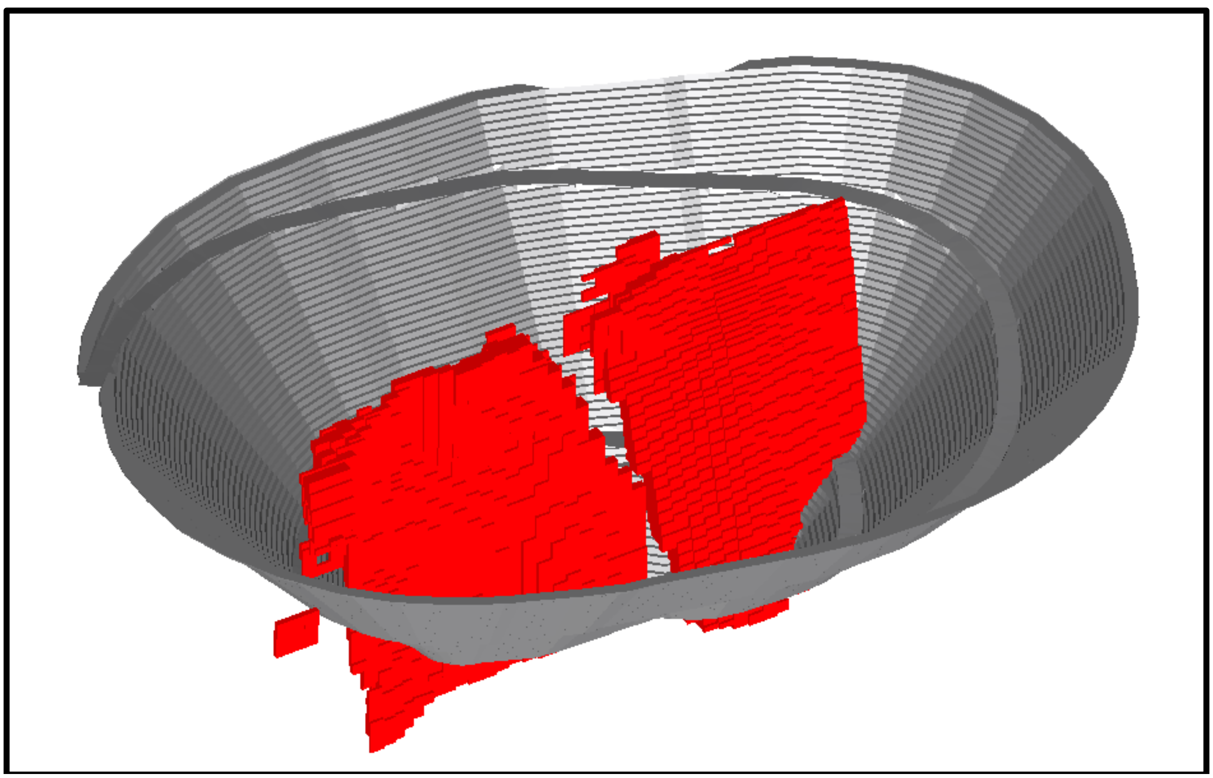
KWG Resources Inc.

NI 43-101 TECHNICAL REPORT ON THE PRELIMINARY ECONOMIC ASSESSMENT OF THE BIG DADDY CHROMITE PROJECT, McFAULDS LAKE AREA, JAMES BAY LOWLANDS, NORTHERN ONTARIO, CANADA.



Effective Date:

Signing Date:

Prepared By: NordPro Mine & Project Management Services Ltd.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

1.0 EXECUTIVE SUMMARY

The Big Daddy Chromite deposit is situated in the McFaulds Lake area in the James Bay Lowlands of north central Ontario. The property lies in NTS 43D16S½ near 86° 14' 11" W longitude, 52° 45' 32" N latitude. The area is situated 280 km due north of the town of Nakina and 258 km due west of the James Bay coastal community of Attawapiskat. The option property is situated southwest of McFaulds Lake.

The area is remote lying far from the nearest paved road at Nakina, 280 km to the south. A power line and road also serve the Musselwhite mine (Goldcorp Inc.) 280 km to the west over much better drained terrain. The area is currently accessible only by float and ski-equipped aircraft which can land on larger lakes.

Property interests at the date of this report are Spider 27½%, KWG 26½% and Freewest 46%. As per an option agreement between these three parties, KWG has the right to increase its interest to 30%.

Readers of this report (Preliminary Economic Assessment) should be cautioned that the current owner of a majority interest (Cliff Natural Resources Inc.) in the Big Daddy Deposit has indicated that it does not support the development of the project as described in this report. Therefore this report should not be relied upon as an indication or representation that the Project will be developed in the manner described herein or at all.

1.1 MINERAL RESOURCE ESTIMATE

The resource estimate previously prepared by Micon International Limited in March 2010 and published in a NI 43-101 Technical Report is based on two scenarios.

- Scenario 1: Focuses on high grade massive material that would produce a lump product which could be smelted directly.
- Scenario 2: Defines a broad zone of mineralization requiring beneficiation to upgrade to smelting grade. The broad zone is constrained by a 15% Cr_2O_3 cut-off envelope but includes internal waste up to a maximum of 3.4 metres.

The results of the resource estimate for both scenarios are summarized in Tables 1-1 and 1-2, respectively.

1.2 PRELIMINARY ECONOMIC ASSESSMENT

1.2.1 Mining

The Big Daddy deposit would be mined by open pit using excavators and diesel powered haul trucks. The nominal mining rate would be 8,000 tonnes per day of potentially economic lump chromite mineralization. Lump chromite mineralization is material which has sufficient grade (greater than 35% Cr_2O_3) that smelting can be directly performed.

The potentially mineable open pit resource is estimated to be 25.4 million tonnes at a grade of 38.02% Cr_2O_3 of Indicated Resources and 13.5 million tonnes at a grade of 37.03% Cr_2O_3 of Inferred Resources, to an ultimate open pit depth of 570 metres. Dilution of 10 percent at zero grade was included as well as losses of 3% that would result from mineralized material

Table 1-1. Summary of the Big Daddy Massive Chromite Resources.

Deposit/Code	Category	Cr_2O_3% Interval	Tonnes x 10^6	Avg. Cr_2O_3%	Cr/Fe Ratio
BD 1 (100)	Indicated	>35.0	12.934	40.74	2.0
		30.0 – 35.0	0.435	33.63	1.8
		25.0 – 30.0	0.017	28.87	1.7
		20.0 – 25.0	0	0	0
		15.0 – 20.0	0	0	0
Sub-total			13.4	40.49	2.0
BD 2	Indicated	>35.0	9.234	41.44	2.0
		30.0 – 35.0	0.520	32.83	1.8
		25.0 – 30.0	0.090	29.36	1.7
		20.0 – 25.0	0	0	0
		15.0 – 20.0	0	0	0
Sub-total			9.8	40.88	2.0
Grand Total	**Indicated**		**23.2**	**40.66**	**2.0**
BD 1 (100)	Inferred	>35.0	6.216	39.34	2.0
		30.0 – 35.0	1.014	33.25	1.8
		25.0 – 30.0	0.005	27.97	1.7
		20.0 – 25.0	0	0	0
		15.0 – 20.0	0	0	0
Sub-total			7.2	38.48	2.0
BD 2	Inferred	>35.0	8.382	40.24	2.0
		30.0 – 35.0	0.609	33.32	1.8
		25.0 – 30.0	0.047	28.35	1.7
		20.0 – 25.0	0.021	22.87	1.5
		15.0 – 20.0	0.042	16.76	1.1
		.01 – 15.0	0	0	0
Sub-total			9.1	39.57	2.0
Grand Total	**Inferred**		**16.3**	**39.09**	**2.0**

Note: The tonnages have been rounded to 3 decimals for grade intervals and to 1 decimal for sub-totals and grand totals.

Table 1-2. Summary of the Big Daddy Chromite Deposit Mineral Resource @ 15% Cr_2O_3 Cut-off

Deposit/Code	Category	Cr_2O_3% Interval	Tonnes x 10^6	Avg. Cr_2O_3%	Cr/Fe Ratio
BD 1 (100)	Indicated	>35.0	13.535	40.22	2.0
		30.0 – 35.0	1.333	32.98	1.8
		25.0 – 30.0	0.447	27.77	1.7
		20.0 – 25.0	0.152	23.34	1.5
		15.0 – 20.0	0.019	17.81	1.1
		0.01 – 15.0	0.001	12.09	0.7
Sub-total			15.5	39.05	2.0
BD 2	Indicated	>35.0	9.622	41.11	2.0
		30.0 – 35.0	1.031	32.97	1.8
		25.0 – 30.0	0.190	28.04	1.7
		20.0 – 25.0	0.007	22.56	1.4
		15.0 – 20.0	0.009	18.46	1.2
		0.01 – 15.0	0.087	7.74	0.6
Sub-total			10.9	39.82	1.9
Grand Total	**Indicated**		**26.4**	**39.37**	**2.0**
BD 1 (100)	Inferred	>35.0	7.097	39.14	2.0
		30.0 – 35.0	1.877	32.94	1.8
		25.0 – 30.0	0.543	27.93	1.7
		20.0 – 25.0	0.349	22.58	1.4
		15.0 – 20.0	0.174	18.33	1.1
		0.01 – 15.0	0.016	9.17	0.6
Sub-total			10.1	36.40	1.9
BD 2	Inferred	>35.0	8.993	39.80	2.0
		30.0 – 35.0	0.986	32.89	1.8
		25.0 – 30.0	0.241	28.06	1.7
		20.0 – 25.0	0.123	23.11	1.5
		15.0 – 20.0	0.059	16.90	1.0
		.01 – 15.0	0.014	11.96	0.9
Sub-total			10.4	38.51	2.0
Grand Total	**Inferred**		**20.5**	**37.47**	**1.9**

(Includes internal waste within the 15% Cr_2O_3 envelope).

Note: The tonnages have been rounded to 3 decimals for grade intervals and to 1 decimal for sub-totals and grand totals.

being sent to the waste stockpile. This estimate was derived utilizing the lump chromite mineralization price of US$325 per tonne Cr_2O_3 open pit shell produced by MineSight. The internal cut-off grade was not used for reporting; rather a cut-off grade of 35% Cr_2O_3 was used since it yields run of mine lump chromite mineralization ready for smelting. Waste mining will total 1.04 billion tonnes for an overall waste to potentially economic lump chromite mineralization average stripping ratio of approximately 27:1.

Over the life of the mine a mixed haulage truck fleet will be used with initially 144 tonnes capacity trucks employed. As the stripping ratios increase 327 tonne capacity haul trucks will be used. Ramp widths for the life of mine would accommodate the larger size of trucks. Ramp widths will facilitate 2 way traffic in the majority of the open pit. One-way traffic haul roads of 26 metres wide would be used toward the open pit bottom.

1.2.2 Product Preparation

The product preparation plant will accept run-of-mine (ROM) lump chromite mineralization. The plant will size material, through a series of crushers and screens, to minus 50 mm (2") in size.

1.2.3 Infrastructure and Support Facilities

The Big Daddy Project location, not close to any major population centres, will require full service infrastructure and support facilities, including manpower accommodation and recreational facilities.

Because of the large product tonnage requiring transport from the mine to customers, a railway line would be constructed from the property to connect to the Canadian National main transcontinental rail line near Nakina. A power line from Nakina to the mine site would be constructed on the railway right of way to supply power to the project. It has been assumed that an airstrip servicing the Ring of Fire area central to all the properties in the area will be constructed in the near future.

The main site infrastructure requirements for the mine would be site roads; haul roads; explosives magazines; mine maintenance shop; warehouse and laydown yard; services/technical/administration office building; camp and recreational facilities; electrical substations and distribution; water supply system and treatment plant; landfill site; and sewage disposal systems.

1.2.4 Capital & Operating Costs

The estimated project pre-production capital expenditures, inclusive of contingencies is approximately $784 million, including an average 20% contingency. A summary of project pre-production capital expenditures is presented in Table 1-3. The largest single capital expenditure component is for the railway line from Nakina to the mine site with total expenditures required of $900 million. For the PEA it has been assumed that 50% of the

Table 1-3. Project Pre-Production Capital Expenditures ($).

Component	Total Expenditure ($)
Mine	$156,190,000
Product Preparation Plant	$ 15,687,000
Railway	$450,000,000
Infrastructure	$138,793,000
Project Management Infrastructure & Mine	$ 18,500,000
Engineering Studies	$ 5,000,000
TOTAL EXPENDITURES	**$784,170,000**

railway line cost will be borne by the project and the remainder by other projects in the area under development or being studied for development.

Life of mine operating costs, as presented in Table 1-4, average $130.37 per tonne of potentially mineable lump chromite mineralization.

Table 1-4. Life of Mine Average Operating Costs per Tonne of Potentially Mineable Lump Chromite Mineralization ($).

Department	Cost ($/t)
Mine	$ 47.28
Product Preparation	$ 1.96
G&A	$ 2.88
Transport to Customer	$ 75.00
Net Smelter Return Royalty	$ 3.25
TOTAL	**$130.37**

1. Rail transport of lump chromite mineralization minus 50 mm includes transport from site, transloading at Nakina and transport to shipping port.

1.2.5 Financial Analysis

The overall level of accuracy of this Preliminary Economic Assessment (PEA) is approximately +/- 40 percent. This Preliminary Economic Assessment includes the use of Indicated Mineral Resources and also Inferred Mineral Resources which are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Therefore, there is no certainty that the results predicted by this Technical Report and the Preliminary Economic Assessment will be realized.

All costs unless otherwise indicated are in Constant 2011 Canadian Dollars.

The expected cashflows are estimated using a lump chromite mineralization price of $US 325 per tonne and an exchange rate of $CDN: $US of 1.00. The expected returns for the project are shown in Table 1-5.

Table 1-5. Expected Project Returns.

	Pre-Tax	After-Tax
Undiscounted Gross Revenue	$12.64 billion	$12.64 billion
Undiscounted Cashflow	$ 6.30 billion	$ 4.30 billion
NPV (8%)	$ 2.48 billion	$ 1.58 billion
NPV (10%)	$ 2.01 billion	$ 1.25 billion
IRR	42.0%	31.8%
Payback Period	2.5 years	2.5 years
NSR Royalty(undiscounted)	$126 million	$126 million

Projected cashflow from the project turns positive in Year 3 of operations. The project payback period is 2.5 years.

The expected cashflow scenario reflects the Big Daddy project funding 50% of the full capital expenditures for the railway from the minesite to Nakina. There are several large projects being studied in the "Ring of Fire" and the scenario investigated in this report is that a railway would be utilized to ship chromite products from and materials and equipment into the area for all potential operations.

The project returns are very sensitive to the capital expenditures required to build the railway line from the property to Nakina, chromite prices and product transport costs. The railway expenditure allocation to the project requires a $450 million (50% of the total $900 million expenditure) investment in the pre-production period. Table 1-6 shows the project returns sensitivities to the critical cost components. Figures 1-1 and 1-2 present the sensitivity analysis graphs for pre-tax and after-tax financial results.

Further opportunities to share the expenditures for construction of the railway and if the project was required to fund the full construction expenditure for the railway were both assessed. Using 25% and 100% allocations of the railway expenditures to the project the pre-tax IRR and NPV would range from approximately 57.2% to 30.2% and $2.87 billion to $2.29 billion.

Changes in the price of lump chromite were assessed within ranges experienced from the 2010 to expected future long term prices. At prices from $US 300 to $US 425 lump chromite the pre-tax NPV and IRR range between $2.05 billion and $4.21 billion and 37.5% and 57.6%, respectively.

Chromite product transport cost has a lesser sensitivity effect on the project financial returns but is still a significant factor in project success. For +or -15% changes in the transport costs the pre-tax NPV and IRR range between $2.29 billion and $2.68 billion and 40.0% and 43.9%, respectively.

Table 1-6. Project Returns Sensitivity Analysis.

Variable	Pre-Tax NPV @ 8% ($billions)	After-Tax NPV @ 8% ($billions)	Pre-Tax IRR (%)	After-Tax IRR (%)
Chromite Price - $350	2.92	1.89	46.2	35.3
Chromite Price - $400	3.78	2.49	54.0	41.7
Chromite Price - $425	4.21	2.79	57.6	44.7
Chromite Price - $300	2.05	1.28	37.5	28.1
Product Transport Cost +15%	2.29	1.45	40.0	30.1
Product Transport Cost -15%	2.68	1.72	43.9	33.4
Capital Expenditure - 15%	2.57	1.66	46.7	35.6
Capital Expenditure +15%	2.39	1.51	38.1	28.7
Capital Expenditure +25%	2.34	1.45	35.9	26.9
Capital Expenditure +50%	2.19	1.32	31.3	23.3

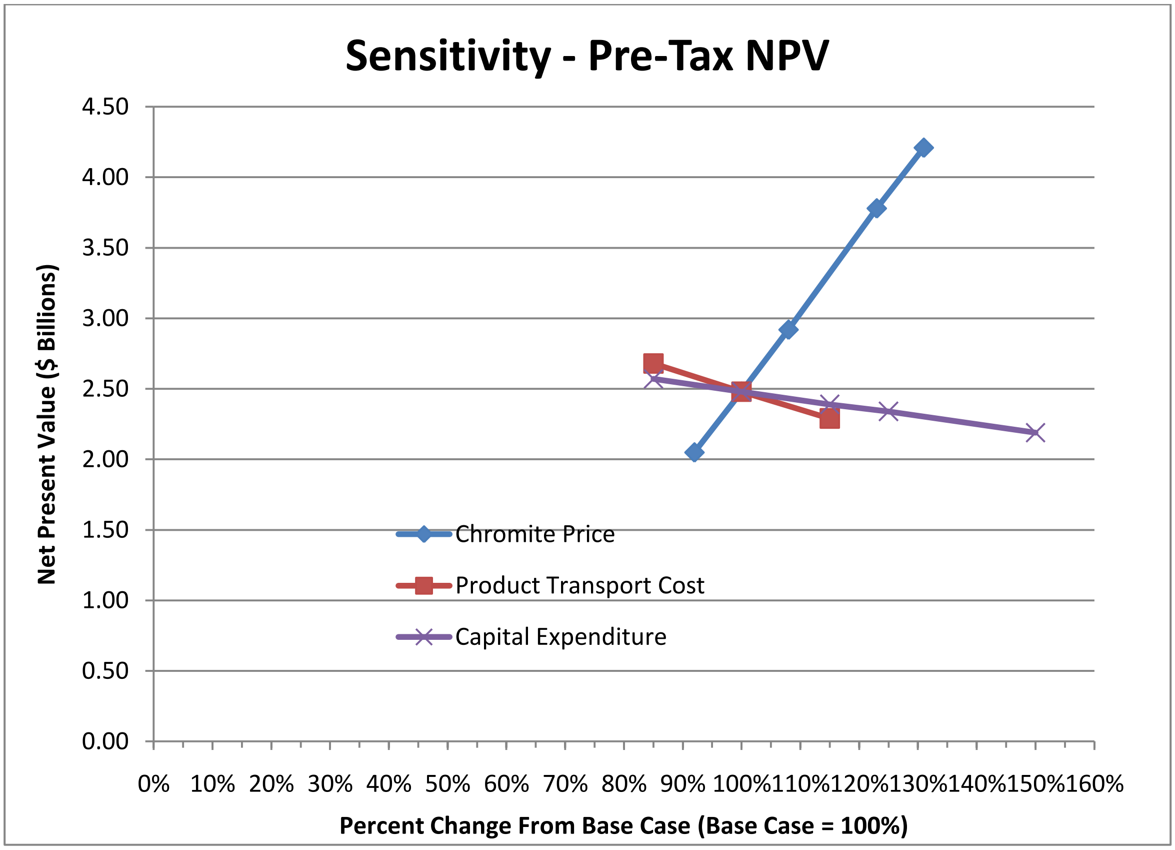


Figure 1-1. Pre-Tax Sensitivity (8%) Analysis Results.

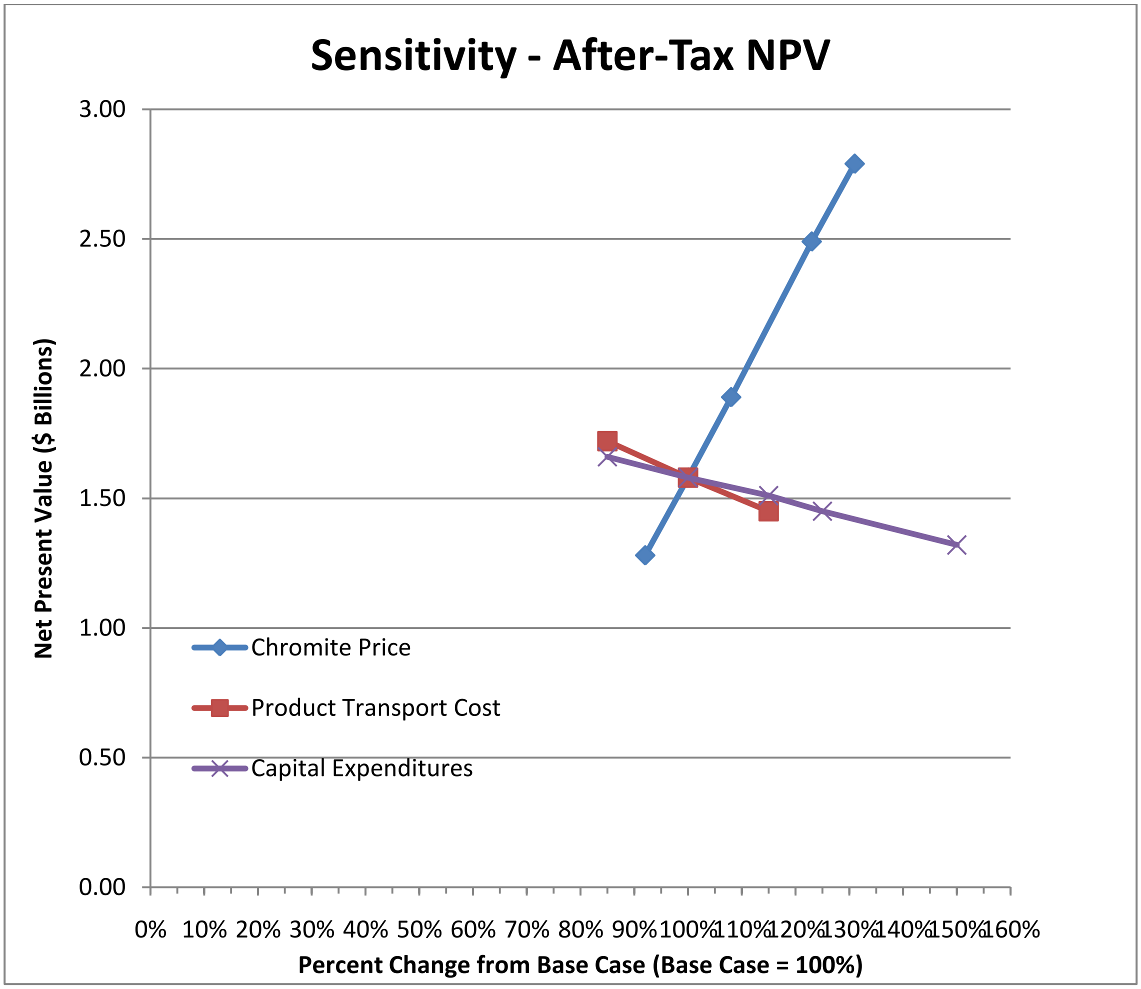


Figure 1-2. After-Tax Sensitivity (8%) Analysis Results.

1.3 RECOMMENDATIONS

Based on the results of this Preliminary Economic Assessment, recommendations are:

1. Advance the Big Daddy Project to the Feasibility Study phase, subject to the joint venture partners reaching agreement on moving the process forward.

2. Undertake diamond drilling to confirm the high grade lump chromite core, which is amenable to shipping for smelting. Also complete diamond drilling to, where possible, upgrade the inferred resources to at least indicated resources, with particular attention to Phase 3 mining resources which is materially comprised of inferred resource.

3. Complete the following technical studies to a Feasibility level of accuracy:

    a. Nakina to minesite railway alignment and railway line design, capital expenditures and operating costs. This would require geotechnical investigations and approximately 10 to 15% of the relevant engineering to be completed.

    b. Main line railway transportation and overseas shipping costs studies.

    c. Chromite marketing and pricing studies.

4. While the overburden, low grade and waste stockpiles have been included in the mine plan a geotechnical study should be completed to assess the stability of the stockpiles and provide geotechnical designs for the overburden and waste stockpiles.

5. Further assess the potential for acid rock and metal leaching from the mine materials which will be exposed.

6. Evaluate environmental assessment requirements in consultation with regulatory authorities and continue social consultation and environmental baseline work.

7. Evaluate overburden stripping options.

8. Perform geotechnical drilling and slope stability analysis.

9. On completion of items 3 to 7, to a level providing sufficient confidence in the estimates to support continuation to a Feasibility Study, update the PEA or prepare a Pre-Feasibility Study (including cost estimate results generated by the technical studies), to continue justification of a Feasibility Study.

10. First priorities in the Feasibility Study Phase should be:

    Updating and improving the geology block model.

    Complete transportation, market and processing trade-off studies.

11. Assess the railway system configuration and capital expenditures to develop the optimum system and investigate options for financing the railway expenditures.

12. The Feasibility Study Phase will require expenditures of approximately $33 million (See Table 1-7). Prior to commencing a Feasibility Study approximately 2.5 years of studies are required. The Feasibility Study would take approximately 2-3 years to complete.

Table 1-7. Feasibility Phase Budget ($).

Component	Total Cost ($)
Trade Off Studies Upgrading	
Product Transport	$ 500,000
Railway	$ 4,000,000
Market Analysis	$ 500,000
Power	$ 20,000
Feasibility Components	
Geology & Resources	$11,700000
Rock Mechanics Studies	$ 150,000
Mine Design & Optimization	$ 200,000
Geotechnical Drilling	$ 2,000,000
Metallurgical Testwork	$ 500,000
Processing	$ 100,000
Infrastructure & Surface Facilities	
Onsite Facilities (Shops, Offices, Camp, etc.)	$ 500,000
Roads & Power	$ 125,000
Environmental & Socio-Economic	
Baseline Studies	$ 1,625,000
Camp & Logistics	$ 4,000,000
General & Administration	$ 150,000
Project Indirects (Project Management, etc.)	$ 50,000
Financial Analysis	$ 50,000
FS Report Preparation	$ 100,000
Expenses - Review Meetings, Site Visit, etc.	$ 100,000
Estimated Total Cost	**$26,370,000**
Contingency	**$ 6,592,000**
Total Cost	**$32,962,000**

2.0 INTRODUCTION

This report was prepared by NordPro Mine & Project Management Services Ltd. ("NordPro") at the request of Mr. Frank Smeenk, President of KWG Resources Inc. KWG Resources Inc. is an Ontario based publicly held company trading on the TSX-V under the symbol ''KWG", with its corporate offices at:

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The following is a Technical Report on the Preliminary Economic Assessment (the "Report") prepared by NordPro Mine & Project Management Services Ltd. for the Big Daddy Property, of KWG Resources Inc. ("KWG"), located in the McFaulds Lake area of north central Ontario, Canada (the "**Property**"). This Report has been prepared in compliance with the requirements of National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("**NI 43-101**") and in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum ("**CIM**"), Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council December 11, 2005 (CIM 2005).

In March 2010 Micon International Limited was retained to estimate mineral resources for the Big Daddy Project and to prepare a report documenting the estimate. The estimate was presented in an NI 43-101 technical report entitled `` Spider Resources Inc. and KWG Resources Inc. Technical Report on the Mineral Resource Estimate for the Big Daddy Chromite Deposit McFaulds Lake Area, James Bay Lowlands, Northern Ontario, Canada" and dated March 30, 2010. The estimate conforms to CIM standards on Mineral Resources and Mineral Reserves definitions.

Readers of this report (Preliminary Economic Assessment) should be cautioned that the current owner of a majority interest (Cliffs Natural Resources Inc.) in the Big Daddy Deposit has indicated that it does not support the development of the project as described in this report. Therefore this report should not be relied upon as an indication or representation that the Project will be developed in the manner described herein or at all.

This PEA report is considered current as of March 31, 2011.

This report is intended to be used by KWG subject to the terms and conditions of their contract with NordPro Mine & Project Management Services Ltd. This permits Spider and KWG to file this report on SEDAR (www.sedar.com) as an NI 43-101 Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation.

NordPro understands that KWG may use the report for a variety of corporate purposes including financings. Except for the purposes legislated under provincial securities laws, any other use of this report, by any third party, is at that party's sole risk.

2.1 SOURCES OF INFORMATION

This Preliminary Economic Assessment has been prepared for KWG by independent consultants, each of whom is a qualified person within the meaning of NI 43-101. These consultants have made a number of qualifications and assumptions, which are described in this study. Subject to the conditions and limitations set forth herein, the independent consultants believe that the qualifications, assumptions and the information used by them is reliable and efforts have been made to confirm this to the extent practicable. However, none of the consultants involved in this study can guarantee the accuracy of all information in this report.

Information contained in this Preliminary Economic Assessment was prepared by the following consultants, working with KWG personnel:

Consultant	Responsibilities
Micon International Limited	Geology and resource estimates
NordPro Mine Project Management Services Ltd.	All aspects of study other than geology, and resource estimates.
EHA Engineering Ltd.	Processing

2.2 INFERRED RESOURCES

This Preliminary Economic Assessment uses Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Therefore, there is no certainty that the results predicted by this Preliminary Economic Assessment will be realized.

2.3 MARGIN OF ERROR

The overall level of accuracy of this study is +/- 40%.

2.4 SITE VISITS

A site visit for this study was not carried out.

2.5 ABBREVIATIONS

All data and information is presented in metric units unless otherwise stated. For this report the following abbreviations were used:

Abbreviation	Description
μm	micrometre
°C	degree Celsius
$/t	Dollars per tonne
$C	Canadian Dollar
$US	United States Dollar
AA	Atomic absorption spectroscopy
AG	autogenous
BD	Bulk Density
CCA	capital cost allowance
CDE	Canadian development expense
CEE	Canadian exploration expense
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetres
G&A	General and Administration
ha	hectares
HDPE	high-density polyethylene
HP	Horsepower
HSS	high speed steel
HST	Federal and Provincial Harmonized Sales Tax
ICP-AES	inductively coupled plasma-atomic emission spectroscopy
ICP-OES	inductively coupled plasma-optical emission spectroscopy
IRR	internal rate of return
km	kilometres
kW	kilowatt
kWh/t	Kilowatt hour per tonne
L	Litre
lb	Pounds
LIMS	laboratory information management system
masl	metres above sea level
m	metres
m/s	metres/second
MCC	motor control centre
mL	Millilitres
mm	millimetres
Mt	Million tonnes
MVA	millions volt-amps
NI	National Instrument (43-101)
NN	Nearest neighbour
NPV	Net Present Value

NSR	Net Smelter Return
OK	Ordinary kriging
PLC	programmable logical controller
Q	Quarter
QA/QC	quality assurance and quality control
QP	Qualified Person
RGO	regular grade ore
ROM	run of mine
RMS	root mean squared
RQD	rock quality designation
SCC	Standards Council of Canada
SG	specific gravity
SRS	standard reference samples
std	Standard deviation
t	Tonnes (metric)
t/m3	Tonners per cubic metre
TMF	tailings management facility
tpd	Tonnes per day
UCS	Unconfined compressive strength

All expenditures and financial information are expressed in constant Q1 2011 Canadian dollars ($) unless otherwise noted.

3.0 RELIANCE ON OTHER EXPERTS

This Preliminary Economic Assessment was prepared for KWG Resources Inc. by independent consultants and KWG personnel as listed in section 2.1.

The authors are Qualified Persons only in respect of the areas in this report identified in their "Certificates of Qualified Persons" submitted with this report.

The description of the property, and ownership thereof, as set out in this report, is provided for general information purposes only.

The existing environmental conditions, liabilities and remediation have been described under the relevant section as per the NI 43-101 requirements. However, the statements made are for information purposes only and NordPro offers no opinion in this regard.

The general descriptions of geology and past exploration activities used in this report are taken from the Micon International Limited Technical Report entitled `` Spider Resources Inc. and KWG Resources Inc. Technical Report on the Mineral Resource Estimate for the Big Daddy Chromite Deposit McFaulds Lake Area, James Bay Lowlands, Northern Ontario, Canada" and dated March 30, 2010.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Big Daddy Chromite deposit is situated in the McFaulds Lake area in the James Bay Lowlands of north central Ontario (see Figure 4-1). The property lies in NTS 43D16S½ near 86° 14' 11" W longitude, 52° 45' 32" N latitude. The area is situated 280 km due north of the town of Nakina and 258 km due west of the James Bay coastal community of Attawapiskat. The option property is situated southwest of McFaulds Lake.

4.2 OWNERSHIP

On 22 April, 2003, Richard Nemis became the recorded holder of six, 16-unit claims (the Nemis Claims), comprising the western two SKF Option claims (P 3012252 and P 3012253) and the four adjoining claims to the north that now comprise the Black Thor (Freewest, 100%) property (shown in pink on Figure 4-2).

On June 17, 2003, Richard Nemis agreed to sell 100% interest in the Nemis Claims to Freewest in consideration of a payment of $10,800 and a 2% NSR royalty. The claims were transferred to Freewest on August 14, 2003.

On August 11, 2003, Freewest caused the three claims that comprise the east part of the SKF property (P 3008269, P 3008793 and P 3008268) to be recorded.

On December 5, 2005, KWG and Spider, as equal partners, agreed to earn a 50% interest in Freewest's property comprising P 3012253, P 3012252, P 3008269, P 3008793 and P 3008268 together with two single claim units (~32 ha) excised from adjoining Freewest claims 302250 and 3022251 for exploration expenditures of $1,500,000 by October 31, 2009 of which $200,000 was to be spent by February 28, 2006. The addition of the two single units permits Spider and KWG to test two EM conductors that extend northwards onto the Black Thor property.

In March, 2009, Freewest, KWG and Spider entered into a letter agreement which forms the basis for the September 10, 2009 agreement described below.

On July 21, 2009, Nemis, Freewest and KWG entered an agreement whereby KWG purchased half of the Nemis NSR (i.e., 1% NSR royalty) which was conveyed to 7207565 Canada Inc., a subsidiary of KWG.

