure A-2: ALS Minerals Specific Gravity Testing (OA-GRA-08 used)

Specific Gravity & Bulk Density

Specific Gravity (SG) and Bulk Density (BD) of ores are often under-characterized in the determination of tonnage and grade of a deposit. Incorrect assumptions or inadequate characterization of these basic rock properties can lead to gross errors in deposit tonnage. SG is determined by weighing a sample in air and in water, and it is reported as a ratio between the density of the sample and the density of water. BD is the density of a material in weight per unit volume, and it is determined by the weight of a sample and the volume of water the sample displaces. Calculations for SG and BD are corrected for air temperature and the density of the wax coating, if a wax coating is used. Solvents other than water are available in some locations; please enquire for more information.

DESCRIPTION	RANGE	CODE	PRICE PER SAMPLE ($)
Specific Gravity on solid objects	-	OA-GRA08	12.40
Specific Gravity on solid objects after wax coating	-	OA-GRA08a	16.05
Specific Gravity on pulps using pycnometer	-	OA-GRA08b	12.40
Specific Gravity on solid objects using pycnometer	-	OA-GRA08c	12.40
Bulk Density by water displacement	0.01 – 20g/cm³	OA-GRA09	12.40
Bulk Density – after wax coating	0.01 – 20g/cm³	OA-GRA09a	19.80
Wax removal charge for SG and BG	-	OA-GRA08wr	5.65