Figure 4-1. Location Map of the Big Daddy Chromite Deposit

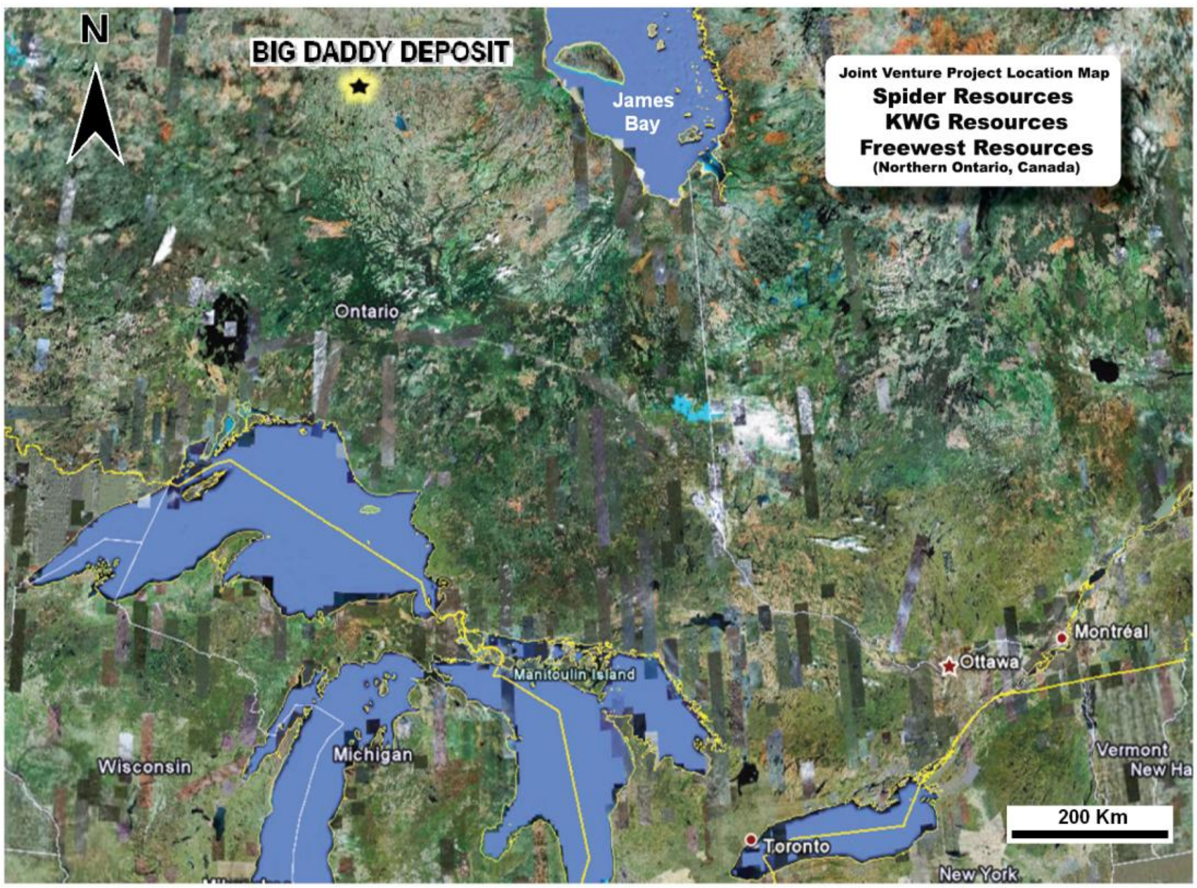
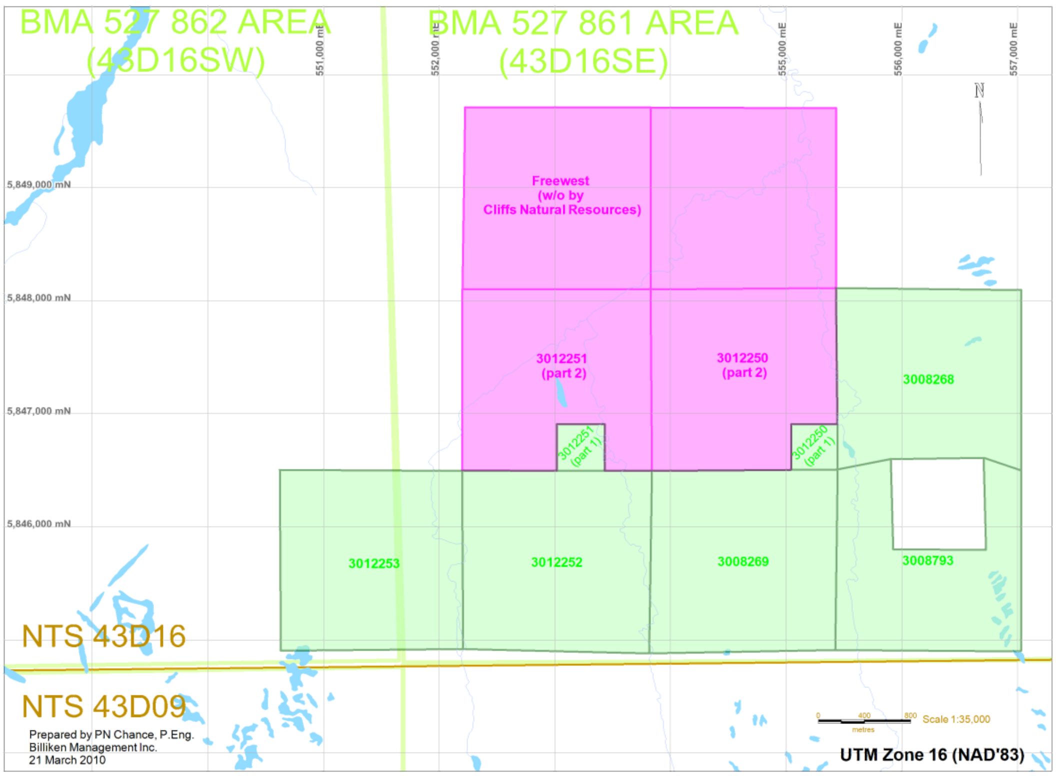


Figure 4-2. SKF Project Claim Map (SKF Option claims are shown in green. Claim locations are "as staked" based on GPS-derived locations of claim posts).



On September 10, 2009, Freewest, KWG and Spider amended and restated the December 5, 2005 agreement, allowing KWG and/or Spider to earn a combined additional 10% interest in the property through annual expenditures of $2,500,000 each within three years ending March 31, earning 3% in each of the first two years and 4% in the last year ending March 31, 2012. The additional 10% may also be earned should one or both parties spend a minimum of $5,000,000 and deliver a positive feasibility study to Freewest by March 31, 2012.

Title of the property was to be transferred to KWG to be held in trust as per the option agreement.

The September, 2009 agreement acknowledged that KWG and Spider had already each earned a 25% interest in the property and warranted that there were no encumbrances on the property beyond the NSR royalty.

Spider (100% owned by Cliffs Natural Resources Inc.) operated the project from inception until March 31, 2010. KWG is presently the operator until March 31, 2011, when the operatorship will be assumed by Cliffs Natural Resources Inc.

As of March 31, 2011 property interests are Spider 27½%, KWG 26½% and Freewest 46%. As per an option agreement between these three parties, KWG has the right to increase its interest to 30%.

In addition KWG Resources Inc. has a 100% interest in Canada Chrome Corporation which in turn has a 100% ownership to potential rights of way (through staked claims) for a potential railway line to the project site.

4.2.1 Royalty Interests

Richard Nemis and KWG Resources each held a 1% Net Smelter Return royalty on claims P 3012252, P 3012253 (Big Daddy deposit), and the adjoining single unit portions of P 3012250 and P 3012251. In Q1 2011 the royalty interest of Richard Nemis was extinguished for the payment of 4 million treasury units valued at $ 0.10 and containing one treasury share and one share purchase warrant.

4.2.2 Other Parties to the Agreement

Freewest Resources Canada Inc. is a wholly owned subsidiary of Cliffs Natural Resources Inc. KWG Resources Inc. is a junior exploration company in which Cliffs Natural Resources holds approximately a 17.7% interest (undiluted) and 18.4% interest (fully diluted).

4.3 OWNERSHIP OBLIGATIONS

The property is in good standing until April and August, 2011. Annual assessment requirements are $31,200. Claim abstracts currently report Freewest as the recorded holder; however, the property is to be held by KWG in trust.

The two westernmost claims and both contiguous single units are subject to a 1% Net Smelter Return royalty currently held by KWG Resources Inc.

4.4 COSTS OF MAINTENANCE

In Ontario, mining rights are acquired by staking out and recording claims in a manner prescribed in the Mining Act (R.S.O. 1990, Chapter M. 14 Section 38 (1)). Claim holders are required to submit proof of permitted exploration expenditures at a rate of $400 per claim unit annually starting prior to the second anniversary of recording until the claims are taken to lease. The annual maintenance costs for 78 units are $31,200. Sufficient eligible work has been completed to retain the property in good standing for many years.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRPAHY

5.1 ACCESSIBILITY

The area is remote lying far from the nearest paved road at Nakina, 280 km to the south. A power line and road also serve the Musselwhite mine (Goldcorp Inc.) 280 km to the west over much better drained terrain. The area is currently accessible only by float and ski-equipped aircraft which can land on larger lakes. Aircraft are available in Nakina and Pickle Lake. Helicopters are essential for local transport, although skidoos and larger tracked vehicles can be used when the ground is frozen and there is sufficient snowpack.

Nakina has a paved 3,500 foot (about 1,000 m) runway. Longer runways are available at Geraldton, (5,000 feet (about 1,500 m), 337 km south) and Pickle Lake (4,500 feet (about 1,400 m), 310 km southwest).

The Canadian National Railway Company (CN) main transcontinental rail line passes through Nakina.

Thunder Bay (540 km south-southwest) is the regional centre with daily air service to the remote communities of Nakina and Pickle Lake. Although the Ministry of Northern Development, Mines and Forestry's (MNDM&F's) development coordinator is based in Thunder Bay, the area lies in the Porcupine Mining Division and is administered from Timmins (about 600 km southeast).

All-weather highways extend to Nakina (Highway 584) and Pickle Lake (Highway 808) from where the gravel North Road extends 193 km to Opapimiskan Lake (Musselwhite mine) to the west.

The Ontario power grid reaches the Victor mine (DeBeers Canada Inc.), 157 km to the east, Nakina, 280 km to the south and the Musselwhite mine 290 km to the west.

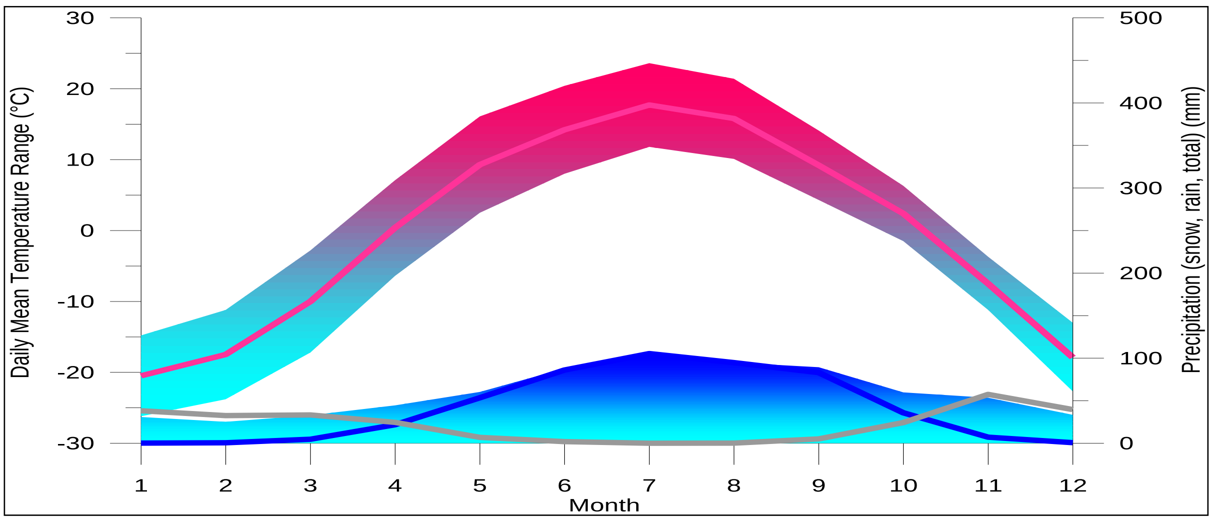
During 2009 Marten Falls Logistics began construction of an airstrip about four kilometres north of Noront's Esker camp. Work is reported as halted due in part to permitting issues.

In 2009 KWG through its subsidiary Canada Chrome Corporation, staked two "rail corridors". Subsequently Canada Chrome Corporation commenced a scoping study pertaining to costs for a rail link to the McFaulds Lake area. The company has completed a geotechnical, soil sampling program along a 340 km corridor to the area from the CN railway line at Exton near Nakina.

5.2 CLIMATE

Mean temperatures range from -20°C in December and January to a peak of 15°C in July (Figure 5-1). Annual precipitation is about 70 cm, of which almost 70% falls as rain with peak amounts during July. Snowfall peaks in November gradually diminishing to March. Typical snow accumulations are about 0.6 m.

Figure 5-1. Annual Mean Daily Temperature and Range, January to December (month 1 to 12).



(Upper curves, left scale) and monthly precipitation (lower curves, right scale) showing rain (blue curve) and snow (grey curve for Pickle Lake (1971-2010 data; source: www.climate.weatheroffice.gc.ca).).

5.3 VEGETATION

The property lies in a broad transition zone between the boreal forest and arctic tundra further north. It is covered by extensive fen and bog complexes with highly variable tree cover intermixed with vast numbers of ponds and lakes. The principal tree species include black spruce (Picea mariana), white spruce (Picea glauca) and tamarack (Larix laricina) (Sjörs, 1959). Caribou grazing locally alters plant community structure (Proceviat et al., 2001).

5.4 FAUNA

While a wide range of animals and birds are reported, those observed include fox, wolf, marten, moose, black bear and woodland caribou. The area lies in the northern range of the woodland caribou, a species at risk in its southern range due to habitat loss (removal of old-growth, boreal forest; Proceviat et al., 2001).

"Winter survival of woodland caribou in the black spruce peatlands of northeastern Ontario also appears to be dependent on the availability of ground and arboreal lichen and arboreal lichen biomass has been shown to be an important parameter identifying late winter habitat selected by this species." (ibid.)

Despite intermittent aerial surveys since 1950, the abundance and migration patterns of the lowland woodland caribou population are not well understood (Magoun et al, 2005).

5.5 FIRST NATIONS

First Nations communities are the principal permanent settlements in the far north of Ontario. Although the mineral rights were surrendered in the James Bay Treaty No. 9 in the early 20th century, recent court rulings combined with the absence of economic opportunity in the region have encouraged First Nations to assert rights to traditional lands. The Marten Falls First Nation asserts that the McFaulds area and KWG's currently proposed access corridor lie within its traditional lands. It is likely that other nearby communities (e.g., Webequie, Fort Hope, Lansdowne House and Summer Beaver) will seek economic advantage from developments in the area.

Prior to 2010 mineral exploration companies acted individually, reaching accommodations and in some cases agreements with First Nations. In early 2010, the Marten Falls First Nation with the support of the Webequie First Nation initiated a logistics blockade of Koper and McFaulds Lakes between January 20 and March 18. While the objectives of the blockade are unclear, the need for sustained engagement to structure development of and eventual mutually-beneficial operations in the area is clear.

The closest communities are members of the Matawa Tribal Council which represents Oji-Cree communities in an arc along the west edge of the James Bay Lowland from Constance Lake in the southeast to Neskantaga First Nation (Lansdowne House) in the northwest. The population of the remote communities is about 3,000 (2006 data; www.matawa.on.ca). The remaining 5,000 Matawa members live off reserve or in road accessible communities around Nakina, Long Lac and Hearst.

During the most recent Big Daddy drill campaigns, a third of the workforce comprised First Nations members. Geotechnical logging, down-hole and GPS surveys and all sampling were carried out to a high standard by First Nations members under supervision of the site geologist.

5.6 LOCAL RESOURCES

Few local resources have been identified to date. In particular there is little evidence of aggregate an essential commodity for mine and infrastructure development.

There is sufficient space on the current property to develop a mine and ancillary installations.

5.7 PHYSIOGRAPHY

The Big Daddy (average elevation 173 m) property lies near the western limit of the Hudson Bay Lowlands, a vast, poorly-drained area extending along the south and west coasts of James and Hudson Bays between the Ontario-Quebec boarder and Churchill, Manitoba (e.g., Brookes, 2010). The area corresponds with the maximum extent of periglacial marine inundation (to 180 m) by the ancient Tyrrell Sea and with that of the western margin of the Hudson Bay Platform. The Platform comprises Lower Paleozoic carbonates and clastics sediments. Remnants of the Paleozoic platform cover strata were reported in drill holes FW-06-04 and FW-08-11, which are collared within 200 m of the chromite sub-outcrop, as well as in holes drilled to the east of the Big Daddy deposit.

Sjörs (1959) describes four distinct landscape features; fens, bogs, black spruce islands and riparian zones. All are evident on the Big Daddy claim:

- Fens are the basic landscape feature characterized by shallow, typically circular ponds, relatively diverse vegetation, higher pH and metal contents.
- Bogs comprise island-like, thick sphagnum accumulations (~3 m) above the local surface with irregular, 1.5 m-deep ponds that form string bogs where gradients are steeper. Plant diversity is low due to acid, nutrient-poor water.
- Ovoid, black spruce islands are elevated 2 m above the surrounding area, commonly with treeless centres. Sjörs (ibid.) encountered frozen ground a metre or so below surface.
- The riparian zone comprises river banks including the area subject to seasonal flooding. Nutrient availability and locally good drainage contribute to a diverse flora including, locally, mature spruce and aspen.

5.8 RELIEF AND DRAINAGE

The Big Daddy project area lies in the Attawapiskat drainage system which consist of one of the two great rivers (the other being the Albany) that drain Northwestern Ontario. These provided convenient access for early explorers and traders. Drainage over the Lowlands is very poor due to the gentle slope (approximately 0.7 m / km).

Relief across the Big Daddy claim is about 4 m, and as much as 7 m above the closest points on the Muketei and Attawapiskat rivers. Water flow along creeks and rivers varies from a maximum in the spring falling gradually until the following spring. During the remainder of the year even local rainfall rapidly reaches the major drainages causing slight increases in water level.

The Big Daddy claim drains to Black Creek which straddles the east claim boundary. From there drainage is north-northwest towards the Muketei River.

6.0 HISTORY

There is no evidence that the current property had been staked or otherwise explored prior to 2003, however, Spider/KWG and others conducted diamond exploration in the area from the early 1990's.

Government survey agencies have carried out very limited, largely reconnaissance work in the area due to the perceived lack of outcrop and the high cost of supporting field programs. Prior to the 1990's there are few records of past exploration beyond a flurry of diamond drilling to the north and west of the current area in the early 1970's following the Kidd Creek (Timmins) VMS discovery.

6.1 GENERAL

Robert Bell (1886) provides the earliest account of the geology of the Attawapiskat and tributaries describing well-exposed Paleozoic stratigraphy along the river and including initial description of Archean rocks exposed in the headwaters of the river. McInnes (1910) travelled along the upper Attawapiskat and adjacent Winisk rivers a quarter century later. During the 1940's the Provincial Government investigated the lignite, gypsum and petroleum possibilities of the James Bay Lowland, drilling several drill holes to basement (e.g., Martison, 1953). The GSC completed regional mapping of the Hudson Bay Platform during the 1970's (e.g., Sandford & Norris, 1975). Although Bostock's (1968) work was of regional scope, he and colleagues reported much outcrop along drainages from the Muketei westwards.

Diamond explorers, Monopros (a subsidiary of De-Beers) and Selco, traced diamond indicator minerals from initial discoveries in the Kirkland Lake area into the Lowland in 1962 culminating in the discovery of the Jurassic-aged, Attawapiskat kimberlites in 1989. In 1971 Inco, Sherritt Gordon, Denison and Kennco drilled base metal targets to the north and west of the current area. During the mid-1990's the then Spider-KWG joint venture tested potential kimberlite targets over a 200 km square area centred on McFaulds Lake, quickly discovering the five, Proterozoic age Kyle diamondiferous kimberlites under Paleozoic cover. Elevated chromite counts were reported in drainage and overburden samples collected during this period marking the earliest report of chromite in the area (Gleason and Thomas, 1997).

The 2002 discovery of chalcopyrite by DeBeers and recognition of VMS mineralization in 2003 by Spider and KWG focused exploration attention in the McFaulds area prompting Richard Nemis and Freewest to cause the claims comprising the current property to be staked. Howard Lahti, PhD, P. Geo., was first to recognize chromite in situ noting two thin beds in drill hole FW-06-03. The Eagle One Ni-Cu-PGE discovery in 2007 precipitated intense exploration effort over the following two years during which time the Blackbird, Big Daddy, Black Thor, Black Creek and Black Label chromite and the Thunderbird vanadium deposits were discovered, Initial resource estimates have been made on all but the last two mentioned deposits (see Section 15 on Adjacent Properties).

6.2 PROPERTY HISTORY

Spider has managed exploration since inception, latterly through Billiken Management Services, Inc. In mid-2007 Billiken was sold to an unrelated party, thus Spider and Billiken have operated at arms-length for almost three years.

Early exploration programs (airborne surveys and ground follow-up) were conducted over contiguous properties. Costs were apportioned according to the work done over each property. For this reason the J (Big Daddy) grid extends over the adjacent properties.

At some time prior to 2007 Probe Mines held an option on the Freewest Claims, completing three short diamond drill holes (F-1 to F-3) that narrowly missed the chromite mineralization. The claims were returned to Freewest before Probe was vested.

The past exploration history, which was reported in the previous 43-101 report, has been summarized in Table 6.1.

6.3 HISTORIC PRODUCTION

The property has no historical resource or reserve estimates and there has been no prior production.

Table 6-1. Summary of Exploration Completed on SKF Property between 1995 and 2008

Year	Company / Contractor	Work completed	Results
Pre-1995		Assessment file search.	No work filed in Spider 3 area.
1995-1996	SPQ & KWG (Burns, 2005)	Fixed wing mag over Spider #3 area. Helimag over 48 targets. Ground mag over selected targets. Modern alluvium sampling. Limited bedrock mapping. Airphoto interpretation. Two diamond drill holes	McFaulds magnetic anomaly detected and detailed [Selected datasets sold to OGS (Operation Treasure Hunt) and subsequently released as MRD.]
2001-2002	DeBeers (Burns, 2005)	McFaulds anomaly designated SP3-0029. Helicopter mag & EM (N-S on 50 m lines, 20 m elevation 1.6 km sq. Ground mag (N-S, 50 m lines). Vertical reverse circulation drill hole (SP3-02-007R),	Results for SP3-0029. • Apparent resistivity correlates with mag over principal anomalies. • Anomaly detailed. • Cut 1.75% Cu/6.5 m below regolith (saprolite) from 25.0 below surface.
2003	Freewest	AeroTEM Fugro	Found numerous EM targets in an arc including McFaulds Lake and extending to south and west.
2003	R Nemis & Freewest	Claim staking	Staked current property
2003	Freewest	Line cutting (J & H); ground mag and Max-Min (Scott Hogg)	Detailed ground targets on J and H grids
2004	Freewest	FWM-04-01 (Grid H, L 37 E, 5+50 S; Az ~120 °, Dip -45°;P 3008269)	19 m overburden, 10 m unknown, 67.5 m gabbro and 93.5 m tuff (mineralized, VMS style pyrite). The gabbro reports elevated Cr & Ni.
2006	SPQ & KWG (Novak, 2006)	FW-06-02 (Grid H) FW-06-03 (353.5 m) & 04 (Grid J) to test coincident HLEM-magnetic anomalies.	Cut ~10 m sulphide mineralization in a fragmental pile. Howard Lahti noted two "massive chrome beds" [1.05 & 0.6 m] in FW-06-03 marking the discovery of in-situ chromite in McFaulds area. FW-06-04 cut hanging wall volcanics containing locally anomalous Ni (but not Cr) concentrations.
2008	SPQ & KWG	FW-08-05 to 23	Drilling defined the chromite mineralization for 400 m along strike and also tested nearby EM anomalies

Note: SPQ = Spider

7.0 GEOLOGICAL SETTING

The edge of the Hudson Bay Platform also marks the maximum transgression (180 m above sea level) of the ancient Tyrrell Sea and of deposition of several metres of thixotrophic, fossil-bearing mud.

The property lies at the western edge of the preserved flat-lying, Lower Paleozoic Hudson Bay Platform, remnants of which were observed on the current property. The Hudson Bay Platform comprises Ordovician to Cretaceous sedimentary strata which reach a maximum known thickness of about 2,500 m in Hudson Bay. Two holes contain saprolite, indicative of an early Paleozoic tropical weathering event (Patrick Chance, personal communication, 2010).

The property lies in the Sachigo greenstone belt of the Oxford–Stull Domain (Stott and Rainsford, 2006) of the Sachigo Subprovince (Figure 7-1). The Sachigo greenstone belt is arcuate, west-facing and 100 km long by 5 km to 25 km wide belt. It is in intrusive contact with granodiorite rocks to the north and west (Atkinson et al., 2009). The Oxford-Stull Domain also contains a series of significant mafic to ultramafic intrusions including Big Trout, Springer, Highbank and McFaulds. Those at Big Trout and Highbank exhibit magmatic layering a characteristic of fertile mafic complexes.

7.1 LOCAL GEOLOGY

Due to poor access, lack of abundant outcrop and limited mapping, local geology has been largely interpreted from airborne geophysical data and constrained by limited and selective diamond drilling (Figure 7-2). The area is underlain by volcanics of the Sachigo belt into which the Ring of Fire mafic-ultramafic complex is intruded. The Ring of Fire complex comprises three elements; the feeder dyke within which the Eagle One Ni-Cu-PGE deposit is contained, the sill or sills containing stratiform chromite deposits, here called the McFaulds Lake Sill, and the ferrogabbro bodies that contain the Thunderbird Fe-Ti-V prospect.

Petrographic and chemical evidence from the Big Daddy property (Scoates, 2009-03) indicate that the McFaulds Lake Sill is a well fractionated, body comprising lower (to the northwest) olivine-rich units overlain by upper olivine-poor units. The principal Big Daddy chromite bodies lie at the top of the olivine-rich unit.

The McFaulds Lake mafic-ultra-mafic sill (elsewhere termed the Ring of Fire intrusion) has been intermittently emplaced along a granodiorite-greenstone contact over a 20 km length of which 15 km between Eagle 2/Blackbird 1 (in the southwest) and Black Thor/Black Label (in the northeast) are known to be mineralized. The Thunderbird vanadium deposit occurs in ferrogabbros which form a distinct magnetic anomaly that lies parallel to and east of the main McFaulds Sill about 9 km northeast of the Big Daddy deposit.

Figure 7-1. Regional Geological Setting of the Superior Province,

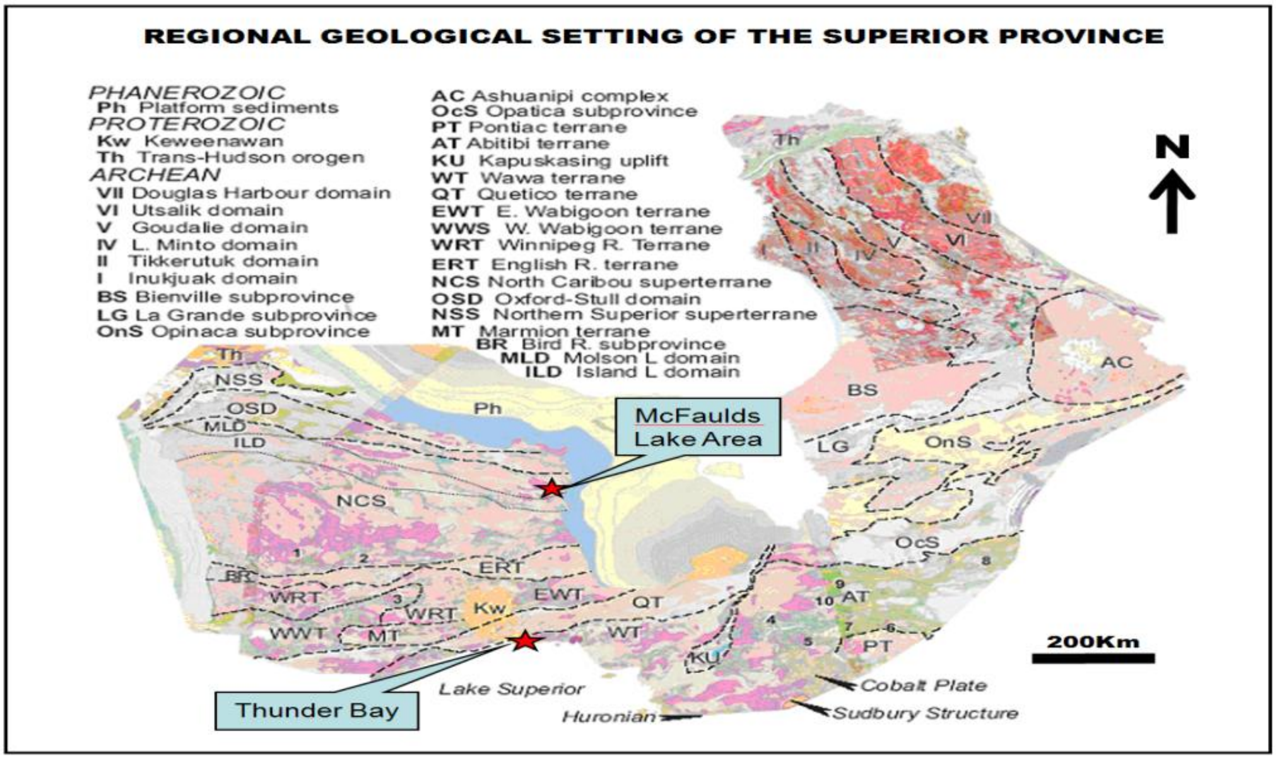
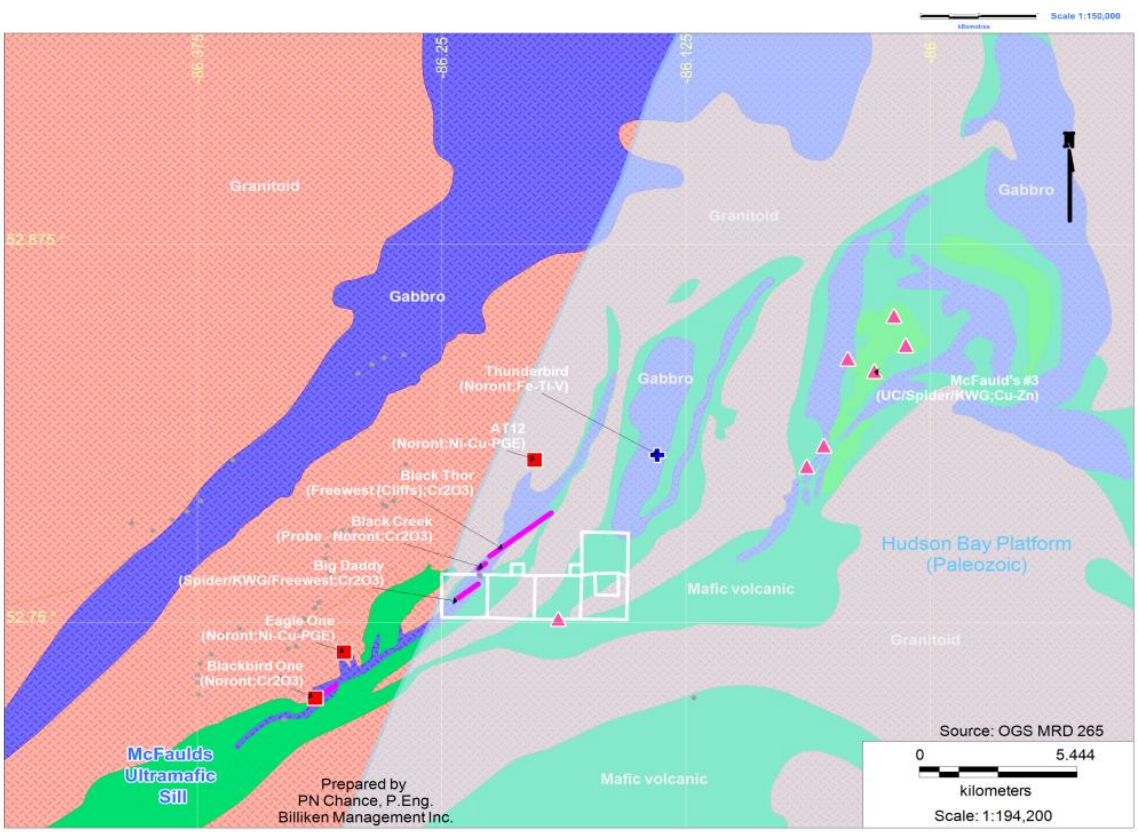


Figure 7-2. Local Geology of the McFaulds Lake Sill showing the Big Daddy Chromite Occurrence.



Symbols: Red - Ni-Cu-PGE, Purple lines - Cr_2O_3, Blue – Fe-Ti-V & Pink VMS.
Modified from OGS MRD 265.

Volcanic rocks in the McFaulds Lake area have a U/Pb zircon isotopic age of 2737 ±7 Ma which is comparable with ages from other parts of the Superior Province of the Canadian Shield (Stott, 2007). It is older than most parts of the Abitibi belt, but similar in age to greenstone belts in Wabigoon and other belts.

7.2 PROPERTY GEOLOGY

The interpreted geology of the property is based on drill holes and ground geophysics (Figure 7-3). Bedrock is obscured by a relatively thin (approximately 10 m) layer of marine clay with the exception of two small areas of peridotite outcrop that straddle the creek near the north property boundary.

Glacial overburden over the deposit area is typically 6 m to 10 m thick but can be as little as 1.6 m (drill hole FW-09-30). It comprises marine clay with a few pebbles and cobbles at the bedrock surface. Locally, overburden may be as much as 13.4 m thick (drill hole FW-08-05).

Saprolite was reported in two holes (FW-04-01 and FW-09-45) drilled on EM targets off the sill. Oxidation (assumed to be due to deep, early Paleozoic weathering) is commonly observed to 50 m below surface but hematite has been reported as deep as 250 m.

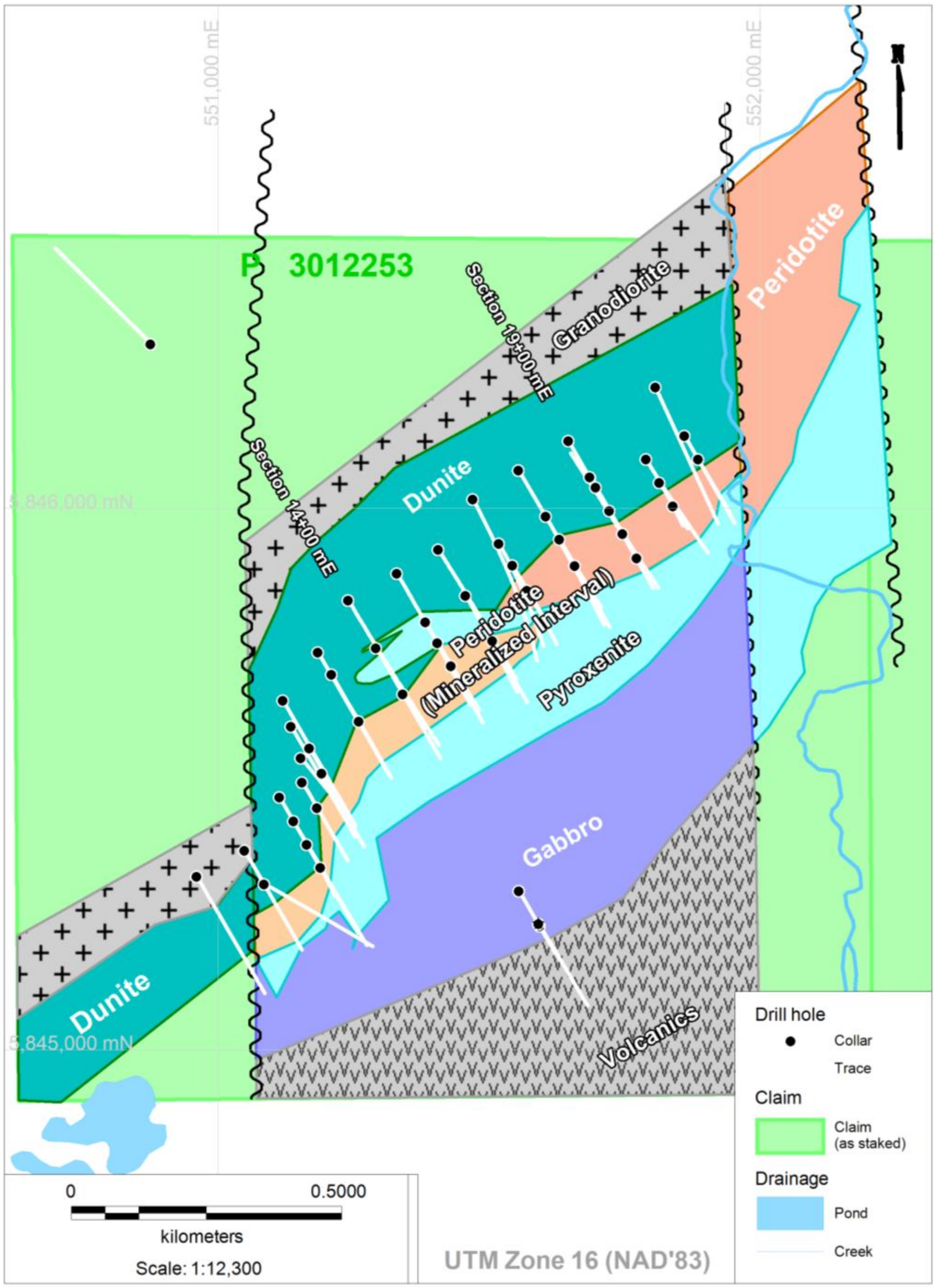
Drilling and geophysical data suggest that the sill segment containing the Big Daddy deposit is about 1,000 m thick. Limited information suggests that the sill thins to the southwest. Observed geologic contacts and limited igneosedimentary structures (e.g., bedding) indicate that the sill has been rotated from an original horizontal to a nearly vertical to overturned position.