Figure A-3: Drill Holes Used in Estimate

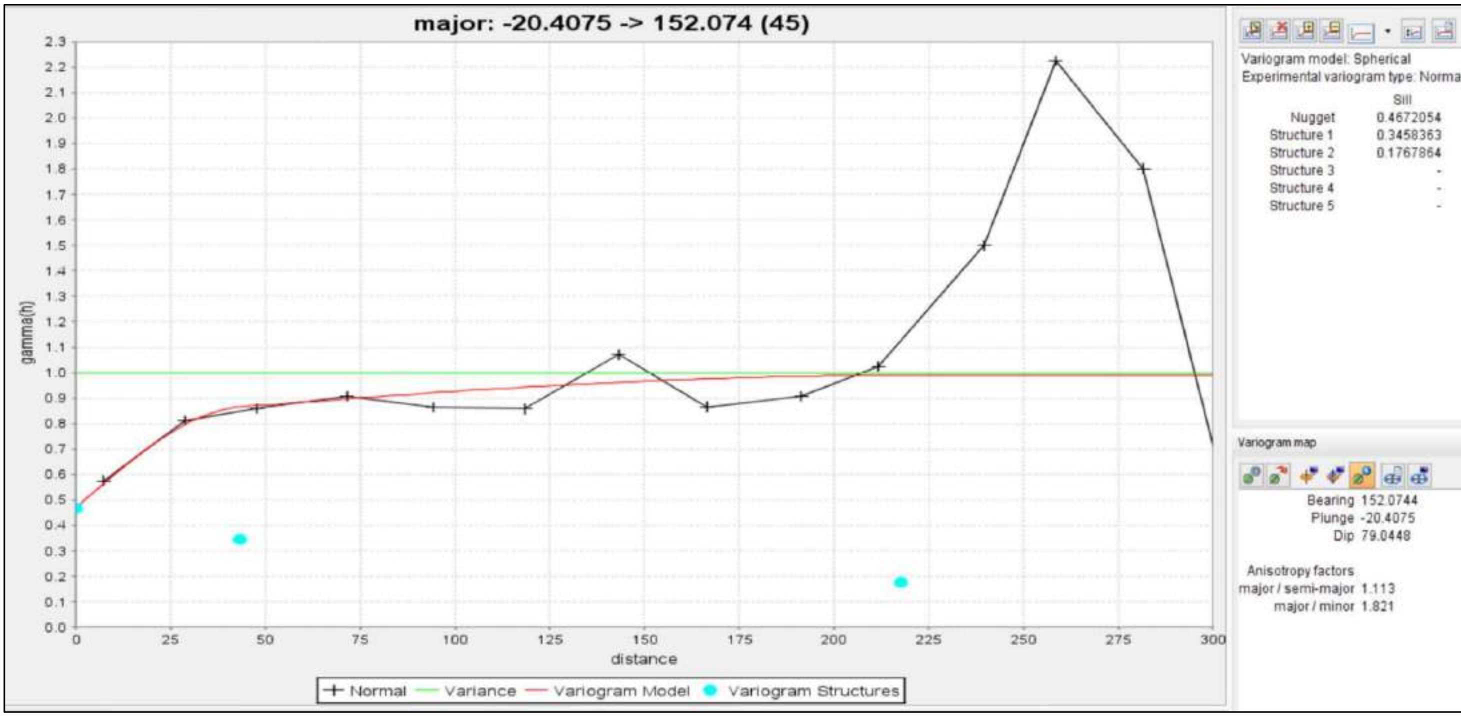
Hole-ID	East	North	Elevation	Length	Azimuth	Dip	Zone	Company	Type
DH1990-01	531,329.0	5,639,121.0	1411.0	39.93	25.0	-80.0	East	Canoxy	NQ
DH1990-02	531,329.0	5,639,122.0	1411.0	47.55	25.0	-55.0	East	Canoxy	NQ
DH1990-03	531,421.0	5,639,039.0	1419.0	60.96	25.0	-45.0	East	Canoxy	NQ
DH1990-04	531,367.0	5,639,080.0	1418.0	71.32	25.0	-45.0	East	Canoxy	NQ
DH2008-01	530,424.95	5,639,562.95	1374.18	141.50	236.0	-46.0	West	Tusk	NQ
DH2008-02	530,474.38	5,639,521.33	1377.49	133.50	210.0	-46.0	West	Tusk	NQ
DH2008-03	530,579.81	5,639,391.19	1378.64	52.20	210.0	-44.0	West	Tusk	NQ
DH2008-04	530,613.29	5,639,465.78	1393.60	82.70	215.0	-44.0	West	Tusk	NQ
DH2008-05	530,612.13	5,639,466.88	1393.75	99.40	139.0	-49.0	West	Tusk	NQ
DH2008-06	530,555.63	5,639,497.00	1386.91	100.00	210.0	-46.0	West	Tusk	NQ
DH2008-07	530,477.00	5,639,524.00	1384.00	82.70	215.0	-47.0	West	Tusk	NQ
DH2014-01	531,369.32	5,639,131.78	1418.00	37.80	200.0	-52.0	East	MGX	BTW
DH2014-02	531,390.88	5,639,108.97	1422.00	54.25	200.0	-52.0	East	MGX	BTW
DH2014-02A	531,390.88	5,639,108.97	1422.00	39.62	0.0	-90.0	East	MGX	BTW
DH2014-03	531,423.69	5,639,098.22	1426.00	65.53	200.0	-52.0	East	MGX	BTW
DH2014-04	531,458.68	5,639,071.05	1430.00	74.20	200.0	-52.0	East	MGX	BTW
DH2014-05	531,493.78	5,639,049.52	1433.00	71.63	200.0	-52.0	East	MGX	BTW
DH2014-06	531,553.37	5,639,034.40	1435.00	36.58	200.0	-52.0	East	MGX	BTW
DH2014-07	531,414.02	5,639,079.45	1424.00	57.91	0.0	-90.0	East	MGX	BTW
DH2015-01	530,457.52	5,639,477.48	1384.11	121.92	205.0	65.0	West	MGX	BTW
DH2015-02	530,541.01	5,639,460.43	1388.92	93.00	205.0	-53.0	West	MGX	BTW