Silicate minerals within the sill have been pervasively altered to serpentinite (serpentine-talc-chlorite), however, original textures are well preserved in both hand specimen and thin section.

Sill stratigraphy, comprising lower (to the northwest) olivine-rich and upper (to the southeast) olivine-poor units, indicates that the sill is strongly fractionated and that the top is to the east (Scoates, 2009-03). The olivine-rich units comprise a lower marginal pyroxenite, dunite, peridotite and chromitite. Overlying olivine-poor units are relatively Cr-poor comprising pyroxenite and gabbros which were observed in intrusive contact with overlying volcanics.

The dunite is typically coarse grained and dull green. While the grain size varies there is little evidence of disruption. Magnetite occurs as rims around former olivine grains, as diffuse patches and in narrow (~1 cm wide), massive veinlets. The latter are strongly conductive. The abundance of magnetite and presence of narrow but highly conductive magnetite veinlets produce large amplitude total magnetic fields and diffuse but persistent AEM anomalies that extend from the Big Daddy claim, northwards onto the Black Creek (Probe) and Black Thor-Black Label properties.

Figure 7-3. Bedrock Geology of the Big Daddy Deposit Based on Drill Hole and Ground Geophysics (UTM Zone 16;NAD'83).



The peridotite is chaotic in appearance, being marked by abrupt grain size changes. Scoates (2009-03) describes an extensive, magmatic breccia unit that reflects a high energy magmatic environment possibly occupying a feeder dyke. Massive chromite fragments were observed in earlier (pre-2009) holes (ibid.) but were rare in subsequent holes.

The peridotite unit also contains the economically significant chromite mineralization of which two intervals were typically observed. The stratigraphically lower unit(s) are characterized by variable (from interval to interval) chromite contents between 15% and 40% Cr_2O_3. The upper massive unit comprises uniform, ~40% Cr_2O_3, grades, often within 1% over tens of metres. The grade of the upper unit is consistent over the deposit with the exception of the southwest part where grades drop to ~38% Cr_2O_3.

Drilling of the Big Daddy deposit has been carried out from footwall to hanging wall so that the peridotite has been well sampled. The unit is marked by frequent faulting and fracturing reflected in poor recoveries, lower RQD's and evidence of deep weathering. While the faulting and fracturing may be important in mine design through-going faulting is not required to resolve continuity between holes or sections. It is suggested that these faults reflect mechanical discontinuities between relatively unaltered massive chromite and pervasively altered, soft, host rocks (Patrick Chance, personal communication, 2010).

The upper contact of the massive chromitite with olivine-poor pyroxenite is sharp, occurring over as little as a centimetre. The pyroxenite comprises a distinctive pale green unit in which pseudomorphs after pyroxene are distinctive. In addition the Cr_2O_3 contents drops from ~40% to less than 1% across this contact.

Gabbros, some in contact with overlying volcanics, were reported in several holes.

Volcanic hanging wall rocks were not encountered during the recent drill campaign. Work on the McFaulds Lake volcanogenic massive sulphides suggests that they reflect a back arc environment (Jim Franklin, personal communication, 2010).

The Big Daddy appears to be contained between north-trending, left lateral faults near section 1000 E and 2100 E where geophysical anomalies appear to be truncated and along which the Black Creek deposit is shifted.

8.0 DEPOSIT TYPES

Primary/orthomagmatic chromite occurs in two types of deposits, stratiform and podiform. These both have comparable mineralogy but contrasting origins. Residual and transported deposits are additional but rarely significant producers (WIM, 2008). The Big Daddy chromite is a typical stratiform deposit by virtue of its setting, host rock lithologies, mineralogy and dimensions.

The current major producers are all stratiform and occur in sills typically emplaced in stable continent environments. Productive sills include the Bushveld (South Africa), Great Dyke (Zimbabwe), Sukinda (Orissa, India), Kemi (Finland) and Ipuera (Brazil).

The collectively important but individually minor podiform deposits occur as very small pods (median tonnage 20,000 t; Singer et al., 1986) in the tectonized base of obducted ophiolites. These deposits are preserved in younger mountain ranges including the Tethyan orogen from the Balkans, through Turkey to Pakistan and India. Similar deposits occur in the North American Cordillera in northern California and Oregon. In exceptional environments, larger, multimillion tonne, podiform deposits have developed (e.g., Kempirasai, Kazakhstan).

Residual secondary deposits are locally significant producers (e.g., Sukinda). Locally accumulations in beach sands may be significant (e.g., Oregon), however, these tend to have low Cr:Fe ratios making them problematic to market.

Stratiform deposits account for 45% of total world chromite production and 95% of reserves. The Bushveld alone accounts for 35% of production. Other significant producers are the Great Dyke, Kemi and Brazilian deposits, which together produce about 10% of the world's total. The many small scale podiform deposits produce the remaining 55% of chromite which enters the market as ores rather than ferrochrome.

8.1 RELATED DEPOSITS

The shear size, emplacement and crystallization processes associated with ultramafic sills give rise to an important group of four related deposit types, of which three have been found in the McFaulds Sill; magmatic massive sulphides (MMS: Ni-Cu-PGE's), stratiform chromite, Fe-Ti-V, and reef-type, low sulphide, PGE deposits (not yet found in the McFaulds Lake area).

MMS deposits (e.g., Eagle One, Voiseys Bay) represent the accumulation of sulphides in traps in the floors of feeder dykes below the main sill. The remaining deposits occur within the cooling sills under a set of crystallization conditions that favour the economically important minerals.

Additional details are available at several on-line sources including USGS (Cox and Singer, 1998), GSC (e.g. Eckstrand and Hulbert, 2008) and BC Department of Mines (Lefebure et al., 1995).

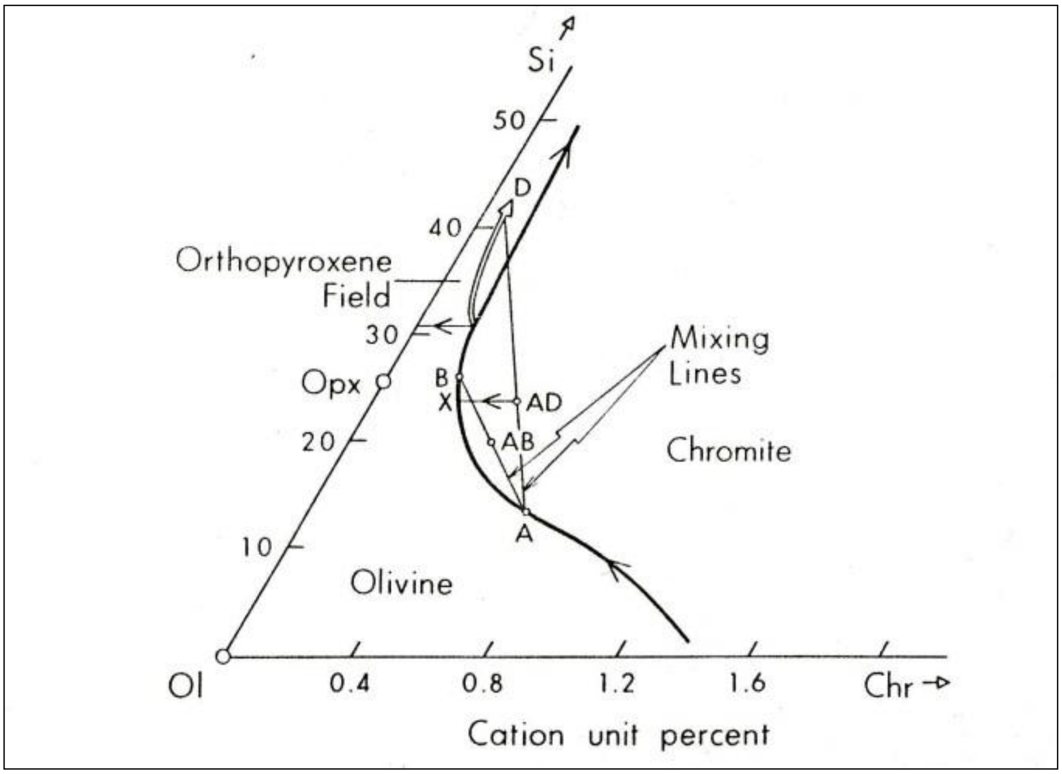
8.2 GENETIC MODEL FOR STRATIFORM CHROMITE

Stratiform chromite deposits are formed by magmatic segregation during fractional crystallization (fractionation) of mafic-ultramafic magma. Stratiform chromite deposits require that chromite be the major and ideally the sole crystallizing phase over an extended period. Irvine (1975, 1977) suggested two mechanisms whereby a chromite saturated picritic tholeiite liquid becomes more siliceous either by contamination (assimilation) with granitic and/or volcanosedimentary material or alternatively by mixing with a more siliceous differentiate of the parent magma, thereby causing chromite to precipitate in the absence of silicate minerals.

On the evidence of field relations and mineralogical data (Jackson 1961, von Gruenewaldt 1979) combined with isotopic studies (Kruger and Marsh 1982, Sharpe 1985, Lambert et al. 1989) it has been shown that large layered intrusions are not the result of single, one-event injections of magma, but are the result of repetitive inputs. Irvine (1977) demonstrated that if a new input of magma was injected into one that had reached a higher level of fractionation, the resultant mixing action could inhibit the fractional crystallization of silicate minerals such as olivine and orthopyroxene and permit the crystallization of chromite alone. This is the mechanism by which layers of massive chromitite can develop, without dilution by cumulate silicates. As illustrated in Figure 8-1 (after Irvine 1977), the mixing of liquid A which is on the olivine – chromite cotectic, with liquid D on the orthopyroxene field may, provided that points on the mixing line lie above the liquidus surface, culminate in a hybrid magma such as AD which will intersect the liquidus in the chromite field on cooling. Hence it will crystallize chromite alone while it moves to point X on the olivine – chromite cotectic, and thereafter it will continue to crystallize chromite and olivine. It has been shown that the decrease in the solubility of chromite in basaltic magma in equilibrium with chromite per degree centigrade fall in temperature is greater at high (1,300˚C – 1,400˚C) than at low (1,100˚C – 1,200˚C) temperature. Due to this concave – upward curvature of the solubility curve, the mixing of two magmas at different temperatures saturated (or nearly saturated) in chromite places the resultant mixture above the saturation curve, which suggests that point AD in Figure 8-1 is likely to lie above the liquidus.

The suggestions by Irvine (1977) are consistent with observations on chromitites in layered intrusions. Most significant amongst these observations is the fact that most of these chromitite layers occur at the base of well defined cyclic units (e.g. Bushveld Complex and Great Dyke in Southern Africa) or at/near the base of similar cyclic units. Further evidence comes from the textures of the underlying rock units which indicate a common cotectic crystallization of chromite with olivine or orthopyroxene showing that the magmas previously in the chambers were saturated with respect to chromite.

Figure 8-1. Phase Relations in the System Olivine-Silica-Chromite as determined by Irvine (1977). (Illustrating the consequence of mixing primitive magma (A) with well fractionated (D) and slightly fractionated (B) variants of the same primitive magma



(Source: Naldrett et al., 1990))

More recently, the crustal contamination hypothesis has been supported by MELTS (Ghiorso and Sack, 1995; Asimow and Ghiorso, 1998) thermodynamic modelling software and textural observations of xenolithic clasts of iron-formation occurring stratigraphically below the massive chromitite layers within the RFI. Workers investigating similar deposits such as the Ipueira-Merado Sill determined, supported by isotopic and textural observations, that crustal assimilation by a primitive and chrome enriched magma was the most likely cause for the formation of the chrome deposit (Marques et al., 2003).

Scoates (2009) speculates that both mixing of primitive magma with fractionated magma (Irvine, 1977) and crustal contamination of the parental magma (Irvine, 1975; Alapieti et al., 1989; Rollinson, 1997; Prendergast, 2008) appear to have had complementary roles in the formation of the Big Daddy chrome deposit. The hanging wall volcanics include both banded iron formation intervals and volcanogenic sulphide accumulations which, if assimilated by the sill, could alter magma chemistry sufficiently to deposit chromite.

8.2.1 Association of Ni-Cu-PGE with Stratiform Chromite

Stratiform chrome deposits are commonly associated with magmatic Ni-Cu-PGE mineralization. For sulphide precipitation to occur, the silicate liquid in the magma chamber must become sulphur-over saturated and this is dependent upon the following factors:

- Melt temperature.
- Oxygen fugacity.
- Magma composition – MgO/FeO ratio, SiO_2 content, and S content.
- Magma recharge

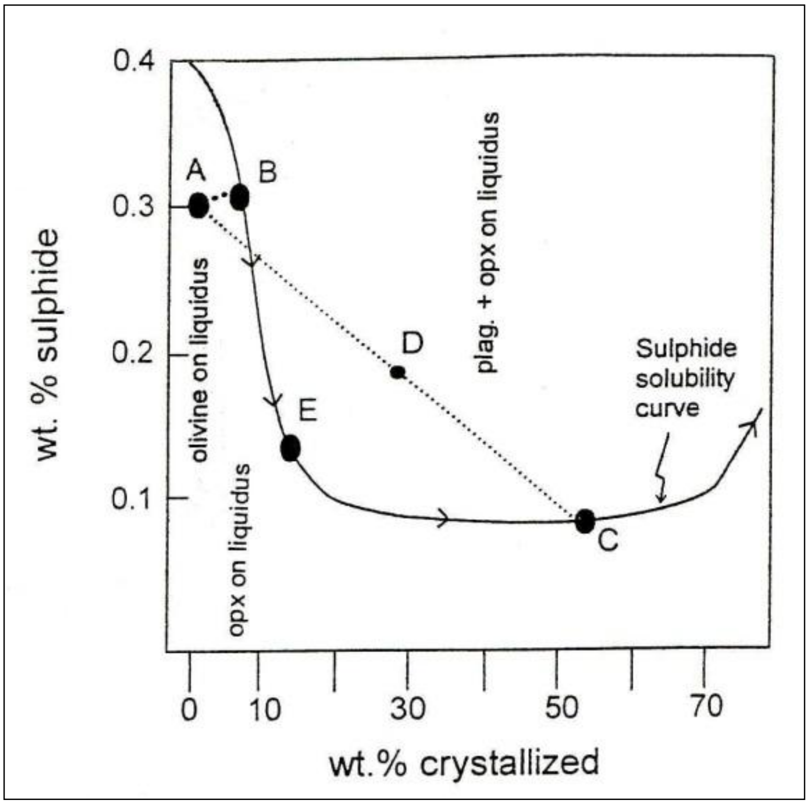
As far as magma mixing is concerned, it is generally accepted (Campbell and Turner, 1986) that layered intrusions have formed through repetitive inputs of magma. These inputs are likely to have been turbulent and thus to have involved significant entrainment and mixing of resident magma within the input. The resulting hybrid would also spread out at the appropriate density level to give rise to turbulently convecting layers. If sulphides formed in the hybrid at this stage, the turbulent mixing and convection would have provided the ideal environment in which they could have developed a high R-factor, and thus have become enriched in PGE. The R factor is defined as the ratio of silicate melt to sulphide melt during sulphide segregation.

Sulphide saturation may be achieved in one of three ways as proposed by Naldrett et al. (1990):

- Fractional segregation where sulphide saturation is attained through fractionation (Figure 8-2).

- Batch segregation where batch segregation of sulphide is achieved through mixing of a primitive magma with an evolved resident magma that is close to crystallizing plagioclase (Figure 8-2).

Figure 8-2. Variation in Solubility of Fe-sulphide in Differentiating Basaltic Magma. (Modified after Naldrett & Von Gruenewaldt, 1989. (Source: Maier et al., 1998))



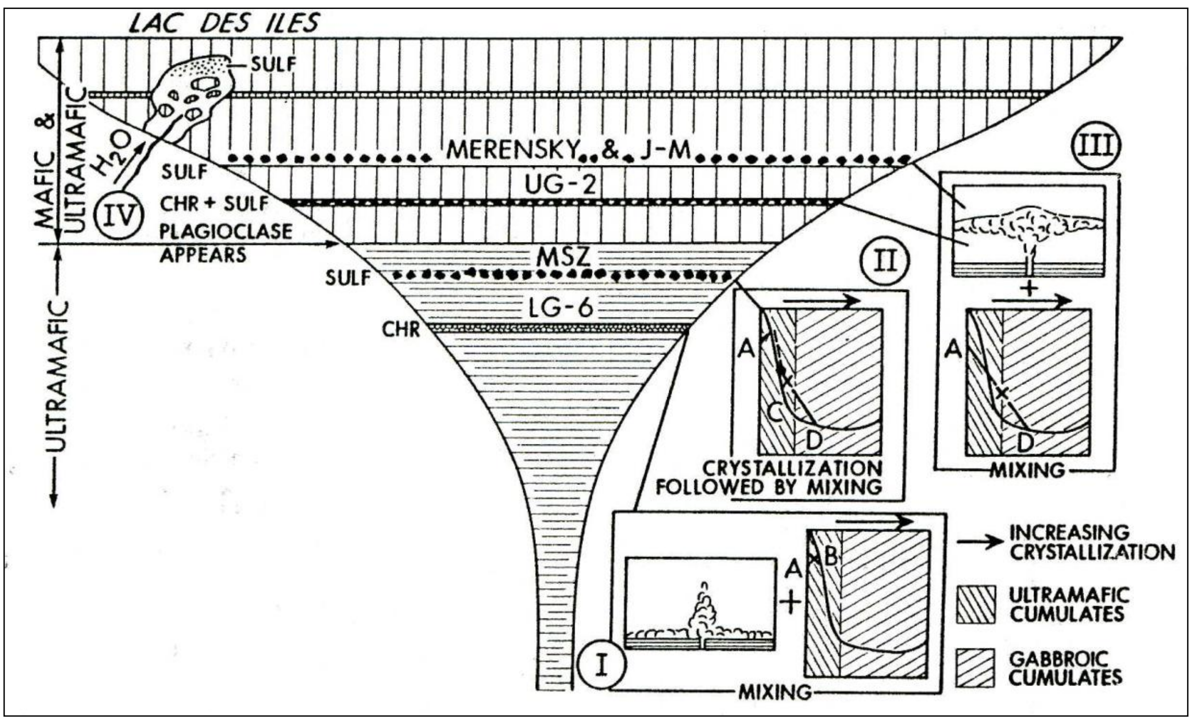
- Constitutional zone refining where sulphide saturation is preceded by volatile-induced partial melting and remobilization of cumulates and sulphides (Figure 8.3, example iv).

The above three processes lead to the formation of different types of deposits as illustrated in Figure 8-3. Subsolidus and deuteric processes are responsible for the modification of the original primary textures in these deposits.

It is important to note that the mixing of fresh primitive magma with that resident in an intrusion can give rise to a chromitite formation regardless of the degree of fractionation of the resident magma, whereas extensive segregation of sulphide will only occur as a consequence of this type of mixing close to or after the stage at which plagioclase saturation has been achieved by the resident magma.

Diagram shows the types of chromitite and PGE-enriched sulphide deposits that can result from fractional crystallization, magma mixing and constitutional zone refining. Mixing of resident magma with primitive magma before plagioclase has appeared on the liquidus of the former is likely to produce sulphide- and, therefore, PGE- poor chromitite (Example I); fractional crystallization may give rise to a PGE-rich layer not associated with the base of a cyclic unit (Example II); mixing of resident magma with more primitive magma after plagioclase is crystallizing from the former may give rise to sulphide- and, therefore, PGE-enriched chromitites or PGE-rich sulphide layers (Example III). Volatile-induced partial melting of cumulates can give rise to constitutional zone refining and the concentration of PGE at the point at which the partial melt becomes saturated in sulphide (Example IV). (Re-drawn after Naldrett et al., 1990).

Figure 8-3. Cross-section through a Hypothetical Layered Intrusion.



9.0 MINERALIZATION

9.1 OVERVIEW

The accumulation of chromite on the Big Daddy property depended on two processes. First, emplacement of the McFaulds Sill along a then near-horizontal contact between underlying granodiorite and overlying volcanic and sedimentary strata; and second, maintenance of the magma temperature and magma composition such that only chromite could crystallize over a prolonged period.

9.2 LOCALIZATION

The chromite mineralization of the Big Daddy deposit and similar discoveries such as the Black Thor and Black Label in the northeast and the Blackbird in the southwest (Figure 7.2) is hosted in the ultramafic unit (i.e. peridotite) of the McFaulds Lake Sill. Mineralization in the Big Daddy segment of the McFaulds Lake Sill occurs within a 65 to 180 metre thick, often brecciated peridotite interval lying stratigraphically above a dunitic footwall and below a pyroxenite hanging wall. The lower contact of mineralization tends to be gradational while the upper is sharp.

Mineralized rock comprises sub-millimetre-diameter, idiomorphic, cumulate, chromite grains. Mineralized intervals are a mixture of chromite and olivine crystals set in a fine grained peridotitic matrix. At lower Cr_2O_3 contents chromite grains are disseminated through the host rock. As concentration increases, bedding becomes evident but disappears at the highest grades (>35%Cr_2O_3) due to uniform crystal size and absence of silicate diluents.

The bulk of the Big Daddy chromite mineralization is manifested as a persistent tabular zone of massive chromite with distinct hanging and footwall contacts and with grades typically >35% Cr_2O_3.

9.3 DISTRIBUTION OF CHROMITE GRADES

Based on information derived from drill hole logs and assay data sheets, the Cr_2O_3 grades are distributed as shown in Table 9-1. In a generalized section, three broad grade-texture zones are evident. The onset of mineralization is marked by intermittent accumulations of heavily disseminated material with occasional massive beds. Stratigraphically above this zone, grades tend to be lower until the massive unit is reached. Grades in the massive unit are consistent and universally high (>40% Cr_2O_3) but fall slightly (36 to 38% Cr_2O_3) in the southern end of the deposit where pyroxene oikocrysts are indicative of lower grades.

Table 9-1. Distribution of Cr_2O_3 Grades.

Mineralization Type	%Cr_2O_3	Remarks
Massive	30 – 50	Dominant type
Banded	20 – 30	Rare type. Individual bands may contain up to 40% Cr_2O_3
Semi-massive	20 – 30	Very minor type
Heavily disseminated	10 – 20	Locally common
Disseminated	1 – 10	Locally common [Background values]

9.4 SULPHIDES AND PGE

Massive sulphides have not been encountered in the chromite-rich zones. However, local sulphide disseminations have been noted within and immediately above the massive chromite layers. The identifiable sulphides are pyrrhotite, chalcopyrite, pyrite and rarely pentlandite.

A small (<10 cm diameter) sulphide-rich accumulation from hole FW-09-33 reported a massive, secondary Fe-Cu-Ni-sulphide assemblage (godevskite, Ni_9S_8 and mackinawite, $(Fe,Ni)_9S_8$ with minor chalcopyrite, chromite and trace millerite (Kjarsgaard, 2009), in a fault or shear zone. This assemblage is typical of low-temperature, hydrothermally emplaced nickel-iron sulphides.

10.0 EXPLORATION

The pre-2009 exploration is summarized under History (Section 6, Table 6.1) and was also described in detail in Micon's (2009) previous report. The following outlines results of the most recent exploration campaigns which follow Micon's (2009) recommendations.

10.1 2009-2010 EXPLORATION

Recent exploration programs reflect implementation of Micon's 2009 recommendations.

10.1.1 QA/QC

In early 2009, Spider/KWG retained Tracy Armstrong to review the assay data set, make recommendations for replicate analyses, review the analytical methods used and recommend appropriate standards and control sample methodologies to ensure quality and to recommend protocols to meet Spider/KWG's objective of rapidly acquiring the high quality data required to fully value the deposit.

Ms. Armstrong concurred with the adoption of XRF as the project's standard method for chrome analyses. She identified several problematic batches which were re-analyzed, and she designed comprehensive QA/QC protocols and supervised the preparation and certification of standard materials (BD-1, DB-2 and BD-3) prepared from coarse rejects from previously submitted samples.

10.1.2 Evaluation of PGE – Potential of Hanging Wall Pyroxenite

During the late summer of 2009, Howard Lahti completed a comprehensive resampling program focusing on PGE's in the hanging wall pyroxenite, taking almost 500 samples. These data show locally anomalous intervals containing up to 2 g/t Pt + Pd, however, there was no evidence of consistently mineralized interval that might reflect potential for a Merensky or Stillwater-style reef. Both these and subsequent data show a marked increase in PGE contents in the upper couple of sample intervals in the massive chromite.

10.1.3 Ground Geophysical Surveys

During 2009 and 2010, gravity, magnetic and pulse EM surveys were completed over the central portion of Grid J. In addition, a Max-Min survey was completed over a small oblique grid cut over the T-11 airborne target in the southwest corner of the property. A grid was also cut over the T-2 target and a hole (FW-09-45) was spotted using existing data.

In early 2009 the J grid was re-chained. Geosig (2009) then completed precise (+/- 0.1 m) Real Time Kinematic GPS levelling, gravity and gradiometer surveys. The gravity data, which were refined by modelling (e.g, Reed, 2009), show a distinct positive anomaly gaining width and magnitude from line 900 E to 1400 E and then continuing to about 2100 E where it is abruptly truncated. Total magnetic intensity data show broad areas lying adjacent to and immediately north of the gravity anomaly.

In late October, 2009, Crone completed pulse EM surveys based on seven loops centred on the gravity anomaly and extending to the north property boundary. Extension of the survey over the southwest corner of the property and across the creek near 2100 E was not possible due to late freeze-up. The survey detected a diffuse but persistent anomaly adjacent to and northwest of the gravity anomaly, coinciding with the total field magnetic anomaly above and earlier airborne anomalies that persist northwards across the Probe property and onto the Freewest property. Hole FW-09-46 was collared in massive chromite and drilled northwest into the sill footwall where it cut a wide interval of magnetite-bearing, serpentinized dunite containing occasional massive magnetite-filled veinlets which were found to be highly conductive.

In January, 2010 Max-Min was completed over the T-11 grid situated in the southwest corner of the property. The data collected were ambiguous. The most significant response was a broad and diffuse anomaly evident only in the higher frequencies suggesting an overburden source. No hole was completed in this area.

10.1.4 T-2 Target

The T-2 target lies on the north property boundary, extending onto the western excised claim unit. Airborne magnetic data suggested a strong, strike-parallel, magnetic feature that extends onto SKF property where it bifurcates and weakens.

A single hole, FW-09-45, tested the target, returning a broad (16 m) interval of pyritic, interflow cherts and volcaniclastics containing trace amounts of chalcopyrite in ampibolitic, fragmental volcanic strata.

10.2 DELINEATION STAGE - 2009/2010 DRILLING

Drilling was completed in two campaigns; late September to mid-November, 2009 and January to early February, 2010. A total of 32 holes were collared on the Big Daddy deposit two of which did not reach the deposit hanging wall. One, FW-09-44, was abandoned due to poor drilling conditions. The second, FW-10-52, was suspended at 195 m prior to intersecting mineralization due a blockade by First Nations.

Holes were spotted and aligned relative to grid pickets. Routine down hole directional surveys with Flexit and Deviflex suggested minimal deviation (<6 m/100 m). North-seeking gyro surveys, run on several casings, generated refined initial azimuths and reported drooping of casings due to low-strength overburden. All casings were subsequently surveyed using a Timble Pro-XRT with an Omni Real-Time Correction (RTC) signal activated, providing accuracies of better than +/-0.4 m.

Logging was enhanced with the adoption of a standard project legend, adoption of GeoTic© for data capture and use of a Niton hand-held XRF to aid discrimination of chromite grade.

In addition magnetic susceptibility, specific gravity, recovery, RQD and additional geotechnical parameters were collected for all holes.

Initial holes (2009) designed to confirm continuity of the deposit were drilled in pairs on sections 100 m apart. Eventually, a third deep hole was added. Intermediate (50 m spacing) holes were added where additional hanging and footwall contacts were required. Many of the in-fill holes that Micon will recommend had been planned for the 2010 campaign but were not drilled due to delays and uncertain supplies due to the blockade initiated by the communities of Marten Falls and Webequie.

Eventually 32 drill holes were completed with between two and four holes per section, spaced 50 or 100 m apart. Section lines are 100 m apart. The layout is depicted in Figures 11.1 and 17.2 and covered a total strike length of 1 km down to a maximum depth of about 365 m.

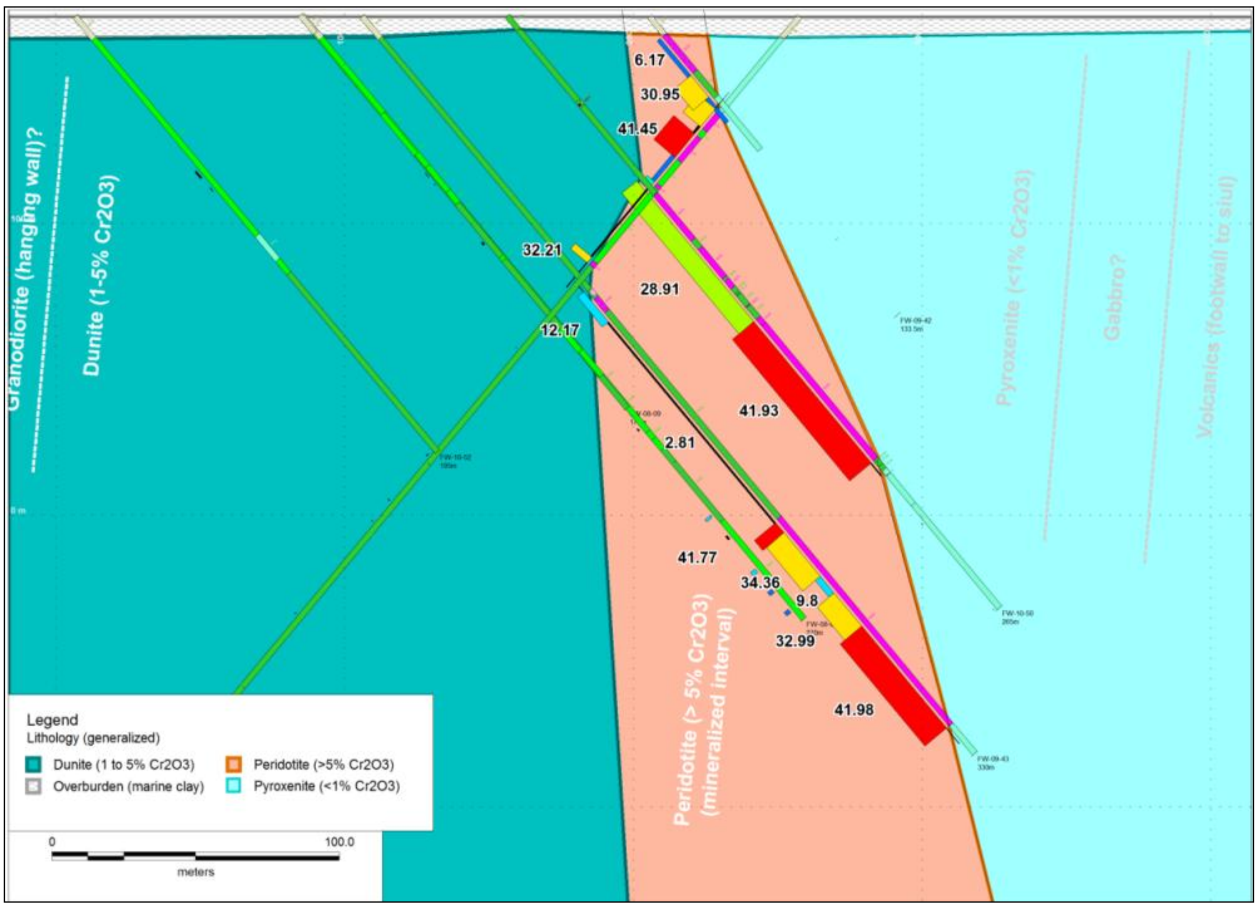
10.3 INTERPRETATION OF EXPLORATION INFORMATION

Although the geophysical techniques were initially aimed at identifying VMS and MMS targets, they were effective firstly in identifying the highly magnetic peridotitic phase of the McFaulds Lake Sill which contains the chromite mineralization, and secondly, in defining the potential chromite zone due to its high density characteristic. The strike lengths of the magnetic anomaly and gravity anomaly match the strike length of the chromite zone; furthermore, the intensity of the gravity anomaly is proportional to the thick massive chromite zone.

Drilling results indicate that the bulk of the Big Daddy deposit consists of massive chromite averaging 40% Cr_2O_3 with Cr/Fe ratio of approximately 2. The thickness of the deposit is variable but averages 17 m and 12 m for the southwest segment (BD 1) and northeast segment (BD 2), respectively. Both segments of the deposit remain open down dip and have yet to be closed off along strike.

The interpreted geology of the Big Daddy deposit is shown in Figure 7.3. A typical section of the deposit is shown in Figure 10-1.

Figure 10-1. Section 19+00 E (looking northeast) showing Pyroxenite cutting down into Massive Chromite Interval. Coloured bars below drill hole traces show chromite-bearing intervals with values below. Section is 400 m long.



11.0 DRILLING

The layout and extent of drill holes covering the Big Daddy deposit is shown in Figure 11-1. Details for each hole are given in Table 11-1.

11.1 2004, 2006 AND 2008 DRILLING CAMPAIGNS

The initial diamond drilling on the SKF claims was conducted in the winter of 2004. In that year, drill hole FW-04-01 was completed in claim block 3008793 (H Grid). Drill hole FW-06-02 (H Grid) and discovery hole FW-06-03 and hole FW-06-04 in claim P 3012253 (J Grid) were drilled in 2006.

The test drilling operations were suspended during 2007 and then revived in the winter of 2008. Between January and December, 2008, nineteen NQ drill holes (6,098 m) were completed on three targets on the Big Daddy claim (J Grid). The drilling completed during this phase defined chromite mineralization over a strike length of 400 m.

11.2 2009/2010 DRILLING CAMPAIGN

A total of 32 holes directed at the Big Daddy deposit (J Grid) were drilled during the 2009/2010 drill campaign. This drilling tested the chromite mineralization to a vertical depth of about 365 m and increased the known strike length of the mineralization from 400 m to about 1,200 m.

11.3 DRILLING PROTOCOLS

11.3.1 Spotting and Surveying of Drill Collars

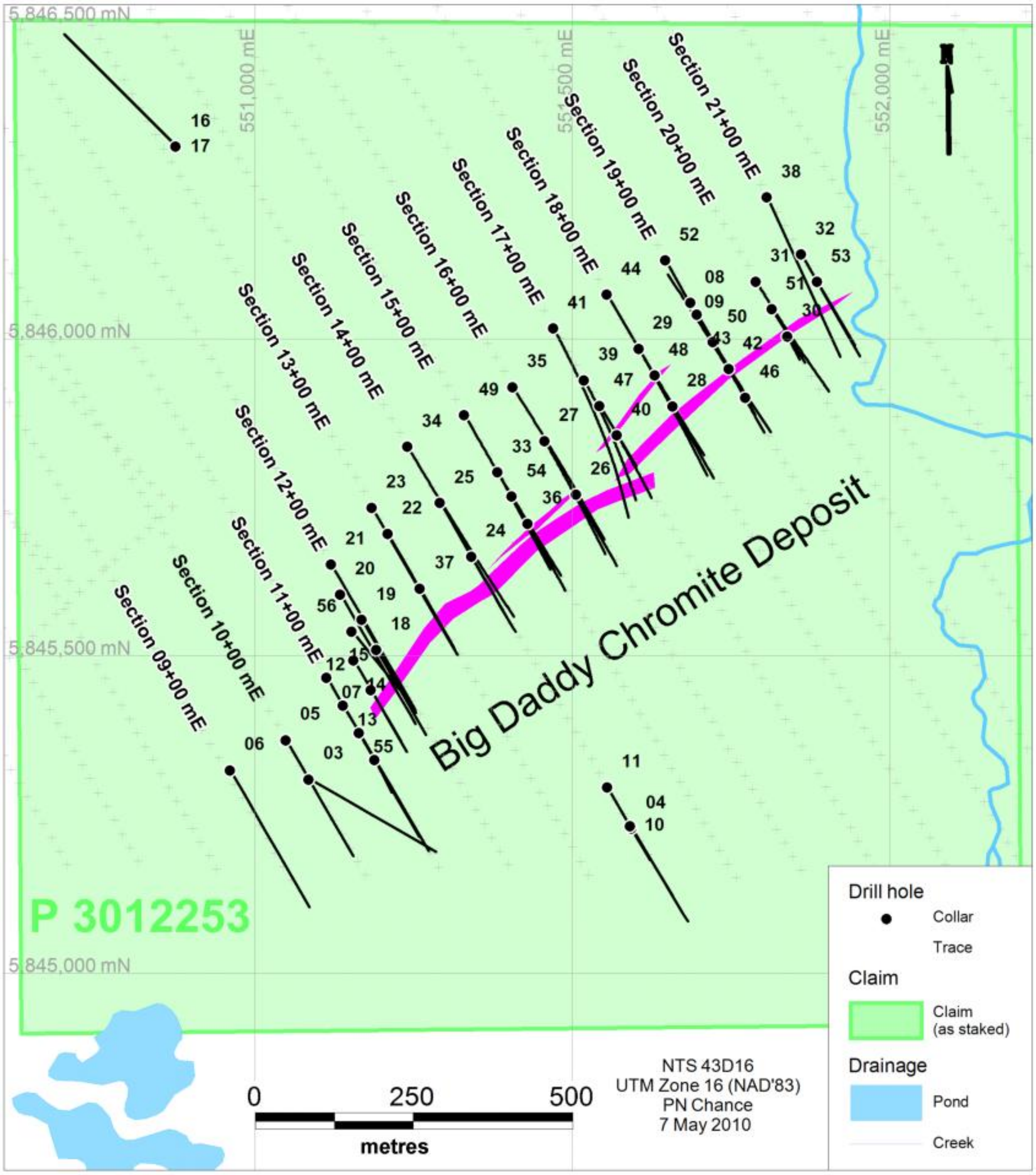
Collars were spotted relative to the 100 m cut lines. In early 2009, the J grid was re-cut and 25 metre-spaced pickets re-chained. Picket coordinates were located by GPS (Trimble GeoXH with post processing using an identical unit as a local base station (positional error is ±0.1 m)) and Trimble ProXRT with Omnistar real time correction (error is ±0.4 m). All coordinates are reported as metres in UTM Zone 16, NAD'83 datum. Elevations are reported as distance above sea level.

Cut lines and many pre-2009 drill pads are also visible on a Quickbird satellite image (circa summer 2008). All data points coincide within approximately 1 m.

Drill hole collars were spotted relative to the cut, J grid and azimuths were taken to be those of the cut lines. Initial collar dips were set using a builders' inclinometer. Azimuths and dips are reported in degrees.

Upon completion of drilling, all collars were surveyed using a pole-mounted, Trimble ProXRT GPS receiver. Buried casings were located using a magnetic pin finder.

Figure 11-1. Plan Showing all Drill Holes Covering the Big Daddy Deposit



Note: The deposit is shown in purple colour

Table 11-1. List of Drill Holes Drilled on SKF Property (2004 – 2010 Programs). (UTM Zone 16, NAD'83)

DDH Id	UTM_E	UTM_N	Elevation	Length	Grid	Line	Station	Azimuth	Dip
FW-04-01	555535.0	5846609.0	170.0	190.5	H	37+00 mS	05+50 mE	130.0	-45
FW-06-02	555368.0	5845849.0	170.0	197.0	H	30+00 mE	09+00 mS	130.0	-50
FW-06-03	551084.5	5845307.3	174.0	353.5	J	10+00 mE	15+28 mN	150.6	-50
FW-06-04	551592.9	5845230.2	170.8	254.0	J	14+00 mE	12+01 mN	329.1	-50
FW-08-05	551048.2	5845369.3	174.7	327.0	J	10+00 mE	16+01 mN	151.1	-50
FW-08-06	550960.2	5845321.7	173.8	384.0	J	09+06 mE	16+00 mN	155.9	-50
FW-08-07	551138.6	5845423.6	172.9	405.7	J	11+00 mE	16+00 mN	149.6	-50
FW-08-08	551685.8	5846058.5	171.3	270.0	J	19+01 mE	18+72 mN	150.7	-50
FW-08-09	551685.4	5846059.1	171.6	176.0	J	19+01 mE	18+73 mN	150.7	-65
FW-08-10	551590.7	5845233.9	170.9	312.0	J	14+00 mE	12+06 mN	149.9	-50
FW-08-11	551554.3	5845294.9	170.7	309.0	J	14+00 mE	12+76 mN	149.2	-50
FW-08-12	551112.6	5845468.1	173.1	354.0	J	11+00 mE	16+51 mN	149.9	-50
FW-08-13	551163.3	5845380.4	172.8	297.0	J	10+99 mE	15+50 mN	150.5	-50
FW-08-14	551181.9	5845448.6	173.6	189.0	J	11+50 mE	16+00 mN	150.3	-50
FW-08-15	551154.8	5845495.5	172.1	240.0	J	11+50 mE	16+50 mN	147.7	-50
FW-08-16	550875.0	5846305.0	174.0	372.0	J	13+19 mE	24+97 mN	315.0	-50
FW-08-17	550875.0	5846305.0	174.0	376.0	J	13+19 mE	24+97 mN	315.0	-65
FW-08-18	551190.7	5845511.9	171.4	255.0	J	11+93 mE	16+50 mN	155.0	-50
FW-08-19	551167.7	5845558.9	171.8	273.0	J	11+97 mE	17+02 mN	145.4	-50
FW-08-20	551134.0	5845599.0	174.0	375.0	J	11+88 mE	17+55 mN	150.0	-50
FW-08-21	551119.3	5845646.5	172.3	447.0	J	12+00 mE	18+04 mN	150.6	-50
FW-08-22	551208.6	5845694.5	172.2	330.0	J	13+00 mE	18+02 mN	149.8	-50
FW-08-23	551183.6	5845735.8	172.4	424.0	J	12+99 mE	18+50 mN	145.8	-50
FW-09-24	551340.5	5845658.8	171.6	219.0	J	14+00 mE	17+00 mN	150.1	-50
FW-09-25	551290.9	5845743.4	172.0	339.0	J	14+00 mE	18+09 mN	148.6	-50
FW-09-26	551505.6	5845756.7	171.4	207.0	J	16+00 mE	16+98 mN	150.7	-50
FW-09-27	551455.7	5845840.7	171.5	321.0	J	16+00 mE	18+00 mN	149.7	-50
FW-09-28	551657.6	5845895.6	171.0	207.0	J	18+00 mE	17+45 mN	150.7	-50
FW-09-29	551603.9	5845986.4	171.3	368.0	J	18+00 mE	18+49 mN	149.4	-50
FW-09-30	551838.4	5846005.7	170.4	65.0	J	20+00 mE	17+51 mN	150.2	-50
FW-09-31	551788.6	5846092.2	171.1	339.0	J	20+01 mE	18+52 mN	148.4	-50
FW-09-32	551859.5	5846135.2	169.8	291.5	J	21+00 mE	18+50 mN	150.1	-50
FW-09-33	551381.9	5845791.6	172.7	267.0	J	15+00 mE	17+95 mN	149.6	-50
FW-09-34	551239.8	5845831.8	171.8	468.0	J	14+00 mE	19+02 mN	150.3	-50
FW-09-35	551405.3	5845925.7	171.7	429.0	J	16+00 mE	18+98 mN	150.2	-50
FW-09-36	551429.4	5845710.4	172.6	192.0	J	15+00 mE	17+01 mN	150.2	-50
FW-09-37	551259.4	5845608.0	172.2	175.0	J	13+00 mE	17+01 mN	150.4	-50
FW-09-38	551805.9	5846225.5	171.2	423.0	J	20+99 mE	19+55 mN	150.7	-50
FW-09-39	551517.7	5845936.5	171.9	328.0	J	17+01 mE	18+50 mN	149.5	-50
FW-09-40	551569.6	5845849.9	171.2	175.5	J	17+00 mE	17+49 mN	150.0	-50
FW-09-41	551469.6	5846018.6	171.8	490.5	J	17+01 mE	19+46 mN	150.9	-50
FW-09-42	551745.9	5845954.4	171.0	133.5	J	19+00 mE	17+51 mN	150.5	-50
FW-09-43	551696.0	5846040.5	171.1	330.0	J	19+01 mE	18+51 mN	150.7	-50
FW-09-44	551553.8	5846071.9	171.4	423.0	J	18+00 mE	19+49 mN	149.7	-50
FW-09-45	552792.0	5846549.0	174.0	228.0	J	31+00 mE	17+32 mN	135.0	-50
FW-09-46	551771.9	5845909.5	171.1	351.0	J	19+00 mE	16+99 mN	329.1	-50
FW-10-47	551542.3	5845896.2	171.2	177.0	J	17+01 mE	17+98 mN	149.1	-50

DDH Id	UTM_E	UTM_N	Elevation	Length	Grid	Line	Station	Azimuth	Dip
FW-10-48	551629.1	5845944.3	171.4	228.0	J	19+00 mE	18+02 mN	150.2	-50
FW-10-49	551329.4	5845881.5	172.1	456.0	J	15+00 mE	18+99 mN	150.2	-50
FW-10-50	551721.2	5845997.0	171.2	265.0	J	19+00 mE	18+00 mN	150.6	-50
FW-10-51	551813.5	5846048.9	170.3	156.0	J	20+00 mE	18+02 mN	148.8	-50
FW-10-52	551645.8	5846126.2	171.1	195.0	J	19+00 mE	19+50 mN	150.8	-50
FW-10-53	551884.7	5846091.9	169.5	182.0	J	20+99 mE	18+01 mN	149.3	-50
FW-10-54	551404.3	5845753.3	172.4	210.0	J	15+00 mE	17+51 mN	150.4	-50
FW-10-55	551188.2	5845338.3	173.7	95.0	J	11+00 mE	15+00 mN	153.1	-50
FW-10-56	551151.9	5845540.2	173.3	241.0	J	11+70 mE	16+94 mN	140.7	-50

11.3.2 In-hole Directional Surveys

In-hole deviations were determined using one of three instruments; Flexit, Deviflex and north-seeking gyro. The Flexit employs a pendulum for inclinations and a magnetic compass to measure azimuth. Magnetic azimuth data are not usable due to the prevalence of magnetite in the sill. The Deviflex employs a pendulum for inclination and deformation of a flexible tube to estimate deflection. The instrument is deployed inside the drill string and is run through the entire hole to correctly estimate deviation. A north-seeking gyro was used to determine the down-hole deviation parameters of 12 holes. Once set, the gyro provides both the dip and azimuth for each station down the hole. Plans to complete both Deviflex and north-seeking gyro readings on a number of holes to assess the quality of the methods were not possible due to scheduling and equipment issues.

A review of results suggests that maximum deviations are less than 6 m per 100 m in both azimuth and dip.

11.4 SUMMARY AND INTERPRETATION OF DRILLING COMPLETED

Forty-two drill holes intersected chromite mineralization. All but three intersections (FW-06-03, FW-09-46 and FW10-56) were in holes collared at a 150° azimuth and -50° dip, thus footwall and hanging wall pierce points are evenly distributed providing good control on the mineralized envelopes.

Core recoveries were excellent particularly for the mineralized intercepts. Table 11-2 provides the composite assay results obtained from drill intersections >25% Cr_2O_3 on the Big Daddy deposit. An interpretation of the geometry of the deposit in plan view is given in Figure 11-1.

The deposit consists of two segments, BD 1 and BD 2 (Figure 17-2) and each segment comprises principal and subsidiary massive chromite bodies. The major massive chromite trends between 050 degrees and 060 degrees following the trend of the gravity anomaly. Based on the current drilling, the main mass of the Big Daddy deposit covers a strike length of 1 km and averages 17 m and 12 m in true thickness for BD 1 and BD 2, respectively. The mineralization has been tested to a vertical depth of about 365 m and remains open down dip and along strike.

Table 11-2. Big Daddy Drill Intercept Summary (>25% Cr_2O_3).

Hole #	Section	Station	Azimuth	Dip	Length	Intercepts								
						From (m)	To (m)	Length (m)	Pd ppb	Pt ppb	Cr_2O_3 %	Fe %	Fe_2O_3 %	Cr:Fe
FW-08-05	10+00 E	1600 N	150º	50º SE	327	251.20	264.00	12.80	101	86	**25.18**		16.68	1.48
						264.00	270.00	6.00	49	41	**34.03**		18.69	1.78
						291.40	298.85	7.45	31	90	**37.00**		22.68	1.60
FW-08-07	11+00 E	1600 N	150º	50º SE	405.7	194.35	205.90	11.55	440	321	**28.63**	14.74		1.33
						209.80	223.20	13.40	88	186	**33.92**	18.67		1.24
FW-08-12	11+00 E	1650 N	150º	50º SE	354	228.25	240.00	11.75	407	177	**34.36**		21.99	1.53
						252.25	260.70	8.45	272	199	**33.23**		25.55	1.27
FW-08-13	11+00 E	1550 N	150º	50º SE	297	74.30	102.00	27.70	138	186	**33.06**		17.29	1.87
						116.35	142.15	25.80	283	205	**34.76**		24.34	1.40
FW-08-14	11+50 E	1600 N	150º	50º SE	189	36.25	81.00	44.75	166	189	**39.30**		20.27	1.90
						81.00	103.50	22.50	201	154	**26.64**		18.54	1.41
FW-08-15	11+50 E	1650 N	150º	50º SE	240	160.15	171.30	11.15	171	146	**34.41**		24.14	1.39
FW-08-18	12+00 E	1650 N	150º	50º SE	255	44.90	46.50	1.60	291	177	**31.77**		25.08	1.24
						104.70	136.60	31.90	67	88	**37.60**	15.61		1.65
FW-08-19	12+00 E	1700 N	150º	50º SE	273	141.50	144.10	2.60	222	199	**31.32**	13.79		1.55
						160.85	161.95	1.10	54	59	**32.16**	20.00		1.10
						183.00	229.50	46.50	189	212	**37.18**	15.30		1.66
FW-08-20	12+00 E	1750 N	150º	50º SE	357	260.10	263.70	3.60	173	153	**31.60**	14.30		1.51
						304.30	336.95	32.65	168	218	**39.56**	14.37		1.88
FW-08-21	12+00 E	1800 N	150º	50º SE	447	376.00	385.80	9.80	67	122	**37.33**		23.23	1.57
						405.00	417.00	12.00	105	144	**35.46**		21.99	1.58
FW-08-22	13+00 E	1800 N	150º	50º SE	330	256.05	262.65	7.60	247	260	**28.55**	10.34		1.89
						263.65	298.50	34.85	170	194	**42.08**	15.92		1.81
FW-08-23	13+00 E	1850 N	150º	50º SE	424	332.30	337.50	5.20	526	297	**37.36**	15.04		1.70
						337.30	351.50	14.00	133	157	**24.54**	11.41		1.47
						351.50	378.00	26.50	98	178	**38.78**	14.92		1.78
FW-09-24	14+00 E	1700 N	150º	50º SE	219	73.50	80.30	6.80	264	229	**41.01**		21.10	1.90
						100.87	132.20	31.33	167	230	**40.63**		23.40	1.70

Hole #	Section	Station	Azimuth	Dip	Length	Intercepts								
FW-09-25	14+00 E	1800 N	150°	50° SE	339.5	232.10	270.35	38.25	167	231	**41.63**		21.04	1.94
FW-09-27	16+00 E	1800 N	150°	50° SE	321	173.30	186.80	13.50	282	245	**36.32**		20.77	1.71
						208.00	246.80	38.80	204	216	**42.99**		20.99	2.00
FW-09-28	18+00 E	1750 N	150°	50° SE	207	38.70	61.10	22.40	117	200	**41.30**		22.16	1.82
FW-09-29	18+00 E	1850 N	150°	50° SE	368	117.00	136.00	19.00	496	231	**40.02**		19.92	1.97
						226.00	230.70	4.70	456	267	**37.90**		20.39	1.82
						234.75	244.30	9.55	319	386	**38.33**		19.70	1.90
						248.60	323.75	75.15	234	248	**43.40**		21.26	2.00
FW-09-30	20+00 E	1750 N	150°	50° SE	77	24.10	32.75	8.65	263	257	**40.92**		22.61	1.77
FW-09-31	20+00 E	1850 N	150°	50° SE	339	207.00	214.50	7.50	184	218	**41.61**		20.49	1.99
						220.50	225.00	4.50	253	390	**36.46**		19.36	1.84
						235.90	264.50	28.60	179	215	**40.26**		19.80	1.99
FW-09-32	21+00 E	1850 N	150°	50° SE	291.5	180.90	186.00	5.10	301	238	**40.78**		22.57	1.77
						188.00	196.15	8.15	270	280	**38.50**		21.45	1.76
						200.60	206.60	6.00	215	190	**37.55**		21.79	1.69
FW-09-33	15+00 E	1800 N	150°	50° SE	267	195.00	203.70	8.70	289	185	**29.92**		22.25	1.32
						203.70	205.60	1.90	198	194	**34.89**		25.86	1.32
						207.60	210.00	2.40	145	197	**29.25**		23.45	1.22
						210.00	221.00	11.00	115	195	**40.29**		25.63	1.54
FW-09-34	14+00 E	1900 N	150°	50° SE	468	343.50	363.00	19.50	235	228	**33.17**		18.38	1.76
						383.50	415.22	32.72	247	252	**41.25**		20.93	1.93
FW-09-35	16+00 E	1900 N	150°	50° SE	429	349.16	355.50	6.34	259	345	**36.95**		28.80	1.25
						364.50	399.00	34.50	318	270	**41.15**		21.25	1.89
FW-09-36	15+00 E	1800 N	150°	50° SE	192	9.80	21.00	11.20	122	179	**40.14**		21.12	1.86
						24.90	38.00	13.10	139	235	**31.22**		20.39	1.50
						47.65	96.00	48.35	162	231	**41.35**		22.03	1.84
FW-09-37	13+00 E	1700 N	150°	50° SE	171	100.00	114.40	14.40	168	200	**41.07**		22.35	1.80
FW-09-38	21+00 E	1950 N	150°	50° SE	423	263.00	266.00	3.00	622	269	**34.10**		22.92	1.46
						390.50	398.00	7.50	240	201	**39.38**		24.58	1.57
FW-09-39	17+00 E	1850 N	150°	50° SE	328	119.10	124.50	5.40	406	206	**36.96**		21.84	1.66
						124.50	138.00	13.50	237	89	**33.44**		21.30	1.54
FW-09-40	17+00 E	1750 N	150°	50° SE	175	79.50	83.60	4.10	240	291	**34.62**		22.76	1.49

Hole #	Section	Station	Azimuth	Dip	Length	Intercepts								
						87.40	102.20	14.80	150	225	**43.11**		20.95	2.01
FW-09-41	17+00 E	1950 N	150º	50º SE	490.5	234.00	235.50	1.50	311	173	**36.41**		28.53	1.25
						262.50	265.50	3.00	339	208	**33.39**		25.52	1.28
						319.50	320.60	1.10	782	257	**31.38**		23.23	1.32
FW-09-42	19+00 E	1750 N	150º	50º SE	133.5	25.50	31.50	6.00	115	210	**36.01**		20.32	1.73
						32.70	35.90	3.20	53	211	**39.45**		20.69	1.87
FW-09-43	19+00 E	1850 N	150º	50º SE	330	225.00	249.00	24.00	190	260	**35.75**		18.99	1.84
						260.00	317.00	57.00	216	241	**40.52**		20.73	1.91
FW-09-44	18+00 E	1950 N	150º	50º SE	423	281.35	314.00	32.65	508	252	**36.33**		19.11	1.86
FW-09-46	19+00 E	1700 N	330º	50º NW	350	43.00	51.10	8.10	117	211	**34.28**		20.39	1.64
						54.10	64.70	10.60	162	203	**41.45**		19.70	2.06
						109.50	112.00	2.50	525	220	**32.21**		19.77	1.59
FW-09-47	17+00 E	1800 N	150º	50º SE	177	66.00	76.20	10.20	297	135	**34.06**		22.34	**1.49**
FW-09-48	18+00E	1800 N	150º	50º SE	228	8.90	10.75	1.85	521	276	**40.44**		21.13	**1.87**
						13.65	28.50	14.85	299	149	**39.52**		21.41	**1.81**
						126.20	132.00	5.80	218	205	**37.79**		20.84	**1.77**
						136.93	144.10	7.17	186	330	**36.29**		20.61	**1.72**
						148.00	180.40	32.40	137	233	**42.51**		21.52	**1.93**
FW-10-49	19+00 E	1900 N	150º	50º SE	456	337.40	338.65	1.25	748	370	**41.47**		23.45	1.73
						346.30	403.30	57.00	237	259	**40.52**		20.58	1.93
FW-10-50	16+00 E	1800 N	150º	50º SE	256	79.50	100.00	20.50	241	243	**38.00**		19.19	1.94
						103.75	124.65	20.90	285	329	**38.13**		19.53	1.91
						135.00	198.30	63.30	211	237	**41.93**		20.97	1.96
FW-10-51	20+00 E	1800 N	150º	50º SE	156	111.55	116.00	4.45	169	226	**40.41**		20.82	1.90
						118.50	133.40	14.90	308	273	**41.02**		22.54	1.78
FW-10-53	21+00 E	1800 N	150º	50º SE	182	99.10	106.75	7.65	176	241	**40.52**		21.95	1.81
FW-10-54	19+00 E	1750 N	150º	50º SE	210	137.60	142.70	5.10	255	267	**26.70**		18.87	1.38
						155.00	181.80	26.80	124	212	**41.46**		22.31	1.82
FW-10-55	11+00 E	1500 N	150º	50º SE	95	10.70	44.00	33.30	239	209	**37.18**		21.21	1.72
FW-10-56	11+50 E	1750 N	150º	50º SE	240	146.67	147.62	0.95	84	191	**37.60**		24.60	1.50
						173.61	223.02	49.41	225	239	**37.86**		20.10	1.84

Notes: 1.Intercept lengths do not equal true widths. 2. Intercepts are as averaged by J. Burns of Spider. 3. Cr:Fe ratios are averages for the intercept for the elements.

12.0 SAMPLING METHOD AND APPROACH

The core logger marked out lithologic units including mineralized intervals. A Niton hand-held XRF was used to more precisely locate assay cut-off (<5% Cr_2O_3) and grade-range limits. Generally the entire mineralized interval plus a minimum of five intervals (~7.5 m) into sub-cut-off material were sampled. In a few places, wide (>20 m) sections containing <5% Cr_2O_3 were encountered within the broadly mineralized zone and were not sampled.

The geologist then marked out end points of sample intervals. All sample intervals were selected within geologically-defined intervals of uniform lithology (including alteration and structure) and then of consistent grade, finally selecting samples of ~1.5 m length. Lower grade "shoulders" on massive intervals and rare lower grade intervals within massive material were sampled separately to ensure that true grade-thickness profiles were captured. A few sample intervals were as short as 0.3 m.

In view of the wide intervals of consistently high grade material, geologists tended to synchronize sample start or finish positions with driller's blocks providing for great uniformity in the sampling process and allowing for consistency between geotechnical and chemical parameters. Once the sample intervals were selected, sample tags were inserted and sample descriptions recorded.

A technician then completed geotechnical observations including core photography, magnetic susceptibility, specific gravity (SG), recovery and RQD, after which samples were cut and packed. The sample cutters maintained a sample log which provided a means of verifying values entered by the logger.

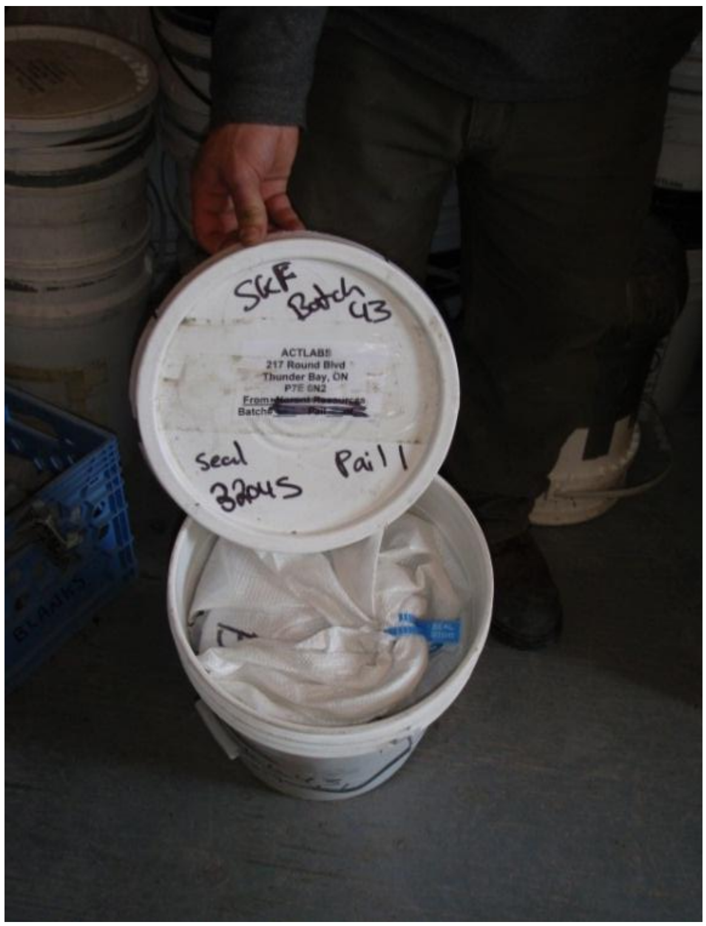
Core cutting was carried out using diamond-embedded blades in a separate tent. Cutters wore face masks, gloves and glasses while the saw mist was vented from the tent. Core cuttings were accumulated and backhauled to a licensed landfill.

Samples were cut by batch, so that each batch was checked, packed and sealed before the next was started.

Samples were placed in 20 L plastic pails, in rice bags, sorted by batch position. Each rice bag (one per pail) was sealed with a numbered locking tag (Figure 12-1). The lid was then secured with locking ties inserted through drilled holes to avoid separation in transit. Samples were shipped by batch (typically three pails).

Pails were transported to Nakina, stored in a secure warehouse and then shipped by bonded carrier to Actlabs in Thunder Bay. Upon receipt Actlabs issued work orders by which the batch was tracked to completion.

Figure 12-1. Sealed Rice Bags Being Placed into Pails.



The sampling process and data capture, evolved over the 2009/2010 drilling program, such that an already low error rate was reduced to near zero. In addition the grade of each interval as, visually estimated by the logger, may be validated against the specific gravity and checked on core photos.

12.1 CONCLUSIONS

Micon believes that the insertion of at least two standards in each sample batch and the monitoring of the analytical results by an independent consultant (i.e. Ms. Tracy Armstrong, P. Geo. – see Section 14) add confidence that the assays reported are reliable.

Given that down-hole surveys were conducted using appropriate methodology and equipment, and that core recoveries were good as described in Section 11, there are no factors known to Micon which might materially impact on the reliability of the results reported by Spider/KWG. The down-hole surveys and good core recoveries also ensured that samples are representative of the deposit.

A summary of the results of the composite samples is given in Table 11-2.

13.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

All on-site at McFaulds Lake sample handling and preparation were carried out by Billiken Management Services under the supervision of Qualified Persons (Lahti and Chance). At no time were employees, officers, directors or agents of Spider, KWG or Freewest involved in the sample selection, preparation and shipping process beyond exercising oversight to ensure that established protocols were being observed.

13.1 QUALITY CONTROL MEASURES BEFORE DISPATCH OF SAMPLES

13.1.1 Pre-2008 Drill holes and Samples

All pre-2008 drill holes and samples were purely of a reconnaissance nature designed to test geophysical anomalies for a variety of metals and no specific QA/QC measures were instituted for those samples.

13.1.2 2008 Analyses

During the 2008 drilling and sampling campaign, Howard Lahti, PhD, P. Geo. instituted an initial QA/QC program which involved inserting split duplicates and blanks in the sample stream.

13.1.3 2009-2010 Analyses

In March, 2009, Spider retained Tracy Armstrong, P. Geo., to institute a comprehensive QA/QC program which was achieved in two parts. First, samples were assigned to specific positions in batches of 35, leaving space for the laboratory to insert internal controls. Company control samples comprised two or three certified standards (Table 13-1), a project "blank", split, coarse reject and pulp duplicates. There were typically six QA/QC samples in each batch of 35.

Other than the insertion of QA/QC samples into sample batches, packing and dispatching the batches from McFaulds Lake, no other task was performed by employees of Billiken.

13.2 LABORATORY DETAILS

All Cr_2O_3 analyses completed in 2009 and 2010 were carried out by Activation Laboratories Ltd. (Actlabs) the principal office of which is in Ancaster, Ontario. Since February 27, 1998 Actlabs has been certified (accredited laboratory number 266) by the Standards Council of Canada as a mineral analysis laboratory with specific ability to analyze Cr_2O_3 by XRF fusion as follows:

"Fusion XRF using PHILIPS PW 2400 XRF Spectrometer (Quantify 15 analytes by X-ray Fluorescence which are fused with lithium and reported in the oxide form - SiO_2, Al_2O_3,

Table 13-1. Standards Used During 2009/2010 Drilling and Re-sampling Programs.

Standard	Cr_2O_3 (%)	Ni (%)	Pb (ppb)	Pt (ppb)	Au (ppb)	Source
OREAS 73A	1.69*	1.41	78	64	14	Ore Research, Australia
SARM 8	48.90					Mintek, South Africa
BD-1	21.60	0.124	182	177		CDN Resource Lab (custom)
BD-2	30.23	0.001	232	261	10.6	CDN Resource Lab (custom)
BD-3	40.75	0.097	234	197		CDN Resource Lab (custom)
PGMS 16			4,660	1,230	1120	CDN Resource Lab (custom)

* Cr (acid digestion)

Fe_2O_3, MnO, MgO, CaO, Na_2O, K_2O, T_iO_2, P_2O_5, Cr_2O_3, Co_3O_4, NiO, Zn, Sn and Cu)." (source: www.actlabs.com).

13.3 SAMPLE PREPARATION

In 2009 and 2010 sample preparation, ICP and fire assays were completed at Actlabs Thunder Bay facility. The material pulps were shipped by bonded courier to ActLabs, Ancaster laboratory for XRF analysis.

The following summary on sample preparation was provided by Actlabs, Thunder Bay: The entire sample is crushed to a nominal minus 10 mesh (1.7 mm), mechanically split (riffle) to obtain a representative sample (about 500 g) and then pulverized to at least 95 % minus 150 mesh (105 microns). (source: www.actlabs.com).

13.4 ANALYSES

Table 13-2 summarizes the sources of Cr/Cr_2O_3 data in the database. However, only the INAA and XRF results are used in the resource estimate. The large number of ICP analyses reflect the effort made to find potential PGE-enriched intervals for which geochemical evidence suggests reasonable potential. The evolution of analytical methods used reflects the growth of the project from Cu-Zn, through Ni-Cu-PGE to chrome.

13.4.1 2006/2008 Analyses

Sample pulps were shipped to Ancaster where all were analyzed by ICP using a four acid digestion (Actlabs Method 1F2, Total Digestion – ICP; Table 13-3). Designated and ICP over-limit samples were analyzed for nickel and copper by Optical Emission Spectrometry (ICP-OES). Precious metals (Au, Pd and Pt) were determined by ICP analysis of a fire assay bead. Samples reporting >1% Cr by ICP were re-analyzed by Instrumental Neutron Activation Analysis (INAA).

13.4.2 2009/2010 Cr_2O_3 Analyses

In early 2009, following a QA/QC review by Tracy Armstrong, XRF analysis of fused borate disks was adopted as the method of choice due to shorter turn-around times, greater laboratory capacity and delivery of the major element oxides and loss on ignition (LOI). A summary of the 2009/2010 analytical procedures is presented in Table 13-3.

13.4.3 INAA versus Fusion XRF

Prior to 2009 INAA was the analytical method of choice due to perceived problems with fusions and limitations of acid digestions. XRF analysis of borate glass disks was adopted as a result of limited reactor capacity (required to irradiate samples), slow turn around time due to the delay between irradiation and counting and the importance of other major element oxides in characterizing potentially marketable products. These changes reflected the suggestions of a chromite expert (S. McQuade, personal communication, 2009).

Table 13-2. Summary of Cr and Cr_2O_3 Analyses by Method.

Method	Count
ICP	5,662
INAA (only)	613
INAA + XRF	377
XRF (only)	2,359

Note: The ICP count includes 505 samples taken from holes that did not intersect the sill.

Table 13-3. Analytical Methods for 2009-2010 Drilling and Resampling Programs.

Code	Method	Description
RX1	Sample preparation	Crush (<5kg > up to 75% passing 2 mm (coarse reject), split (250 g) and pulverize (hardened steel) to 95% passing 105 μ (pulp).
1F2	Total Digestion – ICP	A 0.25 g sample is successively digested with hydrofluoric, nitric and perchloric and finally hydrochloric acids. **Chromite is partially solubilized.** Analysis by Varian Vista ICP.
1C	Exploration - Fire Assay -Au,Pd,Pt-ICP/MS	A 30 g (may be 5 to 50 gram) sample is fired to 1060 °C with fluxes (borax, soda ash, silica, litharge) and an Ag collector for an hour. The lead button is cupelled at 950°C to recover the Ag (doré bead), acid digested and the solution analyzed for Au, Pt, Pd by ICP/MS. **Smaller sample splits are used for high chromite or sulphide samples to ensure proper fluxing and metal recoveries**. LDL's & UDL's are 1 ppb & 30 g/t respectively.
4C	XRF Fusion – XRF	The sample is roasted 1050°C for 2 hours (from which LOI is determined), a glass is formed by fluxing a portion of the roasted material with lithium borate flux. The glass is analyzed on a Panalytical Axios Advanced wavelength dispersive XRF. **The limit of detection is about 0.01 wt% for most of the elements including Cr_2O_3.**

Source: http://www.actlabs.com/list.aspx. (30 March 2010)

Some cross-check analyses conducted under the supervision of independent consultant Tracy Armstrong, P. Geo., showed that the INNA and Fusion-XRF methods yielded the same result for Cr_2O_3. However, other than the problems associated with INAA already mentioned above, the latter was more preferable as it gave a quicker turn around.

13.4.4 Laboratory In-house QA/QC

The ActLabs in-house analytical QA/QC procedures include the following:

- Use of certified reference materials.
- Routine duplicate analyses.
- Use of blanks.
- Participation in round robin analytical exercises.

13.5 SECURITY

A chain of custody was maintained on dispatching the samples to the laboratory. Samples were shipped in complete batches (typically three pails) by backhaul flights to Nakina where they were stored in Nakina Air Services' secure warehouse before being shipped by bonded carrier to Actlabs facility in Thunder Bay.

Upon receipt of the samples in Thunder Bay, ActLabs personnel verified that seals were intact, checked the samples against the included packing slip and entered the batch into LIMS and forwarded a batch receipt, including the batch work order, to the sender, Ms. Armstrong, the client and Billiken's management and database manager. Any discrepancies were checked with the source prior to entry into LIMS. The laboratory's performance on control samples was monitored on a batch by batch basis by Tracy Armstrong, P.Geo. Ms. Armstrong "green-lighted" batches as received and compiled her analyses in reports issued approximately monthly and sent to Spider and copied to Billiken. An example of Ms. Armstrong's reports is in Appendix 2.

13.6 CONCLUSIONS

Micon is satisfied that the sample preparation, security and analytical procedures follow the current CIM exploration best practices guide lines. This ensures credibility of the analytical results used for the resource estimate.

14.0 DATA VERIFICATION

14.1 INTRODUCTION

The data verification conducted by Micon comprised four separate phases as follows:

- Laboratory visit.
- Site visit to the Big Daddy chrome project area at the close of the initial 2008 drilling phase.
- Site visit to the Big Daddy chrome project area during the latter half of the 2009/2010 drilling campaign.
- Resource database validation prior to conducting the resource estimate.

The first two items above were completed in conjunction with the previous 43-101 report (Gowans and Murahwi, 2009). The second two items support the current report and are described below.

14.2 SITE VISIT (OCTOBER, 2009)

14.2.1 Overview

Micon conducted a second site visit to the Big Daddy chromite project area on October 22, 2009 primarily to review QA/QC procedures, the construction of the resource database and at the same time to provide guidance in geotechnical logging of drill cores. In line with Micon's recommendations contained in the March 31, 2009 Technical Report, the SKF project personnel were found to have introduced stringent QA/QC measures under the guidance of QA/QC specialist Tracy Armstrong, P. Geo. These measures include the use of standards (certified reference materials) and blanks and monitoring of the performance of the standards and blanks on a real time basis. Ms. Armstrong also carried out a random selection of some pulps of the earlier (2008) analyses for repeat analyses.