DH2015-03	530,561.33	5,639,433.53	1387.29	65.53	205.0	-30.0	West	MGX	BTW
DH2015-04	530,369.54	5,639,530.12	1368.12	128.02	205.0	-53.0	West	MGX	BTW
DH2015-05	530,311.21	5,639,568.64	1367.78	125.88	205.0	-53.0	West	MGX	BTW
DH2015-06	530,389.55	5,639,606.26	1371.85	114.30	205.0	-50.0	West	MGX	BTW
DH2015-07	530,464.34	5,639,631.55	1380.51	44.20	205.0	-53.0	West	MGX	BTW
DH2015-07A	530,474.13	5,639,627.68	1381.04	18.29	25.0	-60.0	West	MGX	BTW
DH2015-08	530,355.78	5,639,648.01	1375.52	108.20	205.0	-50.0	West	MGX	BTW
DH2015-09	530,685.67	5,639,465.59	1378.52	79.86	205.0	-50.0	West	MGX	BTW
DH2015-10	530,726.86	5,639,441.24	1378.84	43.28	215.0	-50.0	West	MGX	BTW
DH2015-11	530,606.90	5,639,570.42	1380.00	16.76	205.0	-53.0	West	MGX	BTW
DH2015-11A	530,605.59	5,639,571.03	1380.45	21.34	25.0	-60.0	West	MGX	BTW
DH2015-12	530,329.02	5,639,590.27	1359.88	112.80	25.0	-60.0	West	MGX	BTW
B1410-01-01	531,414.55	5,639,072.37	1424.54	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-02	531,413.37	5,639,073.93	1423.88	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-03	531,411.91	5,639,075.69	1423.72	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-04	531,410.04	5,639,078.46	1423.59	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-05	531,408.66	5,639,080.39	1423.43	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-06	531,416.55	5,639,073.89	1424.27	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-07	531,415.52	5,639,075.37	1424.16	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-08	531,414.37	5,639,077.16	1424.01	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-09	531,412.59	5,639,079.58	1423.81	9.14	0.0	-90.0	East	MGX	Perc