14.2.2 SG Determinations

Another important component for the second site visit was verification of the tonnage factor.

Specific gravities were determined using a Totalcomp strain gauge attached to a control unit generally following ASTM standard D5779 – 08. The strain gauge (Figure 14-1) was attached to a bracket on a length of casing driven into the overburden and thus isolated from the core shack floor. A basket allowed pieces of core to be suspended in air and then in water.

Figure 14-1. Technician Working the Totalcomp Strain Gauge.

The operator selected intact pieces of core from each sample interval determined, numbering them in advance to aid correct replacement in the core tray. The apparatus was well damped such that the mass settled to ±0.001 kg in less than a couple of seconds. The masses in air and water were entered in a customized spreadsheet into which a correction for the buoyancy of the apparatus in water was inserted, the mass in air having been tared out. Initially the specific gravities of all mineralized intervals and adjacent wallrock were determined. Over the course of the project the frequency was reduced to every third mineralized and sixth un-mineralized interval with additional determinations across grade changes. A total of 2,216 observations were eventually made.

14.3 RESOURCE DATABASE VALDATION

The resource database validation conducted by Micon involved the following steps:

- Checking for any non-conforming assay information such as duplicate samples and missing sample numbers.
- Verifying collar elevations against survey information for each drill hole.
- Verifying collar coordinates against survey information for each drill hole.
- Verifying the dip and azimuth against survey information for each hole.
- Comparing the database assays and intervals against the original assay certificates and drill logs.

Some minor discrepancies were noted with duplication of sample intervals where duplicate analyses had been conducted. The necessary corrections were made.

14.4 CONCLUSIONS ON DATA VERIFICATION

Based on the foregoing data verification exercises, Micon is satisfied that the database used for the resource estimate in this Technical Report was generated in a credible manner and is representative of the main characteristics of the Big Daddy chromite deposit.

As described in its 2009 Technical Report, Micon had previously taken samples of core and of assay rejects which confirmed the presence of chromite at the grades reported for the Big Daddy deposit.

15.0 ADJACENT PROPERTIES

The following is a description of the properties adjacent to and within the environs of the Big Daddy deposit (Figure 15.1). The resources quoted below, with the exception of Black Thor, are taken from NI 43-101 compliant reports filed on SEDAR. The Black Thor estimate was reported in a press release (January 14, 2010) and the report, which states that it is NI 43-101 compliant, was obtained from Freewest's website in late January, 2010.

Micon has not independently verified the information contained in this section. Micon notes that the information is not necessarily indicative of the character and tenor of mineralization on the Big Daddy property.

15.1 CHROMITE

15.1.1 Black Thor / Black Label

The Black Thor and Black Label chromite deposits (owned by Freewest) are approximately 3 km northeast of the Big Daddy deposit. In early December, 2009 Freewest announced an initial resource estimate on its Black Thor and Black Label chromite properties (Table 15-1).

15.1.2 Black Creek

The Black Creek chromite deposit is adjacent to the Big Daddy deposit. During the second half of 2009, the Probe Mines/Noront Resources joint venture completed 20 holes along a 200 m long gravity anomaly situated in the southeast corner of claim P 4208219. Eleven holes (1,706 m) were drilled towards the northwest on five lines spaced 50 m apart and were completed to between 150 m and 175 m below surface (Probe, 2009; Noront, 2010).

The results of fifteen holes drilled from southeast to northwest (Table 15-2) describe a higher grade interval overlying a lower, less consistently mineralized footwall to the north. These data suggest that the deposit is comparable to the Big Daddy deposit which is the subject of this report.

15.1.3 Blackbird

Noront Resources' Blackbird 1 and 2 chromite deposits are located about 6 km to the southwest of the Big Daddy deposit. The resource estimate is based on 82 diamond drill holes (out of 154 drilled) completed on a 50 m grid. The database included 13,564 samples taken over 11,700 m of core. The area drilled extended along a 1,600 m portion of the sill over a 1,600 m width. Six mineralized zones have been outlined in an 1,100 m long by 800 m wide portion of drilled area, with estimated resources as shown in Table 15-3.

Table 15-1. Resource Estimate by the Sibley Basin Group Ltd. (A. Aubut, 2009).

Tonnes (millions)	Grade % Cr_2O_3	Cut-Off (% Cr_2O_3)
121.9	27.8	20
69.6	31.9	25
36.1	36.1	30
16.7	40.5	35

All resources are in the inferred category

Table 15-2. Black Creek intersections (Probe Mines Ltd, 2009).

Drill Hole	Section	From (m)	To (m)	Width (m)	Cr_2O_3 (%)	Cr:Fe
MJV09-18	0E	37.2	66.4	29.2	32.0	
including		37.2	54.3	17.1	41.6	
MJV09-19	0E	102.0	142.5	40.5	19.5	
including		102.0	116.5	14.5	30.0	
MJV09-20	0E	122.9	138.2	15.3	35.6	
including		124.0	131.5	7.5	40.0	
MJV09-10	50E	52.0	95.0	43.0	26.3	
including		52.0	67.0	15.0	36.1	
MJV09-03	50E	148.6	188.7	40.1	37.4	1.7
including		149.0	174.0	25.0	41.0	1.8
MJV09-04	50E	173.0	202.3	29.3	39.2	1.8
including		173.5	199.1	25.6	42.7	1.9
MJV09-12	100E	131.7	174.3	42.6	34.6	
including		131.7	153.4	21.7	43.1	
also including		166.8	174.3	7.5	41.2	
MJV09-13	100E	158.7	222.3	63.6	33.9	
including		158.7	193.4	34.7	41.4	
MJV09-14	100E	56.2	95.5	39.3	36.8	
including		56.2	80.4	24.2	42.8	
MJV09-11	150E	44.0	78.5	34.5	33.8	
including		44.0	65.0	21.0	37.4	
also including		44.0	59.0	15.0	43.7	
MJV09-05	150E	123.8	174.4	50.6	32.2	1.6
including		123.8	146.0	22.2	43.1	2.0
also including		164.4	171.4	7.0	40.3	1.9
MJV09-06	150E	160.0	224.4	62.4	34.5	1.6
including		160.0	194.0	34.0	41.4	1.8
also including		214.0	222.4	8.4	43.4	1.7
MJV09-17	200E	51.4	82.0	30.6	28.2	
including		51.4	63.5	12.1	40.5	
MJV09-15	200E	107.0	132.0	25.0	34.8	
including		107.0	119.4	12.4	43.7	
MJV09-16	200E	164.0	204.0	40.0	32.0	
including		164.0	173.0	9.0	42.4	

Table 15-3. Summary of Blackbird resource showing all categories (Micon, 2010).

Description	Category	Tonnes x 10^6	Avg. $\%Cr_2O_3$	Cr:Fe
BB2 Massive Chromite	Measured(M)	4.2	36.55	1.94
BB1 & BB2 Massive Chromite	Indicated (I)	3.4	36.08	1.94
BB1 & BB2 Massive Chromite	**Total M & I**	**7.6**	**36.34**	**1.94**
BB2 Massive Chromite	**Total Inferred**	**3.5**	**34.93**	**1.95**
BB2 Intercalated Chromite	Measured (M)	1.0	25.40	1.6
BB2 Intercalated Chromite	Indicated (I)	0.3	26.00	1.57
BB2 Intercalated Chromite	**Total (M & I)**	**1.3**	**25.54**	**1.6**
BB2 Intercalated Chromite	**Total Inferred**	**2.6**	**31.39**	**1.77**

15.2 FE-VA-TI (THUNDERBIRD)

In 2009 Noront Resources tested a prominent magnetic anomaly lying about 2 km northeast of the Freewest-Cliffs property (Figure 15-1). Three shallow holes reported about 0.5% vanadium (V_2O_5) in three ~30 m wide intersections over 900 metres of strike in ferrogabbro. The company suggests that the ferrogabbro is a more evolved portion of the McFaulds Sill.

15.3 MAGMATIC MASSIVE SULPHIDES (NI-CU-PGE) – EAGLE ONE

Noront Resources' (Golder Associates, 2010) current resource estimate describes mineralization as being 30 m thick, extending 125 m along strike and defined to 1,200 m below surface. Elsewhere, the company describes a series of lenses (1B, 1C, 1D, etc.,) (Noront, 2009) or informally, a "string of pearls". See Table 15-4.

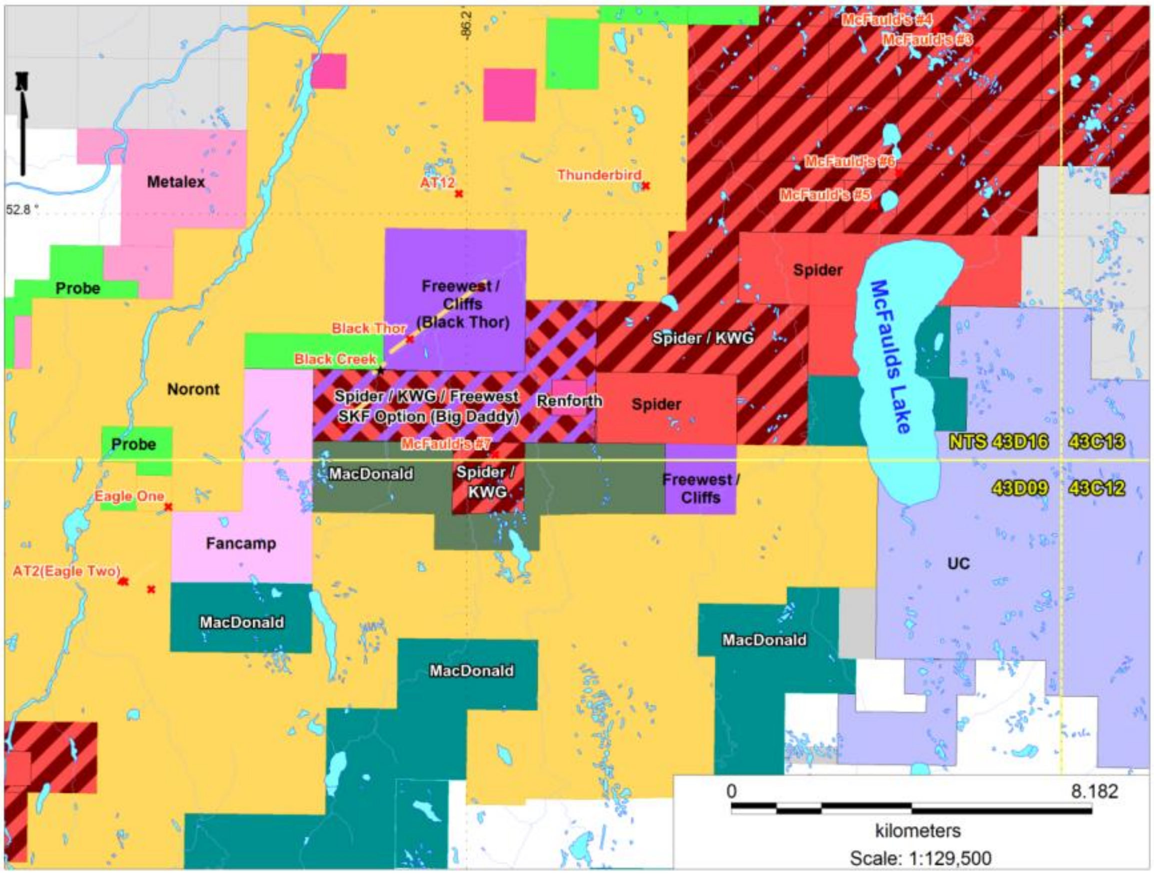
The deposit is reported to be contained in a narrow feeder dyke to the McFaulds sill. The discovery was made in 2007 when Noront gained access to the property and tested coincident airborne EM and magnetic anomalies thought to be similar to those at over the McFaulds VMS deposits.

15.4 VOLCANOGENIC MASSIVE SULPHIDES (CU-ZN) – MCFAULDS DEPOSITS

In 2002 De Beers Canada discovered sulphides in a reverse circulation hole testing an isolated magnetic anomaly immediately to the north of McFaulds Lake. Spider and KWG drilled the McFaulds #1 and #3 prospects in sufficient detail to estimate resources on each deposit (Lahti, 2008). (See Table 15-5).

Other than the De Beers Victor diamond mine located approximately 100 km to the east, there are no producing mines in the James Bay Lowlands.

Figure 15-1. Claim Map of the McFaulds Lake Area (as of 22 April 2010).



SKF property is multi-hatched area at the centre of the map

Table 15-4. Eagles Nest Resource Estimate (Golder Associates, 2010). Indicated and Inferred

Indicated							
CUTOFF (Ni %)	**TONNES**	**Ni %**	**Cu %**	**Pt gpt**	**Pd gpt**	**Au gpt**	**Ag gpt**
0.5	5,943,512	2.31	1.08	1.45	3.82	0.18	3.08
1	4,841,619	2.67	1.23	1.64	4.35	0.20	3.47
2	2,299,495	3.98	1.71	2.28	6.03	0.24	4.50
3	1,250,402	5.31	2.16	2.80	7.63	0.28	5.45
4	842,337	6.21	2.52	2.81	8.90	0.33	6.31
5	600,292	6.91	2.82	2.90	9.94	0.38	6.97
6	399,372	7.64	3.17	2.92	11.09	0.44	7.79
7	259,562	8.24	3.36	2.80	11.96	0.50	8.26
Inferred							
CUTOFF (Ni %)	**TONNES**	**Ni %**	**Cu %**	**Pt gpt**	**Pd gpt**	**Au gpt**	**Ag gpt**
0.5	4,050,123	1.50	0.91	0.83	3.60	0.25	3.54
1	2,650,781	1.88	1.11	0.90	4.21	0.28	4.24
2	685,490	3.28	1.25	0.71	5.39	0.21	4.80
3	280,372	4.60	1.17	0.56	6.33	0.14	4.32
4	164,931	5.40	1.19	0.52	7.14	0.12	4.43
5	91,834	6.12	1.22	0.47	7.93	0.10	4.62
6	44,672	6.81	1.21	0.45	8.81	0.05	4.90
7	15,870	7.52	1.15	0.42	9.22	0.05	4.69

Table 15-5. Summary of Resources on McFaulds 1 and 3 (reported by Lahti, 2008).

Deposit	Class	t	Cu (%)	Zn (%)	Cut off	DDH	Drilled (m)
McFaulds 3	Indicated	802,000	3.75	1.10	1.5% CuEquiv	39	12,114
McFaulds 1	Inferred	279,000	2.13	0.58.	1.5% CuEquiv	15	4,715

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Two phases of preliminary metallurgical testing have been completed using samples from the Big Daddy chromite deposit. The first phase comprised preliminary mineralogical, chemical and beneficiation testing by World Industrial Minerals, Arvada, Colorado, USA (WIM) in 2008.

The second phase consisted of a mineralogical and metallurgical test program undertaken by SGS Lakefield Research Limited, Lakefield, Ontario, Canada (SGS) in 2009. The metallurgical program completed by SGS was scoping in nature. It was designed to provide a preliminary indication of the metallurgical performance with regard to chromite recovery and upgrading potential of the Big Daddy mineralization.

16.1 METALLURGICAL SAMPLES

In July, 2008, quarter core samples taken from drill hole FW-08-05 were submitted to World Industrial Minerals (WIM) in Arvada, Colorado. Eight samples comprising two intervals (264.0 to 268.5 and 292 to 297 m) were tested.

Micon and Spider jointly selected the metallurgical samples in January, 2009 for the SGS test program. Eight composite metallurgical samples and twenty microprobe samples were prepared under the supervision of the Billiken's geological site team. Table 16-1 shows the sources of the metallurgical samples.

The eight metallurgical composite samples, comprising split quarter drill core, were crushed, blended, assayed and tested to investigate chromite recovery and upgrading potential.

A total of 20 samples were selected for Electron Microprobe Probe Analysis (EPMA) of chromite grains identified in thin sections prepared from drill core samples. Samples were selected from drill holes FW-08-05, FW-08-12, FW-08-13, FW-08-18 and FW-08-21.

16.2 MINERALOGICAL AND CHEMICAL ANALYSIS

16.2.1 WIM Preliminary Test Program

The eight samples were submitted to DCM Science Laboratory Inc. of Wheat Ridge Colorado (DCM) for x-ray diffraction (XRD) analysis and The Mineral Lab. Inc., of Lakewood, Colorado for x-ray fluorescence (XRF) analysis. DCM also completed a petrographic study of the samples.

A summary of the XRD analytical results is presented in Table 16-2.

Table 16-1. SGS-L Metallurgical Samples.

Sample ID	Drill Hole	No. of Intervals	Core Length (m)
Sample 2	FW-08-06	17	25.80
Sample 3	FW-08-23	17	25.50
Sample 4	FW-08-15	17	25.50
Sample 5	FW-08-18	16	24.00
Sample 6	FW-08-13	17	25.15
Sample 7	FW-08-22	16	24.35
Sample 8	FW-08-14	17	25.25
Sample 9	FW-08-12	16	20.80

Table 16-2. Summary of XRD Analysis Results.

Phase	17204	172405	172406	172426	172427	172428	172429	172430
Amphibole	-	-	8%	-	-	-	-	-
Chlorite	45%	45%	32%	37%	38%	36%	41%	34%
Pyroxene	5%	3%	-	-	-	-	-	-
Chromite	48%	51%	52%	61%	58%	60%	55%	50%
Talc	-	-	6%	-	2%	2%	1%	13%
Unaccounted	<5%	<5%	<5%	<5%	<5%	<5%	<5%	<5%

A summary of the XRF analytical results is presented in Table 16-3. Only elements and compounds with values above the instrument detection limit are included in the table.

As XRF analyses indicate that the chrome contents are between 18% and 23%, which corresponds to calculated chromite (Cr_2O_3) values of between 26% and 34%.

It is noted that the XRF analysis did not include PGM's, such as Pd, Pt and Rh.

The petrographic analysis showed that chromite grains were generally discrete and high grade. The grains typically had subhedral to euhedral shape and measured from 50 µm to 750 µm in size. The chromite grains tended to be of very high purity and no deleterious inclusions were identified.

The matrix containing the chromite grains is composed of altered chlorite and talc and the mineralogical investigations suggest that chromite could be liberated and recovered using standard mineral processing technology.

16.2.2 SGS-L Preliminary Testwork Program

Metallurgical Samples

Detailed analyses of the SGS-L metallurgical samples are included in Table 16-4.

Microprobe Analyses (EPMA)

A summary of the EPMA test results is presented in Table 16-5.

Microprobe work on 20 samples show that the Cr:Fe ratio of the chromite grains sampled ranges from 1.0 to 1.9. These ratios are lower than expected. The work also shows that the chromite grains are low in SiO_2 (<0.1%), contain about 14% Al_2O_3 and that there is a negative correlation between MgO and Fe. This is expected considering that the spinel structure of chromite generally has a positive correlation between Cr:Fe ratio and MgO content. This work also suggests a higher Cr:Fe ratio for the chromite grains for higher grade chromite samples.

Figure 16-1 compares the Cr:Fe ratio to the Al_2O_3 and MgO analysis. Figure 16-2 plots the FeO and MgO analyses against Cr_2O_3 and shows that as the MgO content of the chromite tends to increase when the Cr:Fe ratio increases. This is probably due to the spinel nature of the chromite and the substitution of Fe with Mg.

Table 16-3. Summary of XRF Analysis Results.

Element /Compound	Units	17204	172405	172406	172426	172427	172428	172429	172430
MgO	%	28	27	24	24	24	23	24	24
Al_2O_3	%	7	9	8	12	11	11	12	10
SiO_2	%	25	22	23	16	18	16	18	23
CaO	%	2.1	1.2	1.5	<0.1	<0.1	<0.1	<0.1	<0.1
TiO_2	%	0.3	0.3	0.4	0.4	0.3	0.3	0.4	0.3
MnO	%	0.2	0.2	0.1	0.2	0.3	0.3	0.3	0.3
Fe_2O_3	%	12	14	14	17	16	18	17	16
V	ppm	635	690	744	785	791	864	804	842
Cr	ppm	180,000	190,000	200,000	230,000	220,000	230,000	210,000	190,000
Co	ppm	135	142	162	176	170	174	155	176
Ni	ppm	1,320	825	1,040	1,120	1,070	921	1,130	819
Zn	ppm	316	348	403	529	518	540	499	567

Table 16-4. SGS-L Metallurgical Sample Chemical Analyses.

Sample ID	Sample 2	Sample 3	Sample 4	Sample 5	Sample 6	Sample 7	Sample 8	Sample 9
Cr_2O_3 % [2]	3.99	7.85	10.1	20.2	35.5	43.3	40.2	34.1
Cr %	2.73	5.37	6.91	13.8	24.3	29.6	27.5	23.3
Fe %	8.46	10.0	9.79	12.4	17.9	15.2	14.2	17.0
Cr:Fe Ratio	0.32	0.54	0.71	1.12	1.36	1.95	1.94	1.37
SiO_2 %	35.6	30.6	30.5	22.9	11.8	8.29	10.1	12.4
Al_2O_3 %	2.42	2.87	4.58	7.46	12.5	13.3	13.5	11.8
Fe_2O_3 %	12.1	14.3	14	17.7	25.6	21.7	20.3	24.3
MgO %	28.6	32.8	30.2	23.4	12.3	13.8	13.8	14.3
CaO %	2.54	0.39	0.23	0.79	0.23	0.09	1.23	0.23
Na_2O %	0.03	0.03	0.05	0.07	0.07	0.07	0.08	0.085
K_2O %	< 0.01	< 0.01	0.03	0.08	0.11	0.05	0.2	0.010
TiO_2 %	0.11	0.1	0.17	0.33	0.53	0.42	0.45	0.40
P_2O_5 %	< 0.01	< 0.01	< 0.01	0.07	< 0.01	< 0.01	< 0.01	0.010
MnO %	0.16	0.07	0.11	0.16	0.4	0.21	0.26	0.31
Cr_2O_3 %	3.99	7.85	10.1	---	---	---	---	---
V_2O_5 %	0.03	0.03	0.05	0.11	0.18	0.17	0.16	0.14
LOI %	14	11.4	9.13	6.34	1.77	0.64	0.33	2.35
Sum %	99.6	100.4	99.1	---	---	---	---	---
Ni %	0.14	0.14	0.14	0.14	0.093	0.11	0.11	0.12
S %	0.15	0.06	0.08	0.22	0.05	0.04	0.03	0.075
Au g/t	0.07	0.02	< 0.02	0.07	0.03	0.03	0.03	0.07
Pt g/t	0.09	0.06	0.14	0.25	0.22	0.19	0.15	0.215
Pd g/t	0.16	0.08	0.23	0.26	0.32	0.14	0.1	0.41
Cr_2O_3 % [2]	---	---	---	20.2	35.5	43.3	40.2	34.05
Fe_3O_4 % [1]	2.2	5.4	2.9	0.6	0	0	0	0

[1] *Magnetic iron minerals using a Satmagan analyzer.*

[2] *SGS noted that chromite minerals are often difficult to digest when submitted for chemical analyses. For this test program, SGS used fusion for the digestion of the samples. Borate fusion was used for the whole rock assay suite (WRA), followed by x-ray fluorescence (XRF) analysis. For samples with greater than 15% Cr_2O_3 content the samples were submitted for a re-assay using a Na_2O_2 fusion, followed by analysis by atomic absorption (AA).*

Figure 16-1. EPMA Samples, Cr:Fe Ratio vs Al_2O_3 and MgO.

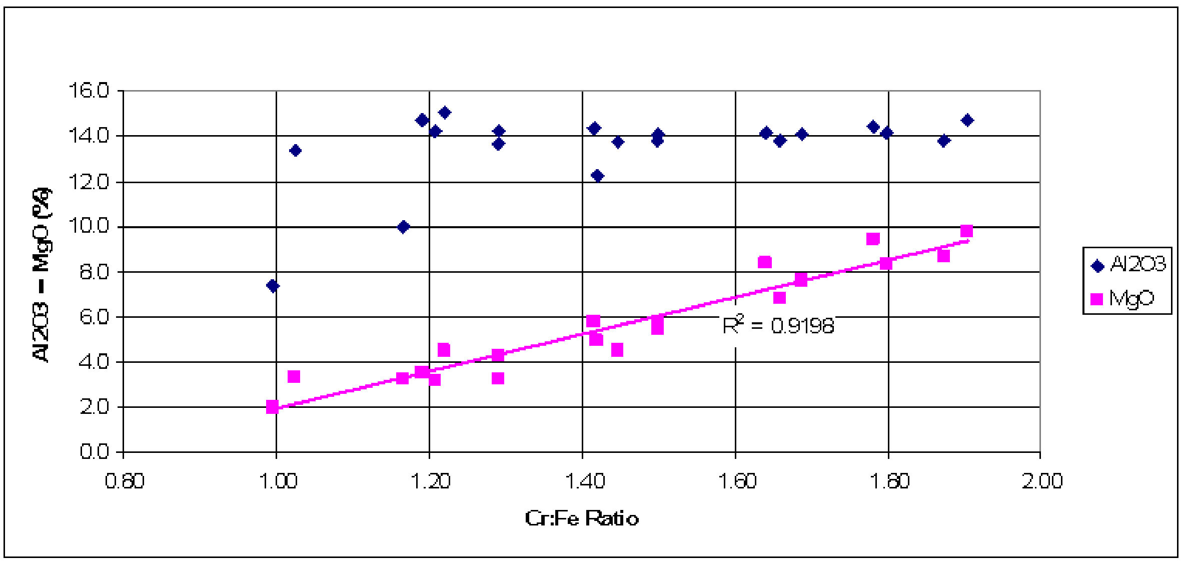
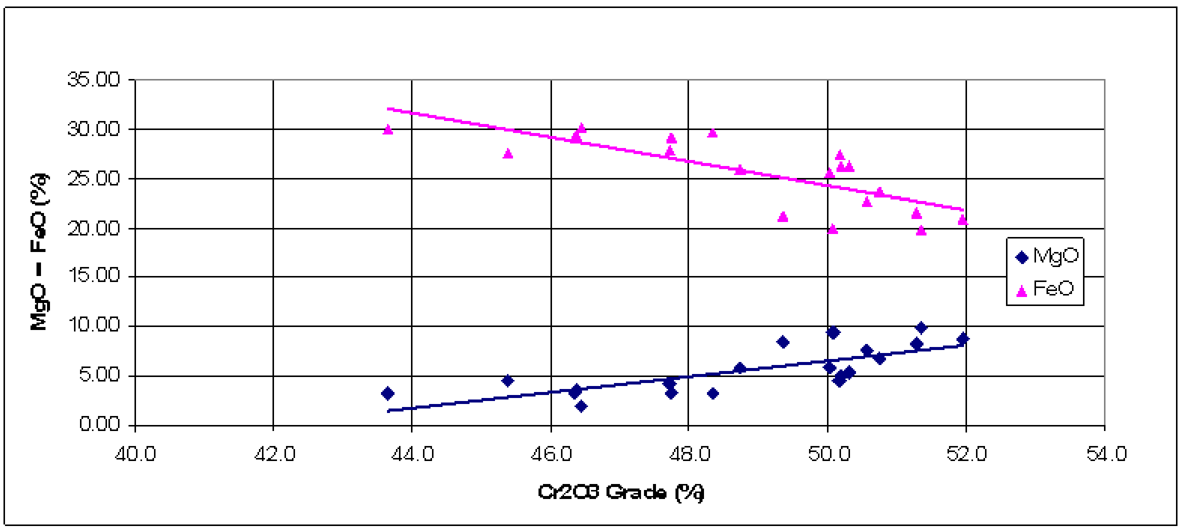


Table 16-5. SGS-L EPMA Results.

(All Units are Percent)

Sample	Cr_2O_3	Fe_2O_3	FeO	Cr:Fe ratio	SiO_2	TiO_2	Al_2O_3	MgO	CaO	MnO	NiO	Na_2O	Total
PS 5-1	51.9	3.95	20.8	1.87	0.12	0.37	13.8	8.69	0.005	0.20	0.061	0.011	100.0
PS 5-2	51.3	4.42	19.7	1.90	0.001	0.45	14.7	9.82	0.005	0.17	0.058	0.014	101
PS 5-3	50.2	3.41	27.4	1.45	0.037	0.62	13.7	4.52	0.009	0.53	0.033	0.023	101
Ave 5	**51.1**	**3.93**	**22.7**	**1.74**	**0.053**	**0.48**	**14.1**	**7.68**	**0.006**	**0.30**	**0.050**	**0.016**	**100**
PS 12-1	48.3	7.54	29.7	1.17	0.059	1.49	9.98	3.26	0.001	0.41	0.090	0.005	101
PS 12-2	50.6	4.12	22.6	1.69	0.15	0.42	14.1	7.63	0.002	0.22	0.092	0.017	100.0
PS 12-3	47.7	5.27	27.8	1.29	0.095	0.70	13.7	4.28	0.002	0.36	0.057	0.010	100.0
PS 12-4	47.7	3.83	29.1	1.29	0.051	0.54	14.3	3.26	0.002	0.38	0.008	0.023	99.2
Ave 12	**48.6**	**5.19**	**27.3**	**1.36**	**0.089**	**0.79**	**13.0**	**4.61**	**0.002**	**0.34**	**0.062**	**0.014**	**100**
PS 13-1	46.4	12.1	30.2	1.0	0.063	0.82	7.40	1.95	0.005	0.44	0.020	0.025	99.5
PS 13-2	51.3	3.91	21.6	1.80	0.034	0.45	14.1	8.33	0.000	0.35	0.037	0.015	100
PS 13-3	50.7	3.62	23.7	1.66	0.052	0.47	13.8	6.85	0.000	0.35	0.009	0.028	99.6
PS 13-4	46.4	5.04	29.2	1.21	0.055	0.61	14.3	3.20	0.002	0.37	0.010	0.021	99.2
Ave 13	**48.7**	**6.17**	**26.2**	**1.42**	**0.051**	**0.59**	**12.4**	**5.08**	**0.002**	**0.38**	**0.019**	**0.022**	**100**
PS 18-1	45.4	5.80	27.5	1.22	0.044	0.69	15.1	4.54	0.004	0.35	0.16	0.008	99.6
PS 18-2	50.3	3.68	26.2	1.50	0.061	0.40	14.1	5.43	0.000	0.21	0.042	0.010	100
PS 18-3	50.1	5.28	20.0	1.78	0.049	0.44	14.5	9.43	0.000	0.21	0.098	0.001	100
PS 18-4	49.4	5.83	21.3	1.64	0.046	0.44	14.2	8.42	0.003	0.47	0.17	0.003	100
Ave 18	**48.8**	**5.15**	**23.8**	**1.53**	**0.050**	**0.49**	**14.5**	**6.95**	**0.002**	**0.31**	**0.116**	**0.005**	**100**
PS 21-1	50.2	5.37	26.3	1.42	0.061	0.46	12.2	4.96	0.005	0.46	0.17	0.009	100
PS 21-2	43.6	8.38	30.0	1.02	0.042	1.26	13.4	3.28	0.003	0.27	0.070	0.000	100
PS 21-3	48.7	4.92	25.9	1.42	0.054	0.54	14.4	5.75	0.003	0.24	0.055	0.025	101
PS 21-4	50.0	4.23	25.6	1.50	0.038	0.47	13.8	5.77	0.004	0.27	0.067	0.021	100
PS 21-5	46.4	5.45	29.4	1.19	0.075	0.64	14.7	3.54	0.000	0.25	0.037	0.016	100
Ave 21	**47.8**	**47.79**	**47.8**	**47.79**	**47.792**	**47.79**	**47.8**	**47.79**	**47.792**	**47.79**	**47.792**	**47.792**	**48**

Figure 16-.2. EPMA Samples, Cr_2O_3 Grade vs MgO and FeO.



16.3 METALLURGICAL TESTING

16.3.1 WIM Preliminary Test Program

Metallurgical testing on the Big Daddy composite sample was performed by Phillips Enterprises LLC of Golden, Colorado. The scope of this preliminary testwork program included gravity separation and flotation of ground material. The work was scoping in nature and significant improvements in results would be expected from more detailed studies.

Table 16-6 provides a summary of the scoping testwork results. These results are based on chemical analyses, which are generally more accurate for chromite determination than the XRF method.

An XRF analysis of the combined concentrate is compared to the average feed analysis in Table 16-7.

Using the XRF analyses presented in Table 16.7, the calculated Cr to Fe ratio of both the average feed and combined concentrate is 1.83. However, using wet chemical methods to analyze for Cr_2O_3, which is more accurate than XRF due to potential incomplete dissolution of chromium using the XRF method, the value of Cr_2O_3 of 46.6% for the combined concentrate equates to a Cr to Fe ratio of 2.07.

Of note is the 11% SiO_2 assay of the combined concentrate which would preclude this product from some of the main chromite markets. However, mineralogical analyses suggest that the chromite grains are relatively pure, therefore additional liberation studies and metallurgical testing would most likely reduce this to an industry acceptable level.

16.3.2 SGS Preliminary Testwork Program

Metallurgical testwork on all eight composite samples comprised gravity separation tests and magnetic separation tests on fine gravity tailings. This work was designed to investigate the upgrading potential of the Big Daddy chromite samples.

In order to ascertain the pre-concentration potential, coarse separation tests (-½ inch) using heavy liquid separation (HLS) and magnetic separation were undertaken on two selected composites. Samples 6 and 9 were selected for these tests.

A scoping sulphide flotation test was undertaken to investigate sulphide-hosted base metals and PGM recoveries.

<u>Gravity and Magnetic Separation</u>

The gravity/magnetic separation test flowsheet developed by SGS is presented in Figure 16-3. This procedure had the following steps.

- The test sample is crushed to pass 20 mesh (850 μm).

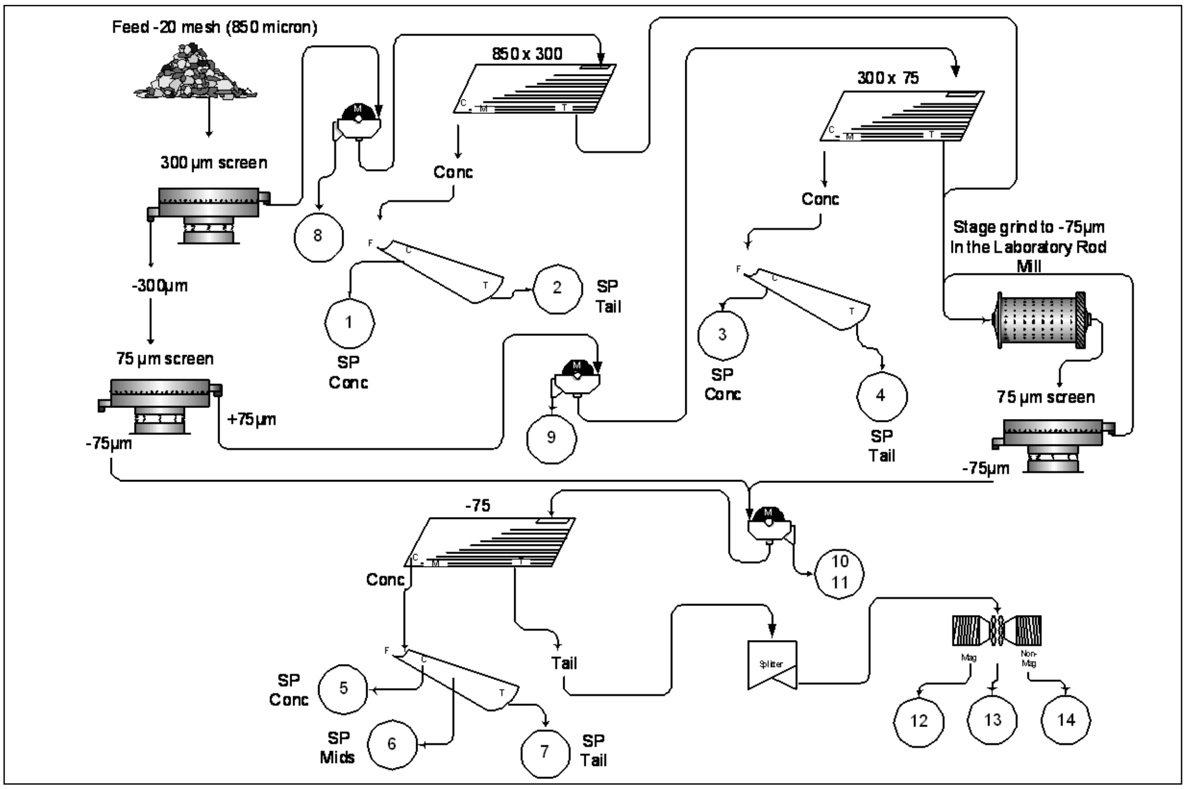
Table 16-6. Summary of Metallurgical Test Results.

Product	Chromite Grade (%)	Chromite Distribution (%)
Gravity concentrate	49	47
Flotation concentrate	43	28
Combined concentrate	47	74
Total Tailings	10	26
Feed	37	100

Table 16-7. Average Feed and Combined Concentrate XRF Analyses.

Stream	MgO (%)	Al_2O_3 (%)	SiO_2 (%)	CaO (%)	TiO_2 (%)	MnO (%)	Fe_2O_3 (%)	V (ppm)	Cr (ppm)	Co (ppm)	Ni (ppm)	Zn (ppm)
Feed	24	11	19	1.6	0.3	0.3	17	814	211,250	167	994	518
Conc.	18	10	11	0.2	0.5	0.2	22	954	280,000	221	761	652

Figure 16-3. SGS Gravity and Magnetic Separation Test Flowsheet.



- In order to enhance recovery as well as upgrading, the crushed sample is then split into three size fractions: 850 x 300 µm, 300 x 75 µm and -75 µm.

- The two coarsest sizes were passed over a Wilfley shaking table and the concentrates were processed on a Mozley mineral separator or a superpanner to further upgrade the heavy concentrate.

- To try and recover non-liberated chromite from the coarse gravity separation tailings, they were stage-ground to pass a 75 µm screen and combined with the original -75 µm fraction.

- Shaking table separation followed by the Mozley mineral separator or superpanner was used to produce a gravity concentrate from the -75 µm material.

- A sub-sample from the fine tailings was tested for chromite recovery by wet high-intensity magnetic separation [WHIMS].

- It is noted that prior to each gravity separation, the magnetic iron minerals were removed by low-intensity magnetic separation [LIMS].

The results from the gravity/magnetic separation tests are summarized in Table 16-8 and Table 16-9.

The results from these tests suggest the following:

- Samples with Cr_2O_3 values of 20% and over (samples 5 to 9) upgraded to potentially marketable chromite concentrates with reasonable recoveries. The two samples grading between 8.0% and 12.3% Cr_2O_3 upgraded to over 40% Cr_2O_3 but with low recoveries.

- There tends to be a positive recovery/feed grade relationship for samples 5 to 9. Also, the Cr:Fe ratios of the respective feed and concentrates were similar suggesting that the ratio cannot be improved with upgrading.

- It is noted that for the low grade samples (samples 2, 3 and 4) the LIMS recoveries were relatively high while for the higher grade samples (samples 5 to 9) the recoveries were low. This suggests magnetite locking, magnetite surface coatings or magnetic chromite grains due to high Fe content.

- Good chromite recoveries (>85%) were maintained for samples 6 to 9 while keeping the SiO_2 content in the concentrate below 5%. The SiO_2 content of sample 5 rose above 5% at just over 70% Cr_2O_3 recovery. The SiO_2 content of the low grade sample (2 to 4) concentrates was consistently high.

Table 16-8. Gravity/Magnetic Separation Test Results – 1.

Sample	Feed		+75 μ Grav Conc			-75 μ Grav Conc			Low-Intensity Magn.		High-Intensity Magnetics		
	Assay, %	Ratio	Cr_2O_3, %		Ratio	Cr_2O_3, %		Ratio	Cr_2O_3, %		Cr_2O_3, %		Ratio
	Cr_2O_3	Cr:Fe	Grade	Recovery	Cr:Fe	Grade	Recovery	Cr:Fe	Grade	Recovery	Grade	Recovery	Cr:Fe
2	4.42	0.35	37.0	5.93	0.83	34.8	4.34	0.77	14.0	56.2	7.05	16.8	0.32
3	7.96	0.56	42.5	1.57	1.28	41.4	1.79	1.09	11.2	92.2	1.96	2.06	0.29
4	12.3	0.76	41.2	11.4	1.26	42.7	7.55	1.19	16.2	65.7	7.20	8.21	0.67
5	20.4	1.17	44.8	22.4	1.47	46.8	25.5	1.49	14.9	7.20	20.5	13.5	1.17
6	35.4	1.35	44.3	57.2	1.37	47.3	19.5	1.37	23.0	0.32	40.8	8.87	1.37
7	42.9	1.88	49.0	51.6	1.89	50.3	4.10	1.89	32.5	0.53	47.6	32.0	1.90
8	40.0	1.96	47.3	52.9	2.02	51.2	16.9	2.10	28.3	0.63	46.4	23.2	1.88
9	34.8	1.43	46.3	33.2	1.43	47.5	10.7	1.39	28.2	0.78	42.0	15.0	1.37

Table 16-9. Gravity/Magnetic Separation Test Results – 2.

Sample	Product	Weight	Assays (%)			Distribution (%)	
		%	Cr_2O_3	S	SiO_2	Cr_2O_3	S
Sample 2	Gravity Conc +75 µm	0.71	37.0	2.96	2.93	5.93	1.92
	Gravity Conc -75 µm	0.55	34.8	0.29	1.44	4.34	0.15
	LI Magnetic Fraction	17.8	14.0	0.14	23.6	56.2	2.31
	HI Magnetic Conc	10.6	7.05	3.18	19.2	16.8	30.7
Sample 3	Gravity Conc +75 µm	0.29	42.5	0.62	2.73	1.57	3.24
	Gravity Conc -75 µm	0.34	41.4	0.36	2.12	1.79	2.19
	LI Magnetic Fraction	65.2	11.2	0.07	26.5	92.2	84.09
	HI Magnetic Conc	8.4	1.96	0.02	34.7	2.1	3.0
Sample 4	Gravity Conc +75 µm	3.41	41.2	0.070	4.28	11.4	3.78
	Gravity Conc -75 µm	2.17	42.7	0.11	2.17	7.55	3.79
	LI Magnetic Fraction	49.9	16.2	0.074	24.2	65.7	58.4
	HI Magnetic Conc	14.0	7.20	0.046	33.7	8.21	10.2
Sample 5	Gravity Conc +75 µm	10.2	44.8	0.051	3.01	22.4	2.90
	Gravity Conc -75 µm	11.2	46.8	0.16	1.63	25.5	9.93
	LI Magnetic Fraction	9.90	14.9	0.79	22.4	7.20	43.6
	HI Magnetic Conc	13.5	20.5	0.14	23.3	13.5	10.5
Sample 6	Gravity Conc +75 µm	45.7	44.3	0.032	3.64	57.2	64.3
	Gravity Conc -75 µm	14.6	47.3	0.022	1.83	19.5	14.11
	LI Magnetic Fraction	0.50	23.0	0.14	17.8	0.32	3.06
	HI Magnetic Conc	7.71	40.8	0.015	6.57	8.87	4.91
Sample 7	Gravity Conc +75 µm	45.2	49.0	0.000	2.77	51.6	0.0
	Gravity Conc -75 µm	3.50	50.3	0.095	0.84	4.10	16.8
	LI Magnetic Fraction	0.69	32.5	0.085	11.4	0.53	2.98
	HI Magnetic Conc	28.9	47.6	0.025	3.40	32.0	36.4
Sample 8	Gravity Conc +75 µm	44.8	47.3	0.010	2.49	52.9	19.4
	Gravity Conc -75 µm	13.2	51.2	0.032	0.80	16.9	18.1
	LI Magnetic Fraction	0.89	28.3	0.073	15.3	0.63	2.83
	HI Magnetic Conc	20.0	46.4	0.026	3.24	23.2	22.9
Sample 9	Gravity Conc +75 µm	24.9	46.3	0.010	3.17	33.2	3.68
	Gravity Conc -75 µm	7.80	47.5	0.11	1.81	10.7	12.5
	LI Magnetic Fraction	0.96	28.2	0.48	15.2	0.78	6.82
	HI Magnetic Conc	12.4	42.0	0.12	5.78	15.0	21.3

Pre-Concentration Tests

Pre-concentration at a relative coarse size, which is common in many commercial chromite beneficiation facilities, was undertaken to see if heavy media separation (HMS) or coarse particle magnetic separation would be feasible. Minus ½ inch portions of samples 6 and 9 were used. Heavy liquid separation (HLS) tests at ¼ inch, 0.85 mm and 0.3 mm resulted in very little upgrading which suggests a smaller than 0.3 mm liberation size for the chromite samples. The coarse magnetic separation results also showed negligible upgrading.

Sulphide Flotation

One sulphide flotation test was performed to determine if a sulphide concentrate with platinum group metals (PGM) minerals can be extracted from the chromite ore. A composite of equal fractions of samples 5, 6 and 9 was used in a 10-kg flotation test. Table 16-10 summarizes the flotation test results.

The flotation test was not optimized and improved results would be expected with a more detailed testwork program.

16.4 RECOMMENDATIONS

Most of the various chemical correlations discussed in the report are interesting but not unexpected. These data would benefit from mineralogical or geo-met investigations. QEMSCAM was included as an option by SGS but initially declined due to budget constraints. This, or similar technology, should be included in the next phase of work undertaken on samples that will be more representative of the potential total mineral resource.

The testwork conducted so far was undertaken using either massive or disseminated material. The coarse beneficiation tests were conducted on massive material. No samples crossed the contact between the 2 types, therefore magnetic and gravity tests to upgrade material were, in effect, inconclusive. It was suggested that future tests should include samples of massive chromite and low grade contact material to ascertain coarse beneficiation waste rejection.

A more detailed metallurgical and geo-metallurgical program of work is recommended using samples representing the mineral resource in order to establish an optimum beneficiation flowsheet.

Table 16-10. Flotation Test Results.

Element	Head Grade	Rougher Recovery	Cleaner Grade	Cleaner Recovery
Sulphur:	0.11 %	71 %	6.4 %	47 %
Palladium	0.32 g/t	65 %	14 g/t	36 %
Platinum	0.22 g/t	46 %	3 g/t	11 %
Gold	0.05 g/t	43 %	1 g/t	19 %

17.0 MINERAL RESOURCE ESTIMATES

Prior to conducting the resource estimate, the integrity of the entire database was validated as per the methodology described in Section 14 of this report.

17.1 DATABASE DESCRIPTION

The mineral resources for the Big Daddy chromite deposit have been estimated from surface diamond drill holes only. The following is a concise description of the database composition and how the master database used in the resource estimate was derived.

17.1.1 Drill Holes and Assays

The Big Daddy deposit has been tested by 48 drill holes of NQ size on a grid of 100 m between lines taking 2 to 4 holes per line at between 50 m and 100 m apart. The layout is depicted in Figure 11.1. The drill holes cover a strike length of 1 km down to a maximum vertical depth of about 365 m. The assay database consists of 2,974 samples of which the principal analyses were for Cr_2O_3, Al_2O_3, Fe_2O_3, Cr, Fe, SiO_2, and PGEs.

17.1.2 Lithology and Mineralization

All drill holes have the major rock types identified and documented in a "from – to" interval format. The major rock types that have been coded include granodiorite, peridotite, harzburgite/dunite, pyroxenite, gabbro, banded ironstone, mafic volcanic rock, intermediate volcanic rock, felsic volcanic rock, mafic/felsic dykes, dolomite and limestone. The overburden averages about 10 m. The mineralization has also been recorded for each interval as being either massive, semi-massive, intermittent beds, heavily disseminated or disseminated.

17.1.3 Survey

The survey information recorded in the files includes collar co-ordinates, dip, azimuth and down-hole survey data. Collars were laid out relative to a surveyed grid (±0.1 m) and verified by GPS (±0.4 m). Down-hole deviations were measured using Flexit, Deviflex or north-seeking gyro (12 collars).

The Big Daddy project area is monotonously flat and therefore a digital terrain model (DTM) is not critical to the estimation of resources.

17.1.4 Specific Gravity (2,216 determinations)

Specific gravity determinations were carried out broadly following ASTM standard D5779 – 08 (Standard Test Method for Field Determination of Apparent Specific Gravity of Rock and

Manmade Materials for Erosion Control) using an apparatus suggested by Dr. James Franklin, a director of Spider Resources.
Specific gravities were determined after the core was logged and marked for sampling but prior to the splitting/cutting of the core samples. Core was broken to about 35 cm or shorter pieces, the pieces were sequentially numbered to facilitate replacement in the core box, then weighed first in air and then in water. Shattered and excessively broken core was not included due to the difficulty in correctly returning it to the core box.

Micon witnessed these SG determinations during its site visit on October 22, 2009 and is satisfied that the dataset generated is representative of the mineralization encountered at the Big Daddy deposit. Based on 2,216 determinations, the SG data have been evaluated by Cr_2O_3 content and are summarized in Table 17-1

17.1.5 Surpac Master Database

The resource estimate was completed using Surpac Version 6.1.3 Software. The Surpac Master Database was created by importing the data described in Sections 17.1.1 to 17.1.4 from Excel spreadsheet files provided by Spider.

17.2 ESTIMATION DETAILS

17.2.1 Overview of Estimation Methodology

The Big Daddy resource estimate has been conducted using a systematic and logical approach involving geological modelling, conventional statistics, geostatistics, creation of interpolation parameters, block modelling, classification based on both geological and mineralization continuity and finally, block model validation.

17.2.2 Geological Modelling/Interpretation

Based on a detailed analysis of the drill hole logs in conjunction with the assays, the major geological domains as encountered down-hole are dunite, peridotite, massive chromite, pyroxenite and gabbro (Figure 17-1). The sequence of appearance of these domains reflects a fractionation trend in the down-hole direction (northwest to southeast) thus confirming the conclusion that the mafic-ultramafic complex (sill) has been rotated.

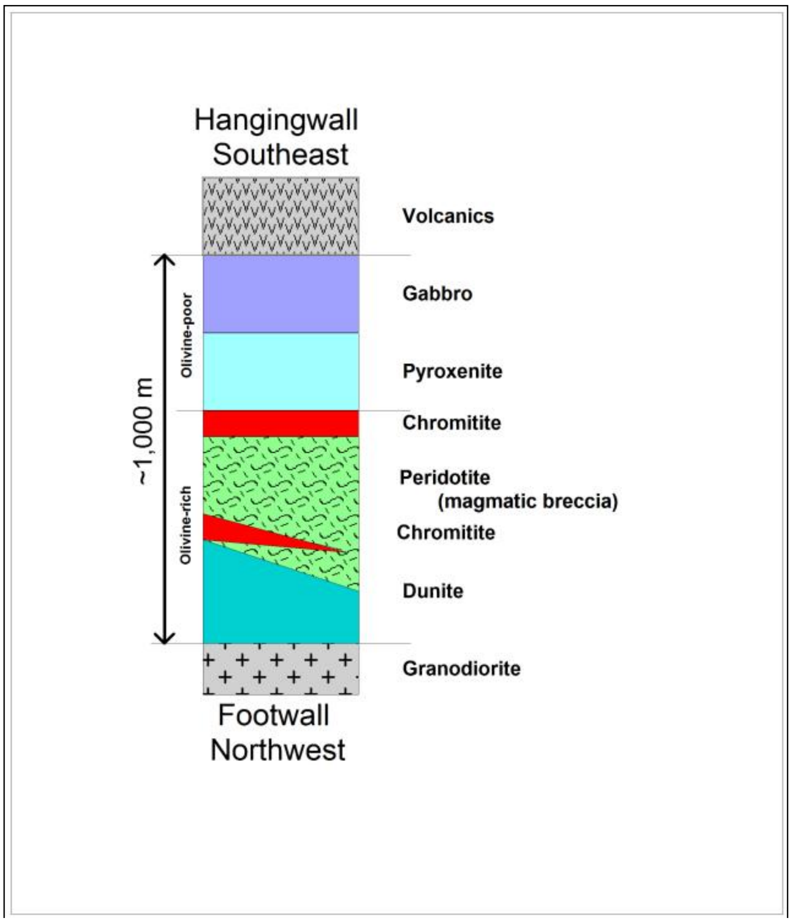
The bulk of the chromite mineralization is confined to the massive chromite domain. However, the peridotite unit does contain sparsely disseminated chromite grains in concentrations varying between 0 and 10% Cr_2O_3. Locally, the chromite mineralization may also occur as either heavily disseminated or semi-massive or intermittent beds within the peridotite. Sectional interpretation of the drilled profiles shows that the massive chromite domain forms a distinct layer with observable continuity laterally and down dip. The deepest drill hole intercept is at a vertical depth of about 365 m below surface, with a true thickness of about 13 m; this thickness suggests that at this depth, the massive chromite layer is far from tapering off or pinching.

A surface trace of the massive chromite domain (based on plots from sectional projections) shows that the Big Daddy deposit comprises two segments which the authors have

Table 17-1. Average Specific Gravity Determinations by Cr_2O_3 Content.

%Cr_2O_3 Range	Density
0 – 15	2.8
15 – 20	3.0
20 – 25	3.2
25 – 30	3.3
30 – 35	3.4
>35	4.0

Figure 17-1. Schematic Lithologic Column for McFaulds Lake Sill, Big Daddy Segment. (Main Geological Domains).



designated BD 1 and BD 2. These are plotted on a gravity map (Figure 17-2) and show a strong correlation between the massive chromite and the gravity anomaly. The subsidiary smaller massive bodies in the footwall are in this report referred to as BD 1 sub and BD 2 sub for BD 1 and BD 2, respectively. A longitudinal section of the two segments is presented in Figure 17-9 which also portrays the distribution of resources. Table 17-2 summarizes the major characteristics of the segments.

The contact/boundary of the massive chromite (labelled as chromitite in Figure 17-1) with both the peridotite and pyroxenite is in the majority of instances very sharp. However, in rare instances the contact with peridotite is gradational from disseminated or intermittent beds or semi-massive chromite. No chromite content high enough to be considered economically significant (>15% Cr_2O_3) has been observed beyond the contact between massive chromite and pyroxenite; therefore the latter boundary is considered critical for geological continuity and has been used in linking massive chromite zones from section to section demonstrating continuity for the entire 450 m to 500 m of each segment.

17.2.3 Statistical Interpretation of Grade Domains

Statistical analysis of the raw data comprising 2,974 samples (2,359 by XRF + 615 by INAA) shows a bimodal distribution (Figure 17-3) representing two extremes, i.e. low grade background mineralization disseminated in peridotite and high grade mineralization in massive chromite. The distribution clearly demonstrates that the mineralization is not fragmented or spread out, but compact. This is consistent with the geological model implying that the high grade mineralization envelope corresponds to the massive chromite geological domain. Using the same graph (Figure 17-3), the top-cut assay for Cr_2O_3 has been set at 45.3% which correspond to the 99.5 percentile. Table 17-3 summarizes the global statistics.

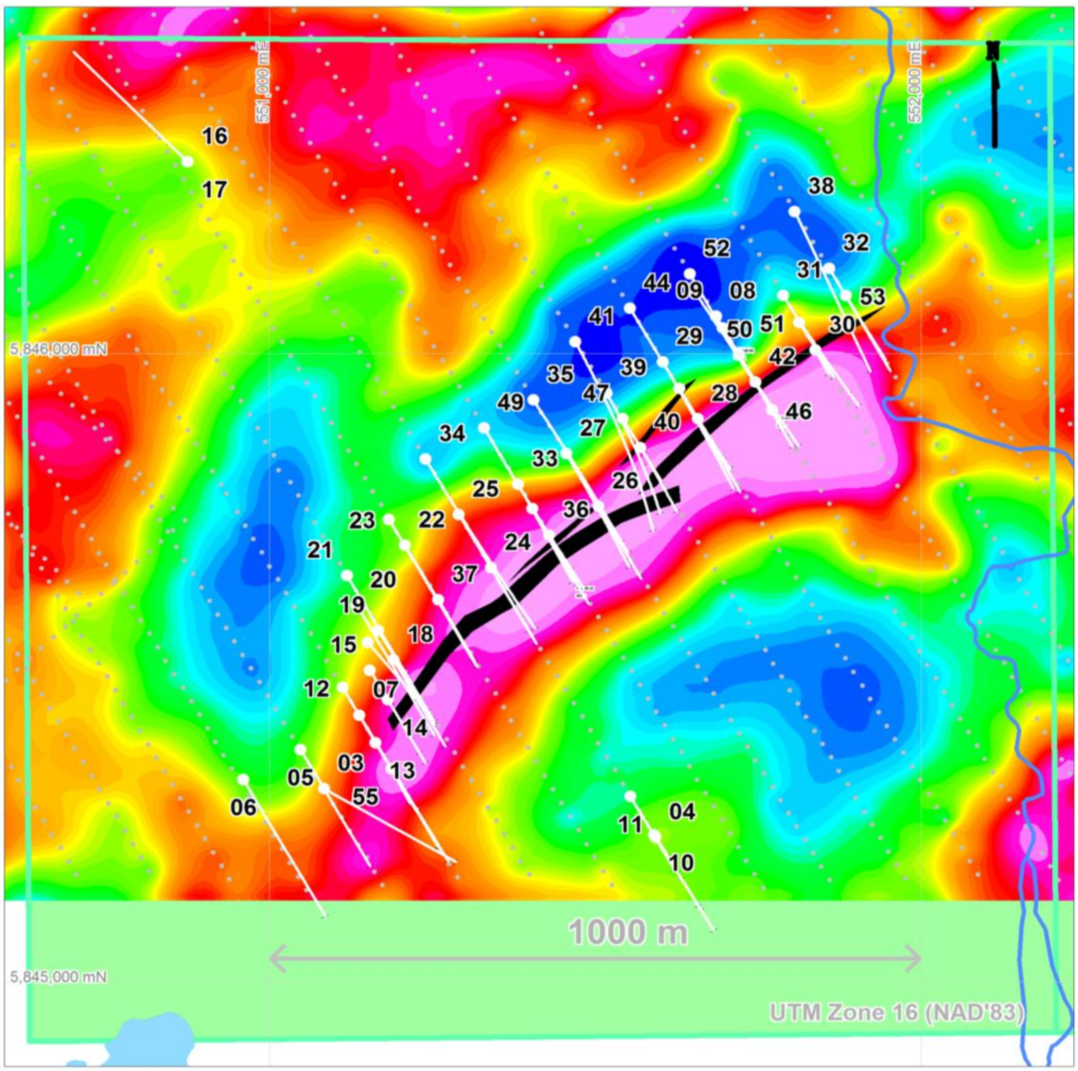
In order to analyze a broader zone of mineralization, a probability plot of the raw data was constructed and a 15% cut-off was selected based on the break in the probability plot (Figure 17-4). The statistics within this zone shows a very strong negative skewness thereby confirming the compactness and high grade nature of the Big Daddy deposit. This is demonstrated in Section 17.2.4.

17.2.4 Composite Data and Grade Domains Statistics

Inspections of drill hole sample intervals augmented by a statistical analysis show that the majority of the sample intervals are 1.5 m. Thus 1.5 m was selected as the standard length (support) and compositing was done to normalize the database to this length.

The statistical distributions of the massive chromite (mineralization domain 1) and the 15% cut-off envelope (mineralization domain 2) are presented in Figures 17-5 and 17-6. The similarity displayed by these distributions is further evidence of a compact distribution of the mineralization. A summary table comparing the statistics of the two grade domains is shown in Table 17-4.

Figure 17-2. Gravity Map Bandpass Filter Gravity (Upper Wavelength is 833 m) with the Massive Chrome Domain Projected to Surface from Sections.



Note: The massive chromite is shown as linear black zones. White dots and numbers denote drill hole collars.

Table 17-2. Summary of the Major Characteristics of the Big Daddy Deposit.

Segment	Approximate Strike Length (m)	Bearing (Degrees)	Dip (Degrees)	Geometry & Mineralization	Avg. True Thickness (m)	Remarks
BD 1	500	Varies between 50 and 60	Varies between -85 East and -90	Tabular; Massive	17	Compact; open down dip; limited potential along strike
BD 2	450	050	Varies between -70 and -80 East	Tabular; Massive	12	Compact; open down dip; limited potential along strike.

(Note: In both cases the footwall subsidiaries are excluded)

Figure 17-3. Histogram of the Raw Assay Data for Cr_2O_3 (%).

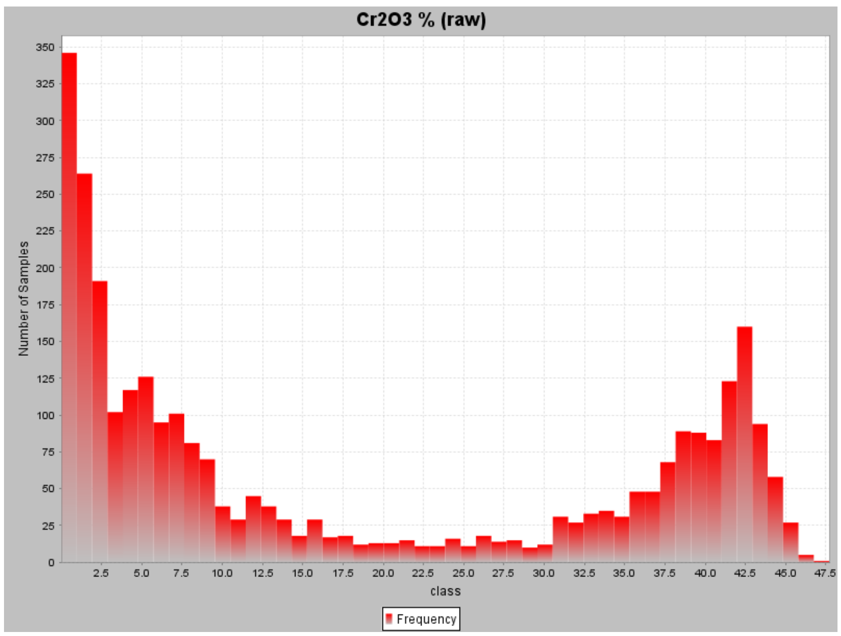


Table 17-3. Global Statistics of the Cr_2O_3 Raw Data.

Variable	Cr_2O_3 %
Lower cut	0.001
Number of samples	2,974
Minimum value	0.005
Maximum value	47.7
	Ungrouped Data
Mean	18.247806
Median	9.435001
Geometric Mean	7.969418
Variance	285.284539
Standard Deviation	16.890368
Coefficient of variation	0.925611
Skewness	0.405344
Kurtosis	1.412646
Natural Log Mean	2.075611
Log Variance	2.697682
10.0 Percentile	0.82
20.0 Percentile	1.8805
25.0 Percentile	2.49
30.0 Percentile	3.69
40.0 Percentile	6.1045
50.0 Percentile (median)	9.435001
60.0 Percentile	20.475
70.0 Percentile	35.275
80.0 Percentile	39.575
90.0 Percentile	42.245
95.0 Percentile	43.27
96.0 Percentile	43.535
97.0 Percentile	43.9
98.0 Percentile	44.345
99.0 Percentile	44.905
99.5 Percentile	45.29
100.0 Percentile	47.7
Sichel-t	30.673167

Figure 17-4. Probability Plot of the Raw Cr_2O_3 Data.

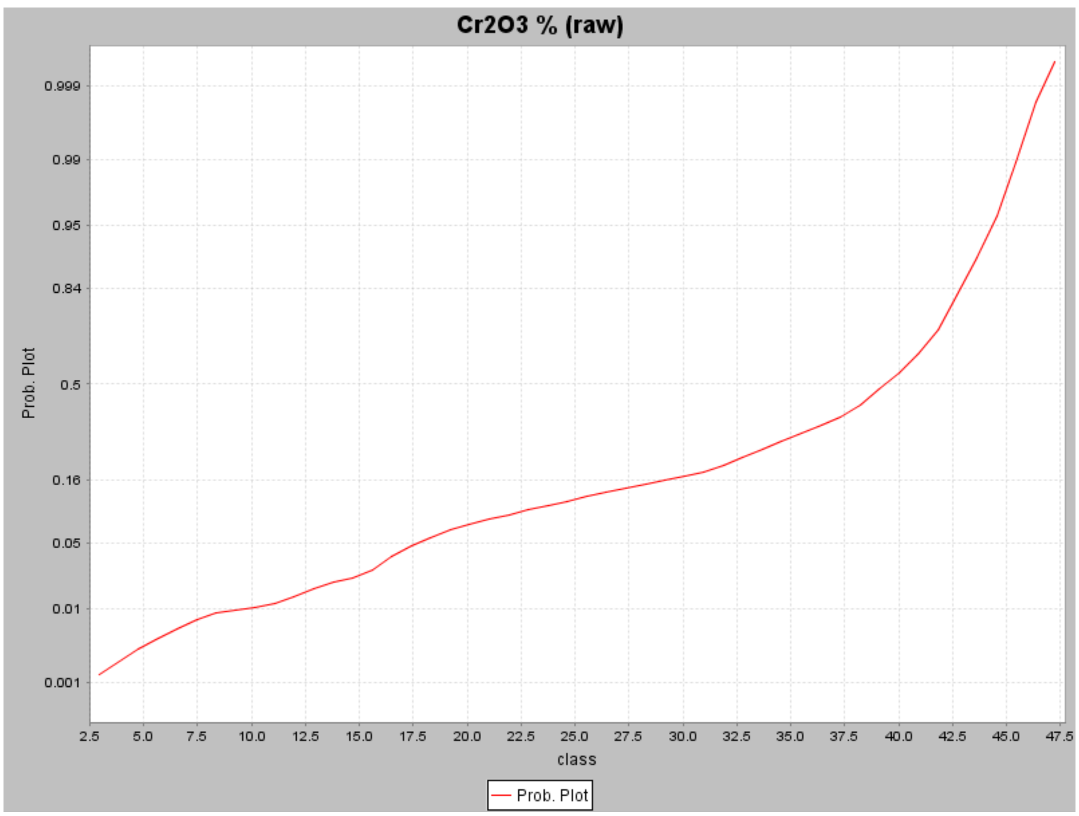
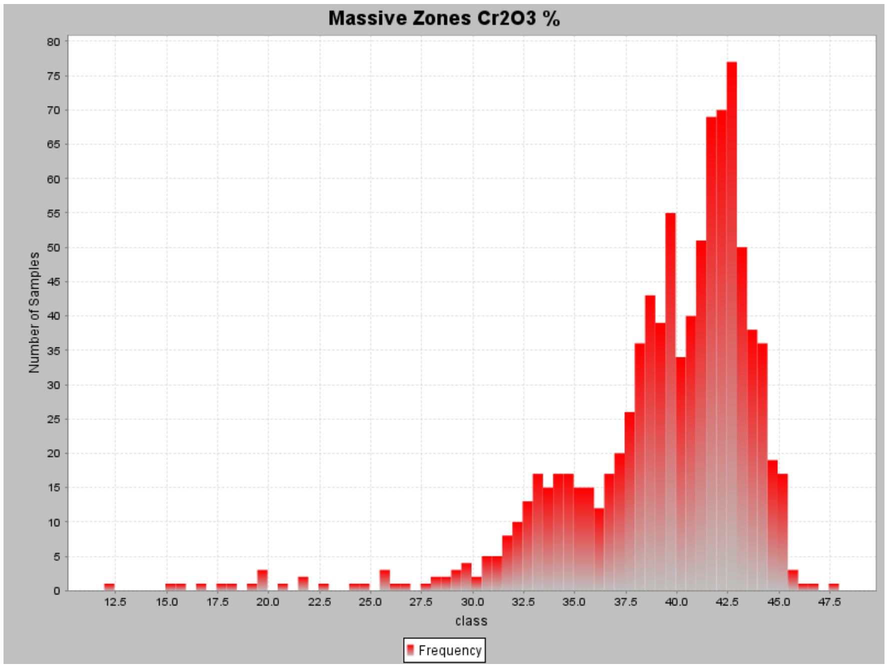
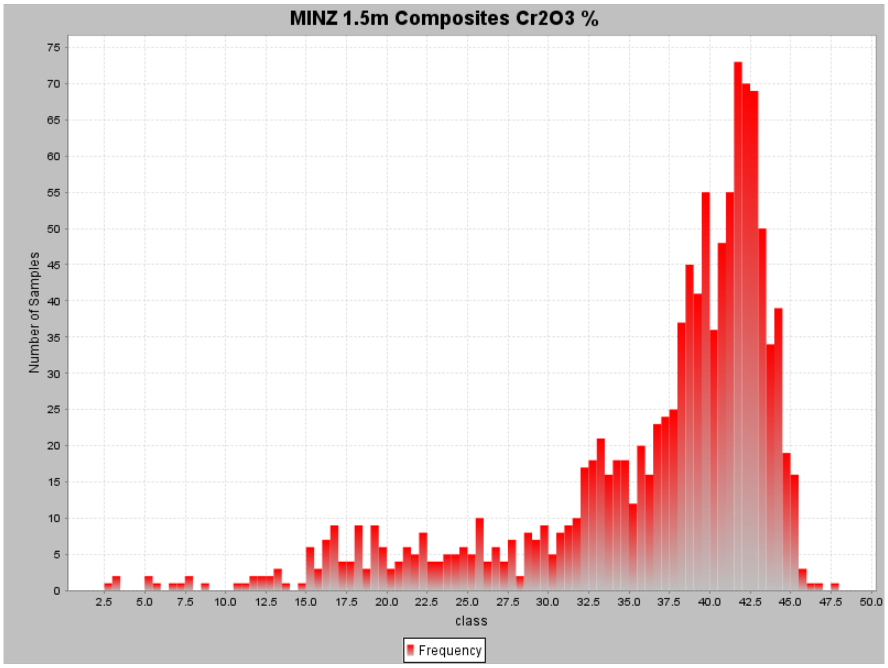


Figure 17-5. Histogram of the Massive Chromite Domain.



(Skewness: -1.90)

Figure 17-6. Histogram of Composites at 15% Cut-off with Internal Waste.



(Skewness: -1.57)

Table 17-4. Composites Summary Statistics of the Massive Chromite Domain and the 15%Cr_2O_3 Cut-off Domain.

Domain	No. of Samples	Min. Value	Max. Value	Mean	Median	Var.	Std	Coef. Var	Remarks
Massive Cr Zone	927	11.94	47.7	39.39	40.61	21.42	4.63	0.12	Includes internal waste in exceptional cases
15% Cut-off Zone	1,149	2.48	47.7	36.50	39.323	66.78	8.17	0.22	Includes internal waste within envelope (maximum 4.5 m)

17.2.5 Cut-off Grade and Economic Parameters

Demand for chromite is mainly for a metallurgical grade product which is around 40% Cr_2O_3 with a Cr:Fe ratio of generally at least 2. Metallurgical grades of this nature currently sell for US$180.00 to US$240.00 per tonne. Currently, the Bushveld complex (South Africa) and the Great Dyke (Zimbabwe) rank high amongst the world producers with many of their operations being underground mines.

The Kemi operations in Finland are mainly open pit with an end of 2009 reserve base of 37 Mt at 26% Cr_2O_3 (Outokumpu, 2009 Annual Report) and a Cr:Fe ratio of about 1.8. However, a portion of the Kemi production is upgraded by means of beneficiation. Thus in the Micon's opinion, two scenarios must be evaluated for the Big Daddy deposit:

- Scenario 1: Focuses on high grade massive material that would produce a lumpy product comparable to South African products with little or no beneficiation.

- Scenario 2: Defines a broad zone of mineralization to match the Kemi situation exploitable by open pit but requiring beneficiation to upgrade.

Hence, resources have been estimated for the massive zone only, and also for the broad zone constrained by a 15% Cr_2O_3 cut-off but including internal waste up to a maximum of 4.5 m. The 15% cut-off is based on the break in the probability plot (Figure 17-4).

17.2.6 Geostatistics

Fundamental geostatistical principles dictate that variography be conducted on data comprising a single population (i.e. samples from geologically homogeneous areas) and on samples representing the deposit (not barren samples). Thus only the massive chromite domain was considered suitable for spatial analysis. The variographic/spatial analysis was conducted to achieve the following:

1. To define the continuity of the mineralization in order to establish (a) the maximum range or distance over which samples and drill hole intercepts may be correlated, and (b) the adequacy of the drilling grid for a resource estimate.