Hole-ID	East	North	Elevation	Length	Azimuth	Dip	Zone	Company	Type
B1410-01-10	531,410.70	5,639,081.54	1423.44	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-11	531,417.98	5,639,075.30	1424.32	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-12	531,417.06	5,639,076.66	1424.15	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-13	531,415.74	5,639,078.28	1424.07	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-14	531,414.03	5,639,080.53	1423.81	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-15	531,411.57	5,639,082.89	1423.70	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-16	531,419.35	5,639,076.79	1425.04	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-17	531,418.46	5,639,078.13	1424.87	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-18	531,417.36	5,639,080.24	1424.83	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-19	531,415.63	5,639,082.41	1424.67	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-20	531,413.50	5,639,084.21	1424.28	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-21	531,420.70	5,639,077.88	1425.49	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-22	531,419.92	5,639,079.54	1425.28	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-23	531,418.93	5,639,082.00	1425.10	9.14	0.0	-90.0	East	MGX	Perc
B1410-01-24	531,417.39	5,639,083.64	1425.26	12.19	0.0	-90.0	East	MGX	Perc
B1410-01-25	531,415.35	5,639,086.15	1424.82	9.14	0.0	-90.0	East	MGX	Perc
DH2016-01	531,421.92	5,639,031.93	1417.85	39.0	205.0	-77.0	East	MGX	BTW
DH2016-02	531,528.93	5,638,975.09	1425.80	94.0	25.0	-45.0	East	MGX	BTW
DH2016-03	531,564.35	5,638,976.45	1430.58	97.00	25.0	-45.0	East	MGX	BTW
DH2016-04	531,478.60	5,639,018.90	1423.56	86.50	25.0	-45.0	East	MGX	BTW
DH2016-05	531,409.37	5,639,066.46	1421.32	46.00	0.0	-90.0	East	MGX	BTW
DH2016-06	531,382.31	5,639,092.63	1418.04	49.00	0.0	-90.0	East	MGX	BTW
DH2016-07	531,356.29	5,639,114.48	1412.93	49.00	0.0	-90.0	East	MGX	BTW
DH2016-08	531,327.90	5,639,127.73	1404.39	31.00	0.0	-90.0	East	MGX	BTW
DH2016-09	531,424.03	5,639,176.08	1376.72	46.00	205.0	-45.0	East	MGX	BTW
DH2016-10	530,439.44	5,639,546.06	1376.99	64.00	0.0	-90.0	West	MGX	BTW
DH2016-11	530,364.86	5,639,590.10	1375.06	65.00	0.0	-90.0	West	MGX	BTW
DH2016-12	530,523.55	5,639,487.93	1386.55	75.00	0.0	-90.0	West	MGX	BTW
DH2016-13	530,509.46	5,639,513.12	1387.29	184.00	205.0	-45.0	West	MGX	BTW
DH2016-14	530,557.04	5,639,366.05	1378.40	128.00	25.0	-50.0	West	MGX	BTW
DH2016-15	530,296.97	5,639,617.07	1361.67	76.00	25.0	-50.0	West	MGX	BTW
DH2016-16	530,288.57	5,639,529.86	1374.69	82.00	0.0	-90.0	West	MGX	BTW