2. To define the optimum parameters for the search ellipse to be used in the interpolation of block grades.

The geometry of the Big Daddy deposit is tabular (stratiform) with the major/principal direction along strike, the semi-major direction down dip and the minor axis across width. Hence, for each segment of the deposit three sets of variograms were computed to cover the three geometrical directions. The experimental variograms and their fitted models are presented in Appendix 3. The down-hole variograms are, as expected, quite stable due to the high density of sample information. The variograms for the major and semi-major axes are generally unstable due to low densities of sample information beyond the 275 m lag. Nonetheless, the variograms give a reasonable reflection of the highly continuous nature of the Big Daddy mineralization. The variogram models were fitted giving weight to the number of pairs in each lag in proportion to the drilling grid and using the variance to establish the sill. A summary of the spatial analysis is presented in Table 17-5.

17.2.7 Interpretation and Application of Spatial Analysis Results

The ranges of influence in the major and semi-major directions are in a broad sense about the same, reflecting the isotropic nature of the massive and compact Big Daddy deposit. The apparent shorter range in the down-hole direction (minor axis) is due only to the restriction imposed by the geometry, i.e. the restricted width of 30 m to 60 m of the deposit.

Taking the lower limit of the major axis reflected in BD 2, the range of influence and, therefore, the maximum distance over which drill intercepts and samples can be correlated is 225 m, indicating highly continuous mineralization. Thus, the drilling grid over the Big Daddy deposit as it stands at approximately 100 m x 50 m, is considered adequate for resource definition to the Indicated category. Similar stratiform chromite deposits in Southern Africa display even higher levels of continuity and ranges of influence.

Based on the ranges of influence, the maximum dimensions of the radii of the search ellipsoid for grade interpolation of the Big Daddy should not exceed 225 m x 225 m x 40 m for an Indicated resource.

The variogram range of influence in the major direction is often used in the categorization of resources. As a general rule, mineral resources are classified as follows:

- Measured Resource when the drill hole spacing is less than the variogram range of influence at 66% or less of the sill. This translates to approximately 110 m for the massive chromite domain.

- Indicated Resource when the drill hole spacing is less than the variogram range of influence at between 66% and 100% of the sill. (100% corresponds to the maximum range of influence beyond which there is no spatial correlation between samples). This translates to 225 m to 250 m for the massive chromite domain.

- Inferred resource when drill hole spacing is beyond the range of influence.

(Reference: PDAC Short course, 2009. “From the Core Barrel to a Resource Estimate.”)

17.2.8 Block Size, Interpolation Search Parameters and Technique

In an ideal situation the longest axis of a block should equal the drill spacing but in practice it is varied between half and a quarter of the spacing. On this basis the longer axis of the block was selected as 25 m. The other dimensions of 10 m and 5 m were based on ideal minimum height and width, respectively, in a selective open pit or mechanized bulk mining situation.

Table 17-5. Summary Results of the Spatial/Variographic Analysis of the Big Daddy Deposits.

Segment	Axis	Direction	Nugget	Structure 1	Range	Bearing	Dip
BD 1	Major	Along strike	0	11	250	60	-90
	Semi-major	Down dip	0	11	200		
	Minor	Down hole	0	21	40		
BD 2	Major	Along Strike	10	14	225	50	-75
	Semi-major	Down dip	10	14	225		
	Minor	Down hole	0	21	40		

In deriving the search radii for the major and semi-major axes, Micon adopted a prudent approach and halved the maximum range of influence as determined by the variography to fit the current spacing between lines of 100 m. For the minor axis, Micon adopted 5 m which is the width of the envisaged mining block.

The inverse-distance-cubed (ID^3) interpolation method was selected as the most ideal to bring out grade patterns inherent in the deposit at a micro-scale due to waste inclusions, particularly for the 15% cut-off domain. The search parameters are summarized in Table 17-6.

For the three passes, the maximum number of samples per drill hole is designed to manage and control the number of drill holes in the interpolation.

For Pass 1, the minimum and maximum number of samples for each interpolation is designed to ensure that the nearest sample(s) is/are accorded the highest weighting and that a maximum of the three closest holes are used in the interpolation.

For Pass 2, the minimum number of samples for interpolation is designed to ensure a minimum of two drill holes in the interpolation while the allowable maximum samples per interpolation are increased to twenty to go beyond the limits of Pass 1.

For Pass 3, the minimum number of samples for interpolation allows the interpolation to fill all the space in the solid. The maximum number of samples per interpolation is increased to 30 to allow the bigger ellipse to find at least a second hole for interpolation.

17.2.9 Block Modelling Description

Domain model solids were created to encompass the limits of the components of the deposit as defined by the geological interpretation. For scenario 1, only the massive chromite intercepts were considered with no allowance for internal dilution, except in <5% of the cases where linking sections dictated otherwise. For scenario 2, the 15% Cr_2O_3 cut-off envelope was used allowing for a maximum of 4.5 m of internal waste. (Note: The 4.5 m allowable internal waste equates to three samples and is just under the envisaged block width of 5 m).

An inclined, rotated, partial-percentage block model (i.e. the percentage of any block that is contained within the domain model is used to weight the volume and tonnage reports), with the long axis of the blocks oriented along an azimuth varying between 065 degrees and 050 degrees (parallel to the average domain orientation) and dipping at between -70 degrees and -90 degrees.

Cr_2O_3 grades and Cr/Fe ratios were interpolated into the individual blocks of the mineralized domains using ID^3. Ordinary kriging was used to run a parallel estimate to validate the ID^3 results.

Table 17-6. Summary of Search Parameters.

Attribute	Pass 1	Pass 2	Pass 3
Major axis search radius (m)	100	200	400
Semi-major axis search radius (m)	100	200	400
Minor axis search radius (m)	5	10	20
Maximum # of samples/drill hole	3	3	3
Minimum # of samples/interpolation	5	3	3
Maximum samples/interpolation	10	20	30
Interpolation method	ID^3	ID^3	ID^3

17.3 CLASSIFICATION CRITERIA AND BLOCK MODELLING RESULTS

17.3.1 Classification Criteria

The mineral resources in this report were estimated in accordance with the definitions contained in the CIM Definition Standards on Mineral Resources and Mineral Reserves that were prepared by the CIM Standing Committee on Reserves Definitions and adopted by the CIM Council on December 11, 2005.

The mineralized material was classified into either the Indicated or Inferred mineral resource category on the basis of a combination of the following factors: (a) confidence in the geological and mineralization continuity, (b) position of blocks in relation to the range of influence as defined by the variographic analysis and (c) and the search ellipse ranges presented in Table 17-4.

17.3.2 Responsibility For Estimation

The Micon staff with responsibility for this resource estimate are Alan J. San Martin, and Charley Murahwi. All are Qualified Persons as defined in NI 43-101, and are independent of the SKF parties.

17.3.3 Statement of Results

Following the concepts and processes described above, the mineral resources for the Big Daddy deposit were estimated and include all blocks that are located within the domain models of the two scenarios. The results of the block model are summarized in Tables 17-7 and 17-8, and are exclusive of the overburden tonnages. The respective block models are presented in Figures 17-7 and 17-8.

17.3.4 Comments

The block model grades for the massive chromite domain as displayed in Figure 17-7 are fully supported by the distribution of drill hole intercept grades seen in Figure 17-9. The distribution of the Indicated and Inferred Resources within the block model is presented in Figures 17-10 and 17-11 for the massive and 15% cut-off domains, respectively.

<u>Indicated Mineral Resource</u>

The CIM Definition Standards for Mineral Resources and Mineral Reserves of December, 2005 state that:

> *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*

Table 17-7. Summary of the Big Daddy Massive Chromite Resources.

Deposit/Code	Category	Cr_2O_3% Interval	Tonnes x 10^6	Avg. Cr_2O_3%	Cr/Fe Ratio
BD 1 (100)	Indicated	>35.0	12.934	40.74	2.0
		30.0 – 35.0	0.435	33.63	1.8
		25.0 – 30.0	0.017	28.87	1.7
		20.0 – 25.0	0	0	0
		15.0 – 20.0	0	0	0
Sub-total			13.4	40.49	2.0
BD 2	Indicated	>35.0	9.234	41.44	2.0
		30.0 – 35.0	0.520	32.83	1.8
		25.0 – 30.0	0.090	29.36	1.7
		20.0 – 25.0	0	0	0
		15.0 – 20.0	0	0	0
Sub-total			9.8	40.88	2.0
Grand Total	**Indicated**		**23.2**	**40.66**	**2.0**
BD 1 (100)	Inferred	>35.0	6.216	39.34	2.0
		30.0 – 35.0	1.014	33.25	1.8
		25.0 – 30.0	0.005	27.97	1.7
		20.0 – 25.0	0	0	0
		15.0 – 20.0	0	0	0
Sub-total			7.2	38.48	2.0
BD 2	Inferred	>35.0	8.382	40.24	2.0
		30.0 – 35.0	0.609	33.32	1.8
		25.0 – 30.0	0.047	28.35	1.7
		20.0 – 25.0	0.021	22.87	1.5
		15.0 – 20.0	0.042	16.76	1.1
		.01 – 15.0	0	0	0
Sub-total			9.1	39.57	2.0
Grand Total	**Inferred**		**16.3**	**39.09**	**2.0**

Note: The tonnages have been rounded to 3 decimals for grade intervals and to 1 decimal for sub-totals and grand totals.

Table 17-8. Summary of the Big Daddy Chromite Deposit Mineral Resource @ 15% Cr_2O_3 Cut-off.

Deposit/Code	Category	Cr_2O_3% Interval	Tonnes	Avg. Cr_2O_3%	Cr/Fe Ratio
BD 1 (100)	Indicated	>35.0	13.535	40.22	2.0
		30.0 – 35.0	1.333	32.98	1.8
		25.0 – 30.0	0.447	27.77	1.7
		20.0 – 25.0	0.152	23.34	1.5
		15.0 – 20.0	0.019	17.81	1.1
		0.01 – 15.0	0.001	12.09	0.7
Sub-total			15.5	39.05	2.0
BD 2	Indicated	>35.0	9.622	41.11	2.0
		30.0 – 35.0	1.031	32.97	1.8
		25.0 – 30.0	0.190	28.04	1.7
		20.0 – 25.0	0.007	22.56	1.4
		15.0 – 20.0	0.009	18.46	1.2
		0.01 – 15.0	0.087	7.74	0.6
Sub-total			10.9	39.82	1.9
Grand Total	**Indicated**		**26.4**	**39.37**	**2.0**
BD 1 (100)	Inferred	>35.0	7.097	39.14	2.0
		30.0 – 35.0	1.877	32.94	1.8
		25.0 – 30.0	0.543	27.93	1.7
		20.0 – 25.0	0.349	22.58	1.4
		15.0 – 20.0	0.174	18.33	1.1
		0.01 – 15.0	0.016	9.17	0.6
Sub-total			10.1	36.40	1.9
BD 2	Inferred	>35.0	8.993	39.80	2.0
		30.0 – 35.0	0.986	32.89	1.8
		25.0 – 30.0	0.241	28.06	1.7
		20.0 – 25.0	0.123	23.11	1.5
		15.0 – 20.0	0.059	16.90	1.0
		.01 – 15.0	0.014	11.96	0.9
Sub-total			10.4	38.51	2.0
Grand Total			**20.5**	**37.47**	**1.9**

(Includes internal waste within the 15% Cr_2O_3 envelope up to a maximum of 4.5m).

Note: The tonnages have been rounded to 3 decimals for grade intervals and to1 decimal for sub-totals and grand totals.

Figure 17-7. Block Model of the Massive Domain of the Big Daddy Chromite Deposit.

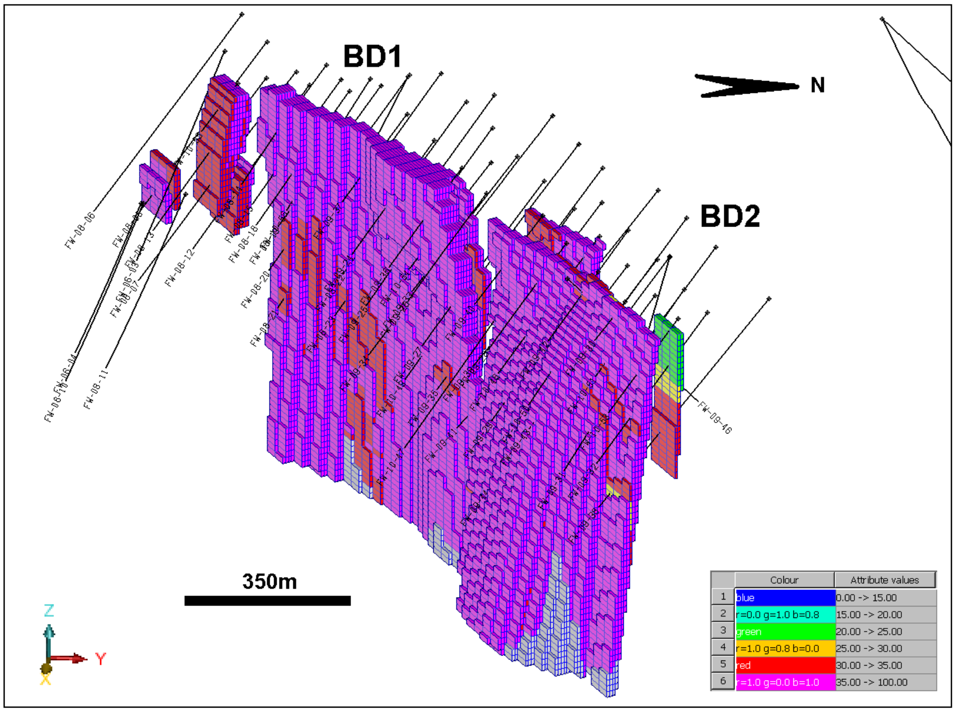
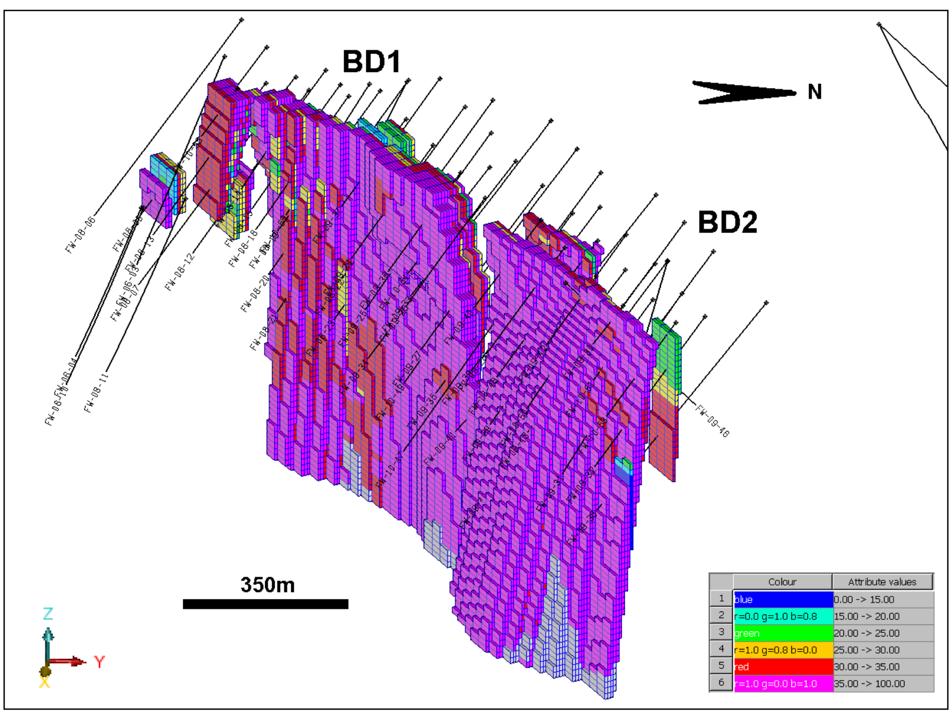


Figure 17-8. Block Model of the Big Daddy Chromite Zone Constrained at 15% Cr_2O_3 Cut-off.



Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

On the evidence of the geological model/interpretation, statistical and spatial analysis, the Big Daddy deposit demonstrates a high level of continuity in the mineralization both in the lateral and vertical sense. The geological continuity is equally demonstrated, although minor displacements of the deposit, if any, may not have been revealed on the 50 x 100 m grid. Nonetheless, the broad zone of continuity along strike (Figure 17-2) and down dip (Figure 17-9) is sufficiently defined to justify the categorization of the drilled part of the deposit as an Indicated resource.

Inferred Mineral Resource

In accordance with the CIM definition of Inferred Resources , the portion of the Big Daddy deposit below the -220 m elevation for BD 1 and -160 m for BD 2, and all satellite bodies the geological continuity of which is questionable, have been categorized as Inferred. The bulk of the Inferred category of the major components of the deposit remains to be drill tested. Nonetheless, the lower limit of the inferred resource (at 600 m below surface) is considered appropriate. This interpretation is based on:

- The large thicknesses of the massive chromite encountered in the line of the deepest holes suggesting that, at between 350 m and 400 m depth, the deposit is not narrowing at depth.

- A Magnetic 3-D inversion which suggests that the ultramafic rocks hosting the chromite mineralization extend to a depth of +/- 1,700 m.

- Experience with similar type deposits: The sill hosting the chromite mineralization is known to extend for a lateral distance of over 12 km from Blackbird in the southwest to beyond Black Thor in the northeast. Thus, a depth extension of 600 m is conceivable and considered conservative by analogy with similar intrusions like the Stillwater, Bushveld and Great Dyke Complexes. The relatively thin (<1 m) chromite layers of the Great Dyke are known to be persistent for several km down dip. Recent geophysical investigations at the Kemi deposit indicate persistent mineralization at great depth. The Big Daddy ultramafic-mafic rocks may be part of a much larger intrusion or magmatic complex, extending at least 50 km along strike (Naldrett, 2009).

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Figure 17-9. Sketch of Longitudinal Section of the Big Daddy Deposit, Looking West.

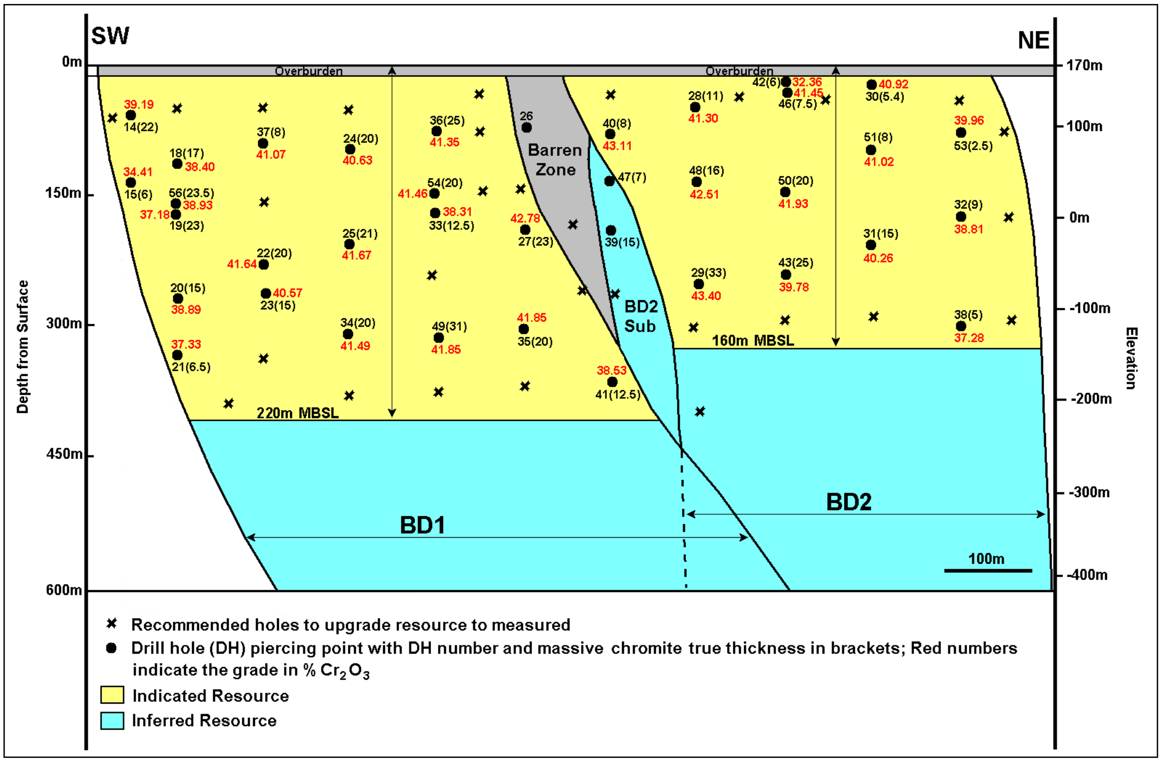


Figure 17-10. Distribution of Resources within the Block Model for the Massive Domain.

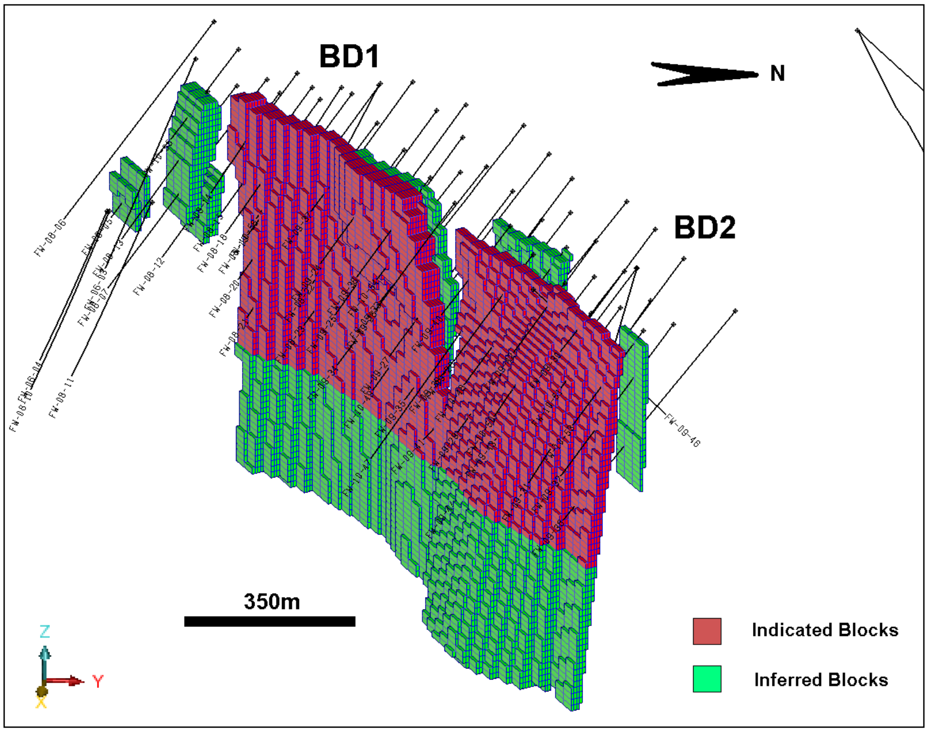
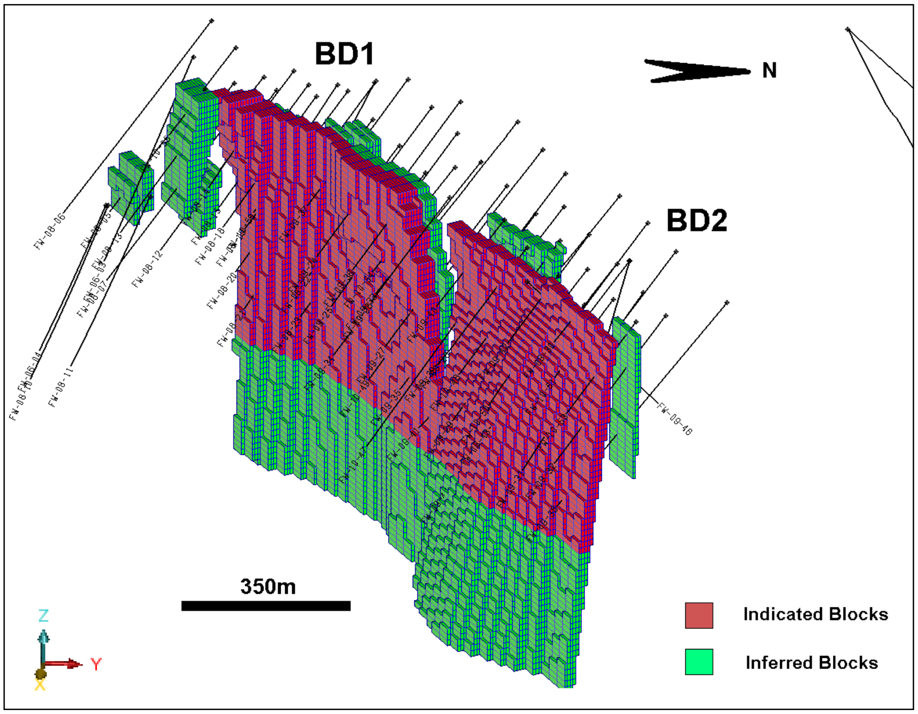


Figure 17-11. Distribution of Resources within the Block Model constrained at 15% Cut-off



17.3.5 Validation

Validation of the block model and tonnages was conducted manually using sectional and polygonal techniques and by ordinary kriging. A comparison of results obtained using ordinary kriging and ID^3 is presented in Table 17-9.

17.3.6 Qualification of the Mineral Resources

Micon believes that at present there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues which would adversely affect the mineral resources estimated above. However, mineral resources, which are not mineral reserves, do not have demonstrated economic viability. Micon cannot guarantee that the SKF parties will be successful in obtaining any or all of the requisite consents, permits or approvals, regulatory or otherwise for the project. Other future setbacks may include aboriginal challenges to title or interference with ability to work on the property and lack of efficient infrastructure. There are currently no mineral reserves on the Big Daddy property and there is no assurance that the project will be placed into production.

17.3.7 Potential Upgrading of the Indicated Resource

It is considered likely that, in order to upgrade the Indicated resource to the Measured category, a few strategically positioned drill holes will suffice. These positions are marked on the sketch long section shown in Figure 17-9. Additional holes are unlikely to improve the grade but may assist in revealing minor displacements.

Table 17-9. Summary of Global Results of ID^3 Versus Ordinary Kriging (OK).

Description	OK Blocks	ID^3 Blocks
Count	15,645	15,645
Mean (% Cr_2O_3)	39.26	39.32
Median (% Cr_2O_3)	40.07	40.25
Variance	10.51	12.49
Standard Deviation	3.24	3.53
Coefficient of variation	0.08	0.09

18.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES.

18.1 MINING

The known Big Daddy mineral resource would be mined by open pit. The resource would support a mining rate of 8,000 tonnes per day, of potentially economic lump chromite mineralization.

Open pit mining will be by conventional diesel-powered equipment, utilizing a combination of blasthole drills, hydraulic excavators, shovels, rubber-tired wheel loaders and a mixed fleet of 144t and 327t trucks. Support equipment composed of graders, track dozers, and a water truck will aid in the mining of the mineral resource and waste. Potentially economic lump chromite mineralization will be hauled and dumped in a crusher located to the west of the open pit. Figure 18.1-1 shows the general facilities layout of the site.

18.1.1 Geotechnical

Detailed geotechnical work has not been completed to date. The general rock types and observed quality of the rock suggests that the wall rocks should be competent. Open pit slopes which are similar to average pit slopes in other Canadian operations have been applied in this study. A full geotechnical program should be initiated in order to optimise pit slopes. This program should include a battery of oriented core holes drilled to test all structures, interpretation of the geotechnical data, rockmass classification, domain delineation and a kinematic pit slope stability assessment using numerical modelling.

18.1.2 Open Pit Optimization

The geological block model was estimated within a mineralized solid with blocks containing a percentage of mineralized material within each block. For optimization and mine design the blocks along the mineralized boundary were diluted in order to simulate the selective mining unit size (SMU) of 5 metres by 25 metres by 10 metres. From this information, a global grade tonnage chart was created to assess the material quantities at various grade cut-offs. This chart is shown in Figure 18.1-2. The bulk of the material within the deposit has a grade between 30% and 45% Cr_2O_3.

Open pit optimization was carried out using the Lerch-Grossman algorithm and MineSite open pit optimization software. After analysis of the pit optimization results, a base case economic open pit was selected. The base case assumed all material (called lump chromite) above a chromite grade of 35% to be amenable to direct shipment for smelting. The lump chromite mineralization requires only sizing to minus 50 mm before shipment for smelting. It has been assumed that this material would receive a price of US$325/t, (the expected metal price for the PEA's conceptual open pit design).

Figure 18.1-1. Mining Facilities Plan.

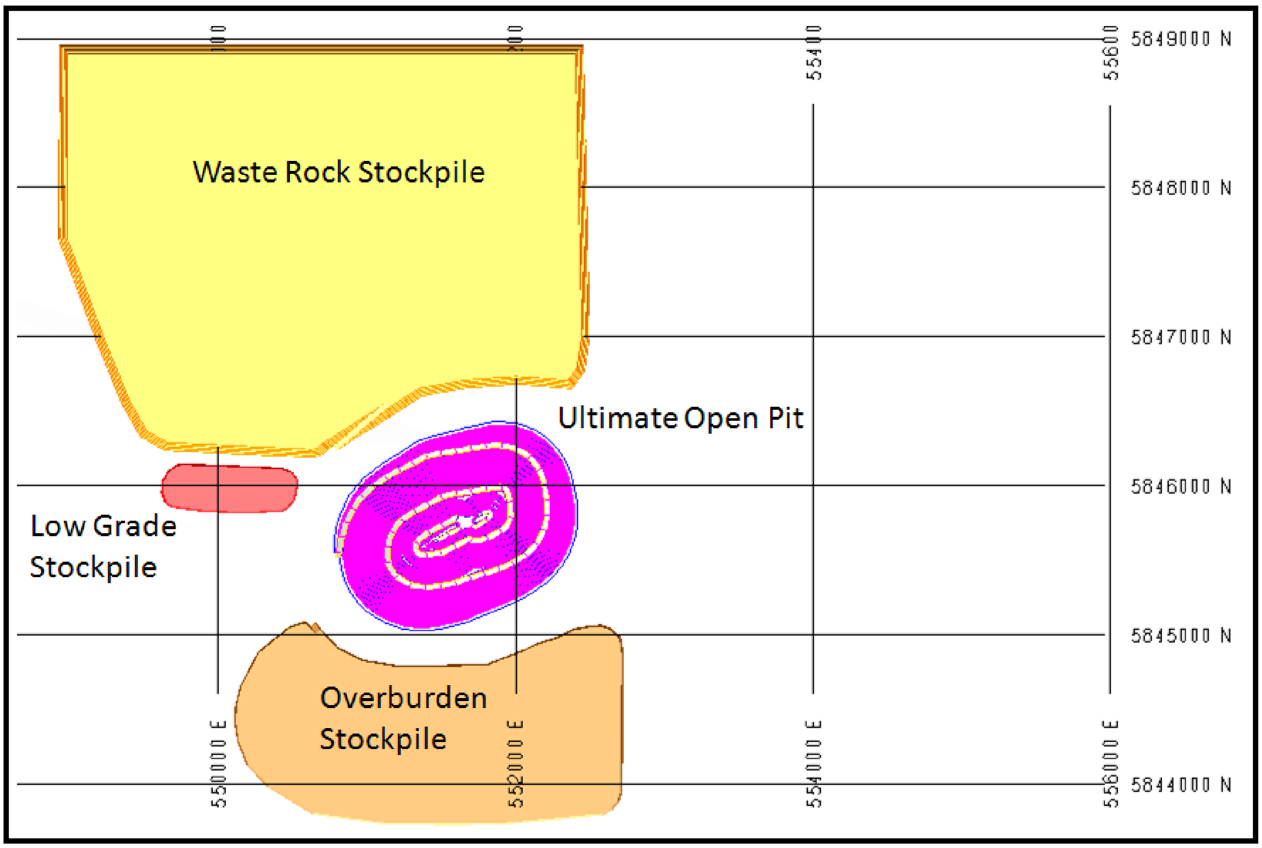
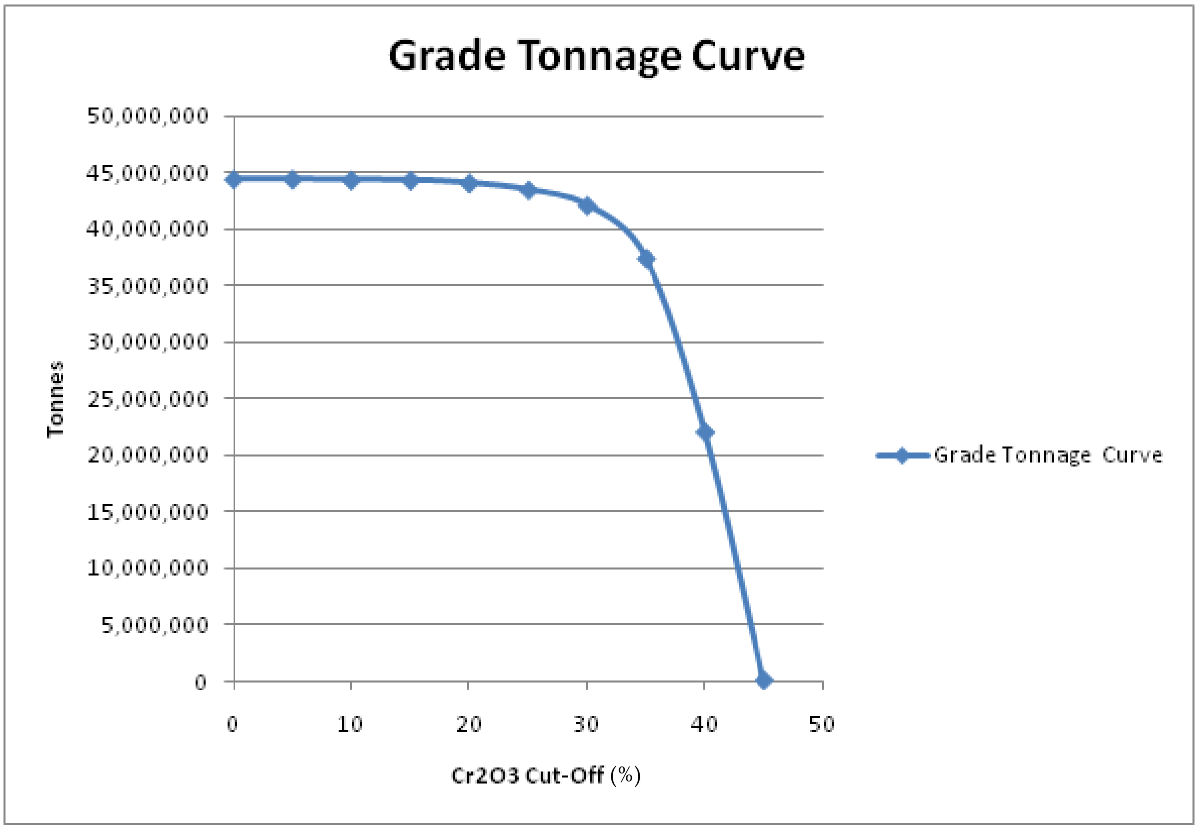


Figure 18.1-2. Big Daddy Grade Tonnage Curve.



Open pit optimization is based on preliminary economic estimations of mining, processing and selling related costs and slope angles. These open pit optimization factors are likely to vary from those reported in the final economic analysis, which are based on the final open pit design and production schedule. The open pit optimization software considered grades and tonnages in the model along with mining and product preparation sizing factors and costs to determine what material could be economically extracted, (through the use of the Lerch-Grossman algorithm).

Because the open pit option considers only material suitable for direct smelting, potentially economic lump chromite mineralization requires an average grade of 35% to 40% Cr_2O_3. For this reason, a minimum cut-off grade of 35% Cr_2O_3 was applied. This cut-off grade ensured that the average grade achieved maintained the best potential direct smelting grade for this deposit. Further, based on current marketing studies, an average grade of 35% Cr_2O_3 appears to be an approximate lower grade value for inclusion as potentially economic lump chromite mineralization. More work on this minimum grade value for potentially economic lump chromite mineralization must be undertaken in future studies to improve and confirm the open pit optimizations.