APPENDIX B – Semi-Variograms

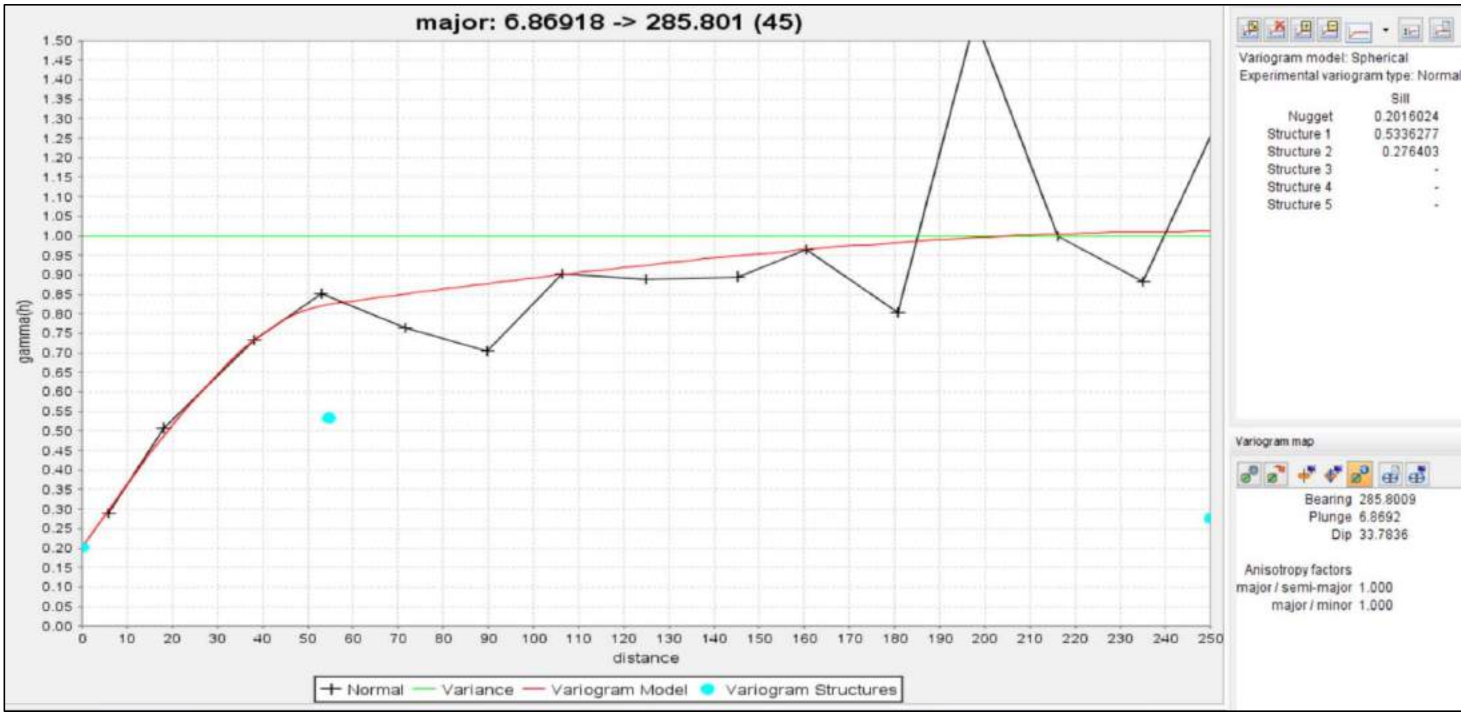
MgO%

Figure B-1: Major Axis Semi-Variogram for MgO%



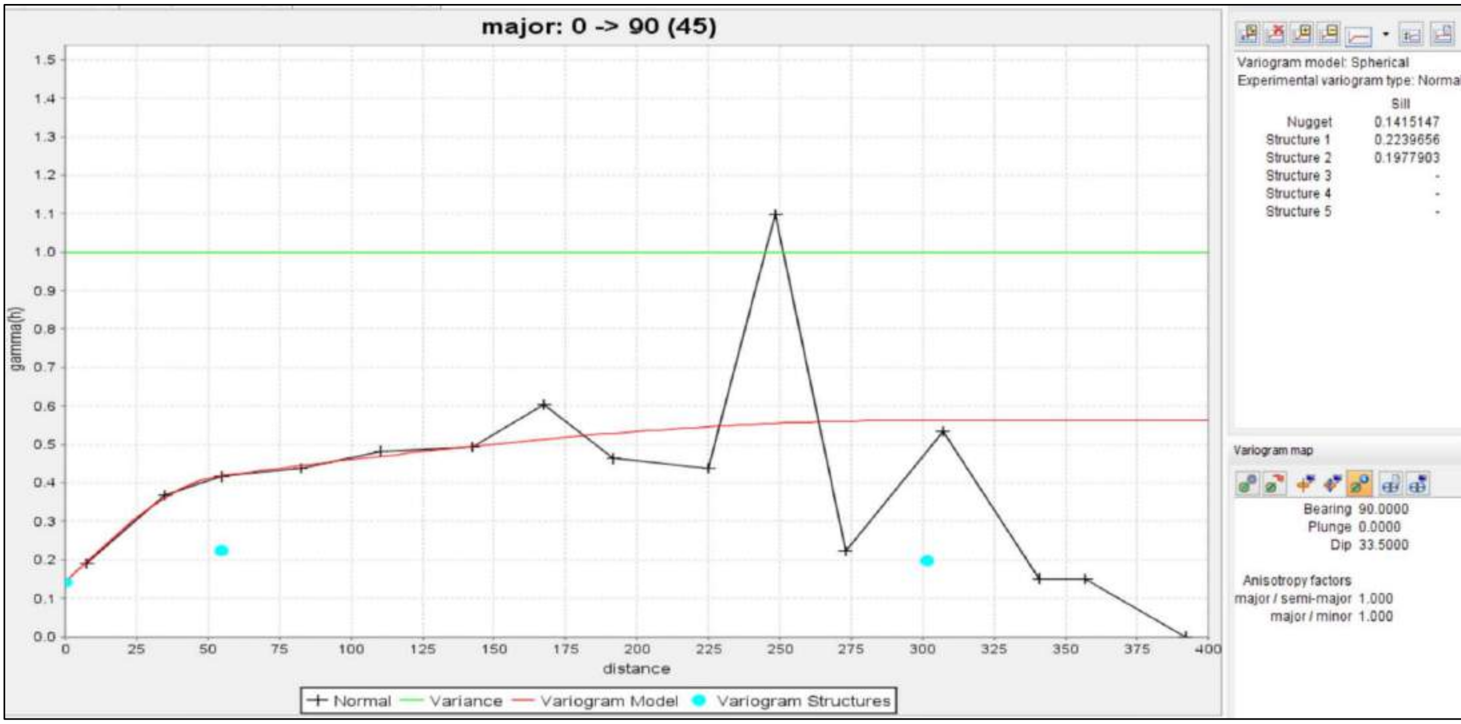
$Al_2O_3\%$

Figure B-2: Major Axis Semi-Variogram for $Al_2O_3\%$



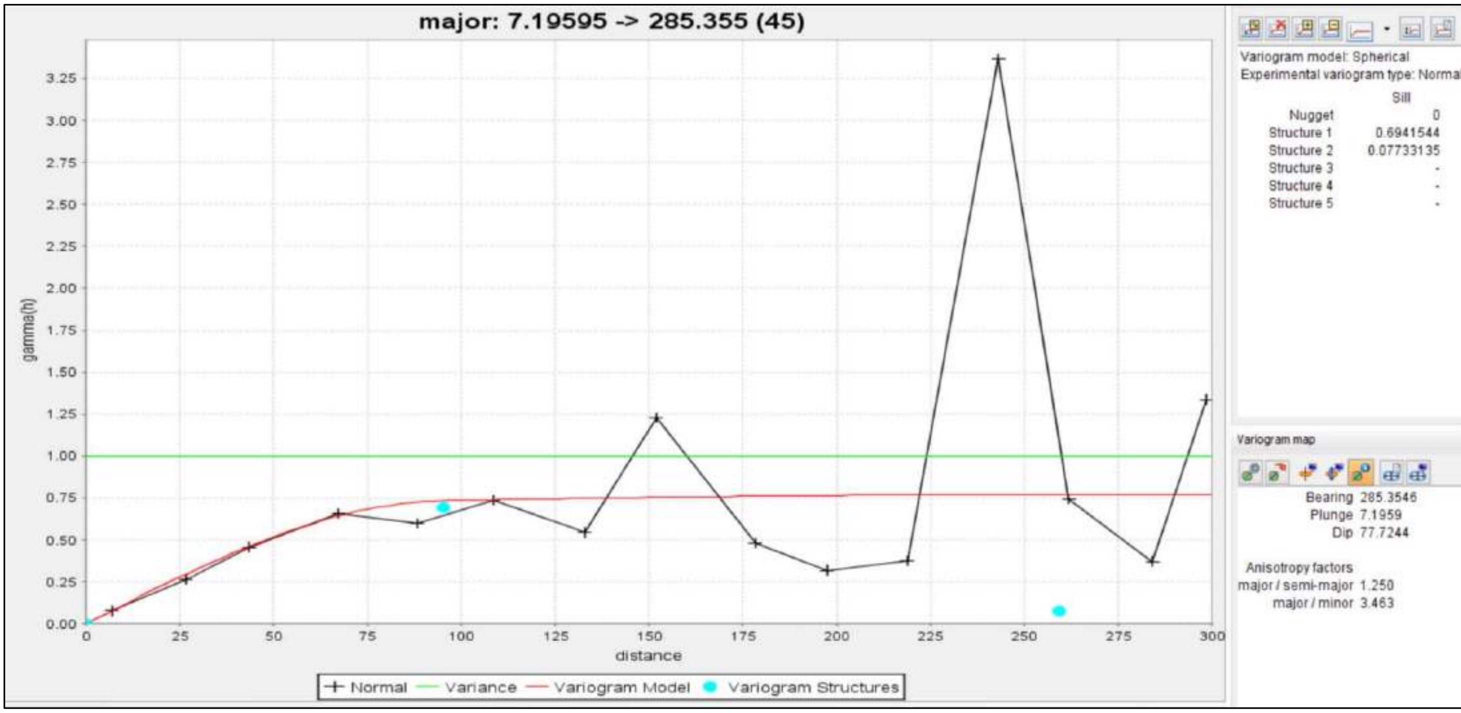
$Fe_2O_3\%$

Figure B-3: Major Axis Semi-Variogram for $Fe_2O_3\%$



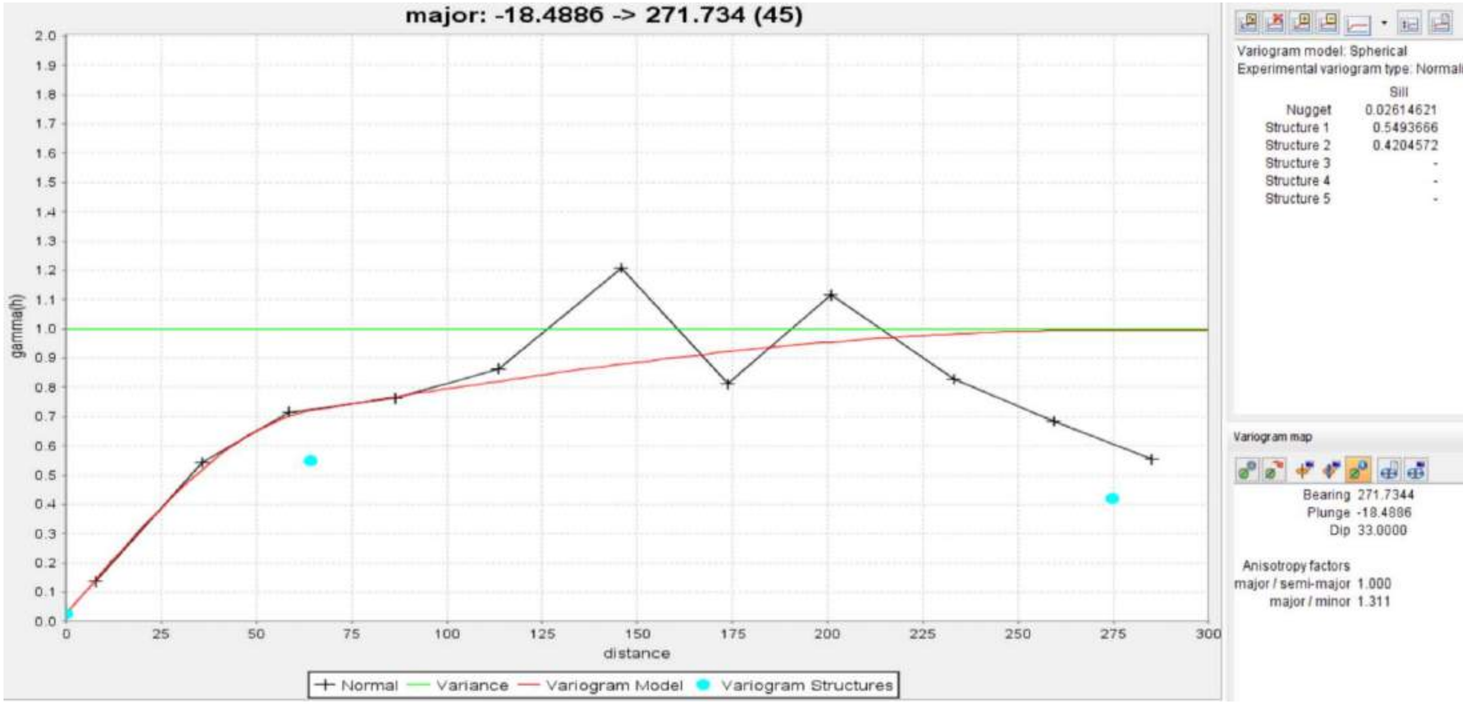
CaO%

Figure B-4: Major Axis Semi-Variogram for CaO%



LOI%

Figure B-5: Major Axis Semi-Variogram for LOI%



SiO_2%

Figure B-6: Downhole Semi-Variogram for SiO_2%