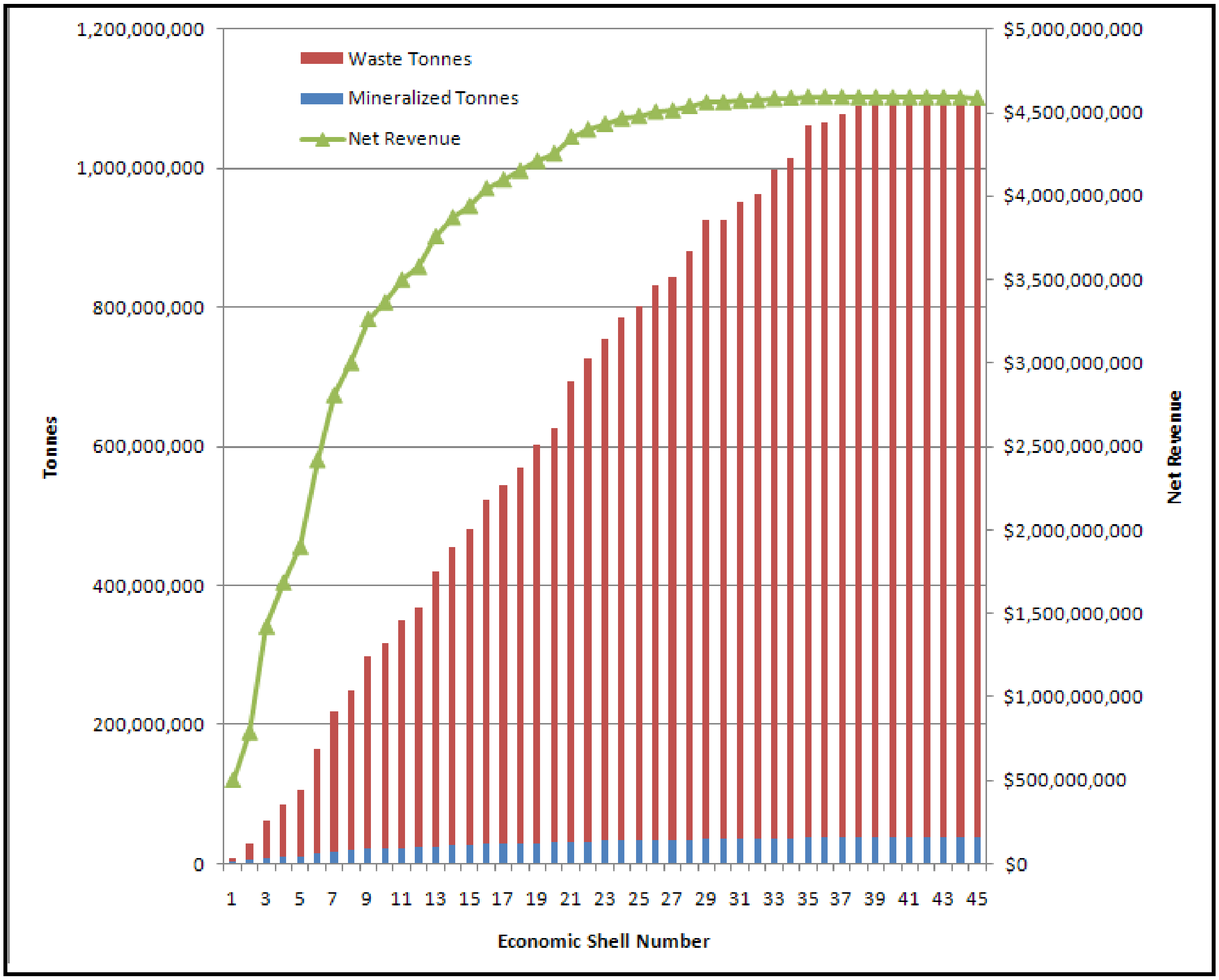
Open pit optimization was undertaken with initially estimated parameters and operating costs as follows:

Waste Density (tonnes/m3)	2.72
Overall Open Pit Slopes (degrees)	45 degrees
Production Rate (ore tonnes per year)	2,880,000
Processing Recovery (%)	100%
Ore Mining Cost ($/tonne)	$2.70
Waste Mining Cost - Overburden ($/tonne)	$3.50
Waste Mining Cost – Rock ($/tonne)	$2.20
Ore Processing Cost($/tonne)	$5.00
General & Administration Cost ($/ore tonne)	$ 2.50
Railway Transport Cost ($/tonne)	$80
Overseas Shipping Cost ($/tonne)	$45
Chromite Ore Price ($US/tonne)	$325
Exchange Rate $CDN:$US	1
NSR Royalty (%)	2%

Figure 18.1-3 shows a graph of open pit shell values that represent how the deposit responds to different revenue factors or price manipulations. An expected lump chromite net price of $196/t was calculated by subtracting the transportation costs and the royalty charge from the assumed lump chromite price of $325/t. On this basis revenue factors ranging from 15% ($29.40/t) to 125% ($245/t) were evaluated. Open Pit 34 represents a revenue factor of 1.0, which equates to the maximum cash flow possible for the deposit from a gross price of $325/t and the costs shown above. The green line on the graph shows the

net revenue for each economic open pit whereas the column plots show the mineralized and waste tonnes respectively.

Figure 18.1-3. Open Pit Shells Values.



The open pit optimization algorithm cash flow for the best-case mining scenario (nested pit) shows a rapid increase until pit 28, after which it flattens out showing incremental stripping is required to liberate a small amount of additional material for little additional cash flow (See Table 18.1-1). Pit 28 would provide the basis for an optimal pit, however, for the current study the base case economic open pit, pit 34, was used as the basis for the open pit design. This open pit has a stripping ratio of approximately 28.5:1 and adds additional net revenue of $54,680,869 over pit 28.

18.1.3 Potentially Mineable Lump Chromite Mineralization Resource Estimate

The potentially mineable lump chromite mineralization open pit resource is estimated to be 25,356,441 tonnes at a grade of 38.02% Cr_2O_3 of Indicated Resources and 13,545,259 tonnes at a grade of 37.03% Cr_2O_3 of Inferred Resources, to an ultimate open pit depth of 570 metres. Dilution of 10 percent at zero grade was included as well as losses of 3% that would result from mineralized material being sent to the waste stockpile. This estimate was derived utilizing the lump chromite price of US$325 per tonne Cr_2O_3 open pit shell produced by MineSight. The internal cut-off grade was not used for reporting. Rather a cut-off grade of 35% Cr2O3 was used as it yields run off mine lump chromite mineralization ready for direct smelting. A total potential diluted mineable resource including indicated and inferred material of 38,901,700 tonnes of lump chromite mineralization at an average grade of 37.69% Cr_2O_3 and 19.72 Fe_2O_3 is produced.

The internal cut-off grade was calculated to determine the amount of material that is available for stockpiling and potential sale and/or processing later in the mine life. An internal cut-off grade of 3.83% was derived from processing and general administration costs. Typically, this internal cut-off is the minimum ore grade required to breakeven from the open pit edge onwards in the processing stream. However, for the current study this material was not given any value during the economic open pit runs and therefore did not play a role in determining the ultimate open pit limits. On this basis a stockpile of 7,572,300 tonnes of indicated and inferred potentially economic lump chromite mineralization grading 28.27% Cr_2O_3 and 17.59% Fe_2O_3 was established and represents the material within the open pit limits above a cut-off grade of 3.83% Cr_2O_3, but below the lump chromite mineralization cut-off grade of 35%.

18.1.4 Open Pit Design

A multi-phase open pit design was created, with 3 phases, where in each phase approximately equal quantities of potentially economic lump chromite mineralization were extracted. Phase 1 and 2 mine lump chromite mineralization classified predominately as indicated and the last phase mines primarily the inferred material found at depth. Open pit design parameters are shown in Table 18.1-2. With further geotechnical evaluation and testing, an optimization of the slope angles can take place.

Table 18.1-1. Open Pit Optimization Results.

Pit #	Reven. Factor (%)	Value ($/t)	Ind. Ore (Tonnes)	Cr2O3 (%)	Fe2O3 (%)	Inf. Ore (Tonnes)	Cr2O3 (%)	Fe2O3 (%)	Total Ore (Tonnes)	Cr2O3 (%)	Fe2O3 (%)	Overburden (Tonnes)	Waste (Tonnes)	Total Waste (Tonnes)	Total (Tonnes)	SR W:O
1	15.00	29.40	2,500,317	39.90	21.46	293,401	37.82	20.92	2,793,718	39.68	21.40	1,933,229	5,632,495	7,565,724	10,359,442	2.71
2	17.50	34.30	4,075,016	40.31	21.40	811,803	38.52	20.36	4,886,819	40.01	21.23	19,945,465	23,317,809	43,263,274	48,150,093	8.85
3	20.00	39.20	7,187,166	40.41	21.19	1,193,603	38.46	20.40	8,380,769	40.13	21.07	5,332,311	54,263,745	59,596,056	67,976,825	7.11
4	22.50	44.10	8,661,366	40.55	21.01	1,386,303	38.45	20.44	10,047,669	40.26	20.93	6,151,898	74,082,835	80,234,733	90,282,402	7.99
5	25.00	49.00	9,846,515	40.64	20.91	1,609,903	38.35	20.43	11,456,418	40.32	20.84	6,977,584	93,739,533	100,717,117	112,173,535	8.79
6	27.50	53.90	13,037,777	40.72	20.73	1,912,453	38.29	20.53	14,950,230	40.41	20.70	8,764,941	150,070,311	158,835,252	173,785,482	10.62
7	30.00	58.80	15,452,439	40.77	20.67	2,221,153	38.27	20.48	17,673,592	40.46	20.64	10,212,014	200,976,753	211,188,767	228,862,359	11.95
8	32.50	63.70	16,741,988	40.80	20.70	2,340,903	38.28	20.45	19,082,891	40.49	20.67	10,961,810	230,514,428	241,476,238	260,559,129	12.65
9	35.00	68.60	18,562,488	40.79	20.74	2,488,223	38.25	20.41	21,050,711	40.49	20.70	12,052,081	276,360,813	288,412,894	309,463,605	13.70
10	37.50	73.50	19,276,538	40.76	20.73	2,556,523	38.21	20.40	21,833,061	40.46	20.69	12,666,645	295,855,126	308,521,771	330,354,832	14.13
11	40.00	78.40	20,204,238	40.74	20.75	2,734,323	38.45	20.44	22,938,561	40.47	20.71	13,448,029	326,004,247	339,452,276	362,390,837	14.80
12	42.50	83.30	20,707,838	40.72	20.77	2,876,873	38.59	20.48	23,584,711	40.46	20.73	13,928,248	344,878,394	358,806,642	382,391,353	15.21
13	45.00	88.20	21,809,587	40.67	20.79	3,365,373	38.94	20.62	25,174,960	40.44	20.77	15,103,998	394,294,110	409,398,108	434,573,068	16.26
14	47.50	93.10	22,111,737	40.66	20.79	4,091,173	39.10	20.79	26,202,910	40.42	20.79	15,852,451	429,051,398	444,903,849	471,106,759	16.98
15	50.00	98.00	22,312,387	40.65	20.80	4,554,973	39.18	20.88	26,867,360	40.40	20.81	16,441,090	452,910,339	469,351,429	496,218,789	17.47
16	52.50	102.90	22,578,687	40.65	20.81	5,371,773	39.25	20.96	27,950,460	40.38	20.84	17,378,799	494,244,055	511,622,854	539,573,314	18.30
17	55.00	107.80	22,726,737	40.64	20.81	5,769,181	39.30	20.98	28,495,918	40.37	20.85	17,888,518	516,177,785	534,066,303	562,562,221	18.74
18	57.50	112.70	22,787,787	40.64	20.81	6,297,787	39.33	21.03	29,085,574	40.35	20.86	18,368,898	540,841,027	559,209,925	588,295,499	19.23
19	60.00	117.60	22,921,237	40.63	20.82	6,867,372	39.40	21.07	29,788,609	40.35	20.88	19,066,735	571,884,497	590,951,232	620,739,841	19.84
20	62.50	122.50	23,039,937	40.62	20.83	7,286,922	39.42	21.10	30,326,859	40.33	20.89	19,632,600	596,360,759	615,993,359	646,320,218	20.31
21	65.00	127.40	23,170,187	40.61	20.83	8,479,522	39.48	21.17	31,649,709	40.31	20.92	20,881,611	661,465,898	682,347,509	713,997,218	21.56
22	67.50	132.30	23,259,937	40.60	20.84	9,033,222	39.49	21.20	32,293,159	40.29	20.94	21,534,653	694,584,151	716,118,804	748,411,963	22.18
23	70.00	137.20	23,314,537	40.60	20.84	9,499,272	39.49	21.22	32,813,809	40.28	20.95	21,993,715	722,623,454	744,617,169	777,430,978	22.69
24	72.50	142.10	23,380,787	40.59	20.85	9,977,622	39.50	21.25	33,358,409	40.27	20.97	22,626,637	752,997,896	775,624,533	808,982,942	23.25
25	75.00	147.00	23,395,287	40.59	20.85	10,243,772	39.51	21.25	33,639,059	40.26	20.97	22,922,368	769,072,595	791,994,963	825,634,022	23.54
26	77.50	151.90	23,422,837	40.59	20.85	10,705,872	39.50	21.27	34,128,709	40.25	20.98	23,447,041	798,187,111	821,634,152	855,762,861	24.07
27	80.00	156.80	23,430,687	40.59	20.85	10,872,122	39.51	21.28	34,302,809	40.25	20.99	23,680,331	808,680,665	832,360,996	866,663,805	24.27
28	82.50	161.70	23,473,437	40.58	20.85	11,435,122	39.51	21.31	34,908,559	40.23	21.00	24,427,430	846,787,418	871,214,848	906,123,407	24.96
29	85.00	166.60	23,498,237	40.58	20.86	12,047,422	39.51	21.34	35,545,659	40.22	21.02	25,145,892	889,568,575	914,714,467	950,260,126	25.73
30	87.50	171.50	23,498,237	40.58	20.86	12,065,972	39.51	21.34	35,564,209	40.22	21.02	25,165,331	890,929,805	916,095,136	951,659,345	25.76
31	90.00	176.40	23,498,237	40.58	20.86	12,411,272	39.52	21.35	35,909,509	40.21	21.03	25,526,872	914,777,002	940,303,874	976,213,383	26.19
32	92.50	181.30	23,498,237	40.58	20.86	12,584,771	39.52	21.36	36,083,008	40.21	21.03	25,824,080	927,496,608	953,320,688	989,403,696	26.42
33	95.00	186.20	23,521,887	40.58	20.86	13,024,223	39.52	21.38	36,546,110	40.20	21.04	26,530,975	961,112,218	987,643,193	1,024,189,303	27.02
34	97.50	191.10	23,535,787	40.58	20.86	13,239,573	39.53	21.39	36,775,360	40.20	21.05	26,862,919	978,436,039	1,005,298,958	1,042,074,318	27.34
35	100.00	196.00	23,568,237	40.57	20.86	13,795,901	39.51	21.41	37,364,138	40.18	21.06	27,870,519	1,023,839,644	1,051,710,163	1,089,074,301	28.15
36	102.50	200.90	23,568,237	40.57	20.86	13,847,251	39.51	21.41	37,415,488	40.18	21.07	27,939,316	1,028,142,036	1,056,081,352	1,093,496,840	28.23
37	105.00	205.80	23,568,237	40.57	20.86	13,991,651	39.52	21.42	37,559,888	40.18	21.07	28,214,094	1,040,306,127	1,068,520,221	1,106,080,109	28.45
38	107.50	210.70	23,574,287	40.57	20.86	14,130,701	39.51	21.42	37,704,988	40.17	21.07	28,520,907	1,052,827,873	1,081,348,780	1,119,053,768	28.68
39	110.00	215.60	23,574,287	40.57	20.86	14,145,551	39.51	21.42	37,719,838	40.17	21.07	28,541,934	1,054,006,858	1,082,548,792	1,120,268,630	28.70
40	112.50	220.50	23,574,287	40.57	20.86	14,145,551	39.51	21.42	37,719,838	40.17	21.07	28,541,934	1,054,006,858	1,082,548,792	1,120,268,630	28.70
41	115.00	225.40	23,574,287	40.57	20.86	14,145,551	39.51	21.42	37,719,838	40.17	21.07	28,541,934	1,054,006,858	1,082,548,792	1,120,268,630	28.70
42	117.50	230.30	23,574,287	40.57	20.86	14,153,351	39.51	21.42	37,727,638	40.17	21.07	28,551,293	1,054,793,822	1,083,345,115	1,121,072,753	28.71
43	120.00	235.20	23,574,287	40.57	20.86	14,153,351	39.51	21.42	37,727,638	40.17	21.07	28,551,293	1,054,793,822	1,083,345,115	1,121,072,753	28.71
44	122.50	240.10	23,574,287	40.57	20.86	14,175,551	39.51	21.42	37,749,838	40.17	21.07	28,585,555	1,056,887,331	1,085,472,886	1,123,222,724	28.75
45	125.00	245.00	23,574,287	40.57	20.86	14,296,951	39.52	21.43	37,871,238	40.17	21.08	28,807,326	1,068,013,005	1,096,820,331	1,134,691,569	28.96

Table 18.1-2. Mine Design Parameters.

Design Parameter	
Bench Height	10 metres
Bench Face Angle (Batter Angle)	76 degrees
Inter-ramp Angle	50 degrees
Minimum Berm Width	5 metres
Overall Slope Angle	45 degrees
Double Lane (Haulage Ramp)	36 metres
Single Lane (Haulage Ramp)	26 metres
Haulage Ramp Gradient	10 percent

The stripping ratio of approximately 26.6:1 (after dilution) will allow over the life of the mine use of a mixed truck haulage fleet. As the stripping ratios jump in later years of production 327 tonne haul trucks will be used and ramp widths for the life of mine will be built to accommodate this size of truck. Open pit haul roads of 36 metres width allowing for 2 way traffic were used in the majority of the open pit. One-way traffic haul roads of 26 metres were used toward the open pit bottom.

The inclusion of haul roads and creation of a practical pit design when compared with open pit optimization results, shows a 2.9% increase in stripping ratio, 0.5% decrease in waste generation, and 3.2% decrease in feed tonnage, if the quantity of mineralized resource defined in the open pit optimization is to be targeted. Table 18.1-3 shows the designed open pit results in comparison to the base case economic open pit.

The ultimate open pit has dimensions of 1,670 metres long by 1,250 metres wide by 570 metres depth. Figure 18.1-4 shows a rendering of the Big Daddy deposit and the open pit portion outline created using MineSight.

18.1.5 Mine Production Schedule

The open pit design was divided into benches. The open pit was then scheduled by bench phases.

The potentially mineable open pit lump chromite resource would be mined at a rate of 8,000 tonnes per day or 2.9 million tonnes per year, of lump chromite mineralization. Open pit operations would be carried out on two 12 hour shifts, seven days per week, for 360 working days per year.

Table 18.1-4 shows a summary of the production schedule. The open pit would be developed in three phases as shown in Table 18.1-5. The quantities of materials scheduled to be mined in each year are shown in Table 18.1-6. Table 18.1-6 indicates that:

- The overburden would be stripped from the Phase 1 open pit area and from a section of the Phase 2 open pit area in the pre-production period (years -2 and -1).

- Phases 1 and 2 would be concurrently mined in years 1 to 6, at a nominal rate of 2.9 Mtpy of lump chromite mineralization.

- Phases 2 and 3 would be concurrently mined in years 7 to 10, at a nominal rate of 2.9 Mtpy of lump chromite mineralization in years 7 to 9. In year 10, the production rate of lump chromite mineralization would decrease to approximately 1.2 Mtpy as a result of the increased waste stripping requirements.

- Phase 3 mining would occur in years 11 to 16. Major waste rock stripping efforts in years 11 and 12 reduce lump chromite mineralization production to 304 kt and 783 kt, respectively. Phase 3 is scheduled to produce approximately 2.8 Mtpy of lump chromite mineralization in years 13 to 16.

Table 18.1-3. Summary of Resource within Pit Design (Undiluted Comparison).

	Ind. (mt)	Cr2O3 (%)	Inf. (mt)	Cr2O3 (%)	Total (mt)	Cr2O3 (%)	Ovb. (mt)	Waste (mt)	Total Waste (mt)	Total (mt)
Pit 34	23.56	40.57	13.8	39.51	37.4	40.18	27.9	1,023.8	1,051.7	1,089.1
Design	23.54	40.58	12.6	39.52	36.2	40.21	26.7	1,020.0	1,046.7	1,082.9

Figure 18.1-4. Open Pit 3D Rendering.

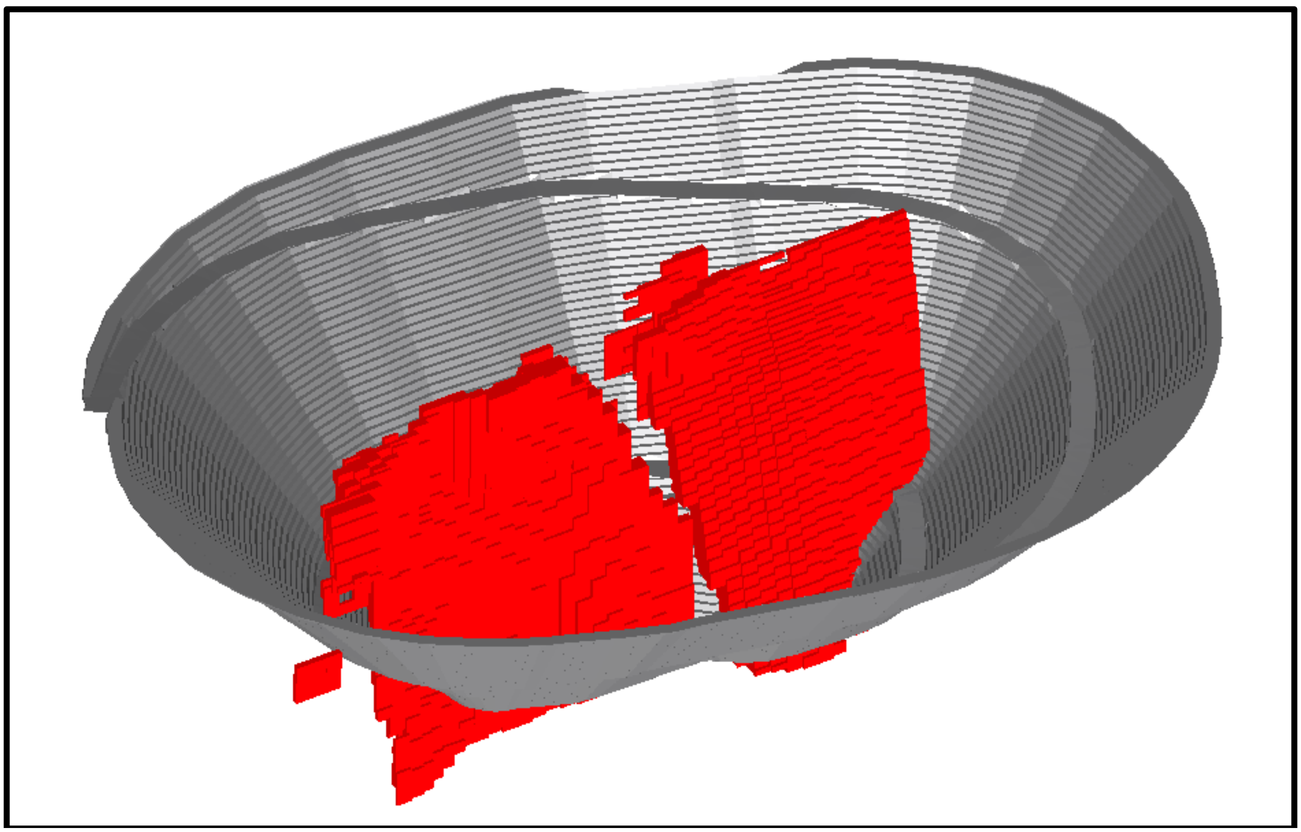


Table 18.1-4: Production Schedule Summary

Year	Direct (Tonnes)	Cr2O3 (%)	Fe2O3 (%)	Stockpile (Tonnes)	Cr2O3 (%)	Fe2O3 (%)	OVB (Tonnes)	Rock (Tonnes)	Total Waste (Tonnes)	Total (Tonnes)
-2							5,032,000		5,032,000	5,032,000
-1							3,356,256		3,356,256	3,356,256
1	2,120,897	36.97	19.94	912,222	28.07	17.90	6,234,103	50,877,641	57,111,744	60,144,863
2	2,698,443	37.37	19.82	1,255,715	27.79	17.77	0	65,500,000	65,500,000	69,454,158
3	2,880,000	37.67	19.36	847,350	28.09	17.61	0	65,500,000	65,500,000	69,227,350
4	2,880,000	37.94	18.87	696,547	28.11	17.35	0	65,500,000	65,500,000	69,076,547
5	2,880,000	38.12	18.88	592,681	29.32	17.65	3,463,084	62,036,916	65,500,000	68,972,681
6	2,880,000	37.73	18.49	556,126	29.21	17.13	8,649,768	72,850,232	81,500,000	84,936,126
7	2,880,000	38.42	19.67	541,827	26.95	17.11	0	81,500,000	81,500,000	84,921,827
8	2,880,000	38.55	19.77	319,213	27.98	17.70	0	81,500,000	81,500,000	84,699,213
9	2,880,000	38.19	20.03	216,140	29.45	17.02	0	81,500,000	81,500,000	84,596,140
10	1,262,611	38.28	20.39	248,729	28.12	14.10	0	81,500,000	81,500,000	83,011,341
11	304,842	35.75	17.98	159,241	25.49	17.20	0	81,500,000	81,500,000	81,964,083
12	783,231	36.69	20.38	184,352	22.17	15.85	0	81,500,000	81,500,000	82,467,583
13	2,880,000	37.28	20.35	416,510	30.68	18.91	0	81,500,000	81,500,000	84,796,510
14	2,880,000	37.48	20.31	258,177	30.16	19.33	0	31,193,501	31,193,501	34,331,678
15	2,880,000	37.30	20.31	260,238	29.91	18.66	0	18,185,506	18,185,506	21,325,744
16	2,931,675	36.91	20.27	107,272	29.31	19.56	0	7,172,266	7,172,266	10,211,213
Total	38,901,700	37.69	19.72	7,572,340	28.27	17.59	26,735,211	1,009,316,062	1,036,051,273	1,082,525,312

Table 18.1-5. Open Pit Production by Phase.

Phase	Overburden[1] (kt)	Direct Shipping Mineralization[2] (kt)	Stockpiled Low Grade Mineralization[3] (kt)	Waste Rock (kt)	Waste:Ore Stripping Ratio[4]
1	8,388	14,209	3,851	148,463	11:1
2	6,234	11,893	1,905	252,371	22:1
3	12,112	12,798	1,815	608,481	49:1
Total[5]	**26,735**	**38,901**	**7,572**	**1,009,316**	**27:1**

1 The overburden in the Phase 1 open pit area, and a portion of the overburden in the Phase 2 open pit area will be stripped during the preproduction period.

2 The scheduled direct shipping mineralization includes inferred resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by technical, environmental, permitting, legal, taxation, political, marketing or other issues. The quantity and grade of the reported inferred resources are uncertain in nature and there has been insufficient exploration to define these inferred resources as an indicated or measured mineral resource and it is uncertain if further exploration will result in them being upgraded to an indicated or measured mineral resource category.

3 It is assumed that low grade mineralization (>3.83% Cr_2O_3 and <35% Cr_2O_3) would be stockpiled. The current PEA does not include the reclaim and processing of the low grade stockpile.

4 Stripping ratio based on tonnes of overburden, stockpiled low grade material, and waste rock : tonnes of lump chromite mineralization.

5 Totals may not sum exactly due to rounding.

Table 18.1-6: Open Pit Phases Production Schedule

Year	Phase 1				Phase 2				Phase 3				Phases 1-3
	OVB[1] (kt)	Direct Shipping Mineralization (kt)	Stockpiled Low Grade Mineralization (kt)	Waste Rock (kt)	OVB[1] (kt)	Direct Shipping Mineralization (kt)	Stockpiled Low Grade Mineralization (kt)	Waste Rock (kt)	OVB[1] (kt)	Direct Shipping Mineralization (kt)	Stockpiled Low Grade Mineralization (kt)	Waste Rock (kt)	Total Direct Shipping Mineralization (kt)
-2	5,032												
-1	3,356				1,870								
1		2,097	902	37,544	3,117	23	9	13,332					2,120
2		2,653	1,230	47,745	1,247	44	24	17,754					2,698
3		2,793	756	35,578		86	90	29,921					2,880
4		2,773	575	16,857		106	121	48,642	6,056				2,880
5		2,393	314	9,601		486	278	52,434	6,056				2,880
6		1,496	71	1,134		1,383	484	38,781				32,934	2,880
7						2,878	538	27,812		1.7	2	53,687	2,880
8						2,849	265	14,542		30	53	66,957	2,880
9						2,843	89	8,678		36	126	72,821	2,880
10						1,192	2	470		70	246	81,029	1,262
11										304	159	81,500	304
12										783	184	81,500	783
13										2,880	416	81,500	2,880
14										2,880	258	31,193	2,880
15										2,880	260	18,185	2,880
16										2,931	107	7,172	2,931
Total[2]	**8,388**	**14,209**	**3,851**	**148,463**	**6,234**	**11,893**	**1,905**	**252,371**	**12,112**	**12,798**	**1,815**	**608,481**	**38,901**

[1.] Overburden (OVB) stripped by contractor.
[2] Totals may not add exactly due to rounding.

Detailed open pit plans were not developed as part of the present PEA. As indicated above, the rate of potentially economic lump chromite mineralization production decreases due to the substantial amount of waste stripping required during the transition to Phase 3 in years 10 through 12. From a mining and cash flow perspective there is a possibility that the low grade stockpile could be used to supplement production during this period. Again, no value has been given to the low grade stockpile in the present PEA.

18.1.6 Waste Stockpile

Low grade, overburden and single waste rock stockpiles would be located around the north, south, and west rim of the open pit. The location and extent of the dumps can be seen in the general layout (Figure 18.1-1).

A swell factor of 30% was assumed for the stockpiled material. The overall stockpile angles used were 37°, 22°, and 29° for the waste rock, overburden, and low grade stockpiles, respectively. The overburden and low grade stockpiles were limited to a maximum height of 10 metres. Required stockpile volumes based on the design are 371,100,000, 17,377,900, and 3,290,000 cubic metres (of loose material) for the waste rock, overburden, and low grade stockpiles respectively. The stockpile parameters used for each case are shown in Table 18.1-7.

Condemnation drilling should occur prior to mining in order to ensure there is no potentially mineable material underneath the proposed stockpiles.

18.1.7 Mine Operations

It is assumed that the open pit would be operated using the mine owner-operator's equipment and labour force, with the assistance of an overburden stripping contractor, mine equipment supplier maintenance personnel, and an explosive supplier.

<u>Overburden Stripping</u>

The overburden, surface muskeg and underlying soils, would be stripped by a contractor. The stripping work would involve initial clearing, ditching and drainage works; muskeg removal in winter months; and soil excavation to expose bedrock. The excavated overburden materials would be stored in designated overburden stockpile areas.

<u>Drilling and blasting</u>

The direct shipping mineralization, low grade material and waste rock would be drilled and blasted using conventional drilling and blasting equipment and technologies. Drilling would be done using conventional diesel-powered track-mounted drills equipped for cold

Table 18.1-7. Stockpiles Design Parameters.

Parameter	Value
Waste Rock Stockpile	
In-situ Density	2.72 tonnes / cubic metre
Lift Height	10 metres
Batter Angle	50 degrees
Berm Width	5 metres
Overall Angle	37 degrees
Double Haulage Lane	36 metres wide
Overburden Stockpile	
In-situ Density	2.00 tonnes / cubic metre
Lift Height	10 metres
Batter Angle	27 degrees
Berm Width	5 metres
Overall Angle	22 degrees
Double Haulage Lane	36 metres wide
Low Grade Stockpile	
In-situ Density	3.0 tonnes / cubic metre
Lift Height	10 metres
Batter Angle	37 degrees
Berm Width	5 metres
Overall Angle	29 degrees

weather operations. The explosives and blasting accessories would be supplied by an explosive supplier. The PEA is based on the use of nominal 152 mm (6 inch) diameter blastholes, 10 m high benches, and bulk emulsion explosive. The projected powder factors for an assumed 4.7m x 4.7 m pattern in lump chromite mineralization and low grade material, and an assumed 5.5m x 5.5 m pattern in waste rock are 0.25 kg/t and 0.24kg/t, respectively.

Loading and haulage equipment

The loading and hauling equipment requirements were estimated taking the production schedule, projected haulage profiles and cycle times (Table 18.1-8), selective and bulk excavation requirements, effective working hours per shift, expected field conditions and other relevant aspects into consideration. The PEA is based on the use of a mixed haulage truck fleet:

- 144 t capacity haulage trucks would be used to haul lump chromite mineralization, low grade material, and waste rock from the Phase 1 open pit, and lump chromite mineralization and low grade material from Phases 2 and 3.

- 327 t capacity haulage trucks would be used to haul waste rock from Phases 2 and 3.

144 t Capacity Haul Trucks

The Phase 1 lump chromite mineralization, low grade material, and waste rock would be excavated using 18 m^3 bucket capacity diesel-powered hydraulic shovels and 144 t capacity haulage trucks. The haul truck operational parameters are shown in Table 18.1-9. This equipment would also be used to mine lump chromite mineralization and low grade mineralization in Phases 2 and 3. The lump chromite mineralization would be hauled to the crusher. The low grade material would be hauled to the low grade stockpile.

The PEA is based on the use of Komatsu PC4000 (18 m^3) type shovels and Komatsu HD1500 (144 t capacity) type haulage trucks. It is expected that the mine would obtain quotes from several established suppliers as part of its equipment selection process.

327 t Capacity Haul Trucks

The main waste rock stripping fleet includes 60 m^3 capacity electric cable shovels and 327 t capacity haulage trucks. This equipment would be used to strip waste rock in Phases 2 and 3. The waste rock would be hauled to a designated waste stockpile.

The PEA is based on the use of P&H 4100 (60 m^3) type electric-powered cable shovels and Komatsu 960E (327 t capacity) type haulage trucks. It is expected that the mine would obtain quotes from several established suppliers as part of its equipment selection process.

Table 18.1-8. Estimated Haul Truck Cycle Times.

Year	Estimated Haulage Truck Cycle Times				
	Phase 1	Phase 2		Phase 3	
	144 t trucks[1] (minutes/trip)	144 t trucks[1] (minutes/trip)	327 t trucks[2] (minutes/trip)	144 t trucks[1] (minutes/trip)	327 t trucks[2] (minutes/trip)
1	14 – 17	14 - 17	14		
2	16 – 19	16 - 20	14		
3	18 – 22	19 - 22	16		
4	21 – 24	21 - 25	18		
5	25 – 29	25 - 29	20		
6	27 – 31	28 - 33	22	17-21	17
7	30 – 34	31 - 35	24	18-22	17
8		34 - 38	25	20-24	19
9		37 - 41	27	23-27	20
10		40 - 44	29	26-30	22
11				34	25
12				35-39	27
13				40-44	30
14				45-49	32
15				48-52	34
16				52-56	36

1 The 144 t capacity haul trucks would be used to haul direct shipping mineralization to the crusher, and low grade material to the low grade stockpile. The table presents the estimated cycle time range.

2 The 327 t capacity trucks would be used to haul waste rock to the rock stockpile.

The estimated cycle times for the haulage trucks are summarized in Table 18.1-8. Selected haulage truck parameters used to estimate the truck fleet requirements are summarized in Table 18.1-9.

The numbers of trucks and shovels were estimated based on the projected truck and shovel operating hours and projected equipment availabilities. The numbers of shovels and trucks required in each year is summarized in Table 18.1-10. Detailed open pit plans were not developed as part of the present PEA and as such there is a possibility that the pit equipment fleet requirements will be improved upon and optimized in subsequent technical studies for the Project.

Ancillary Equipment

The mine operations would be supported by a wheel loader, bulldozers, a wheel dozer, road graders, water trucks, a cable reeler, ditching excavator, pick-ups and field service vehicles including a mobile crane, a fuel and lube truck, a truck tire manipulator, mechanical/electrical field service trucks, and portable lighting stands.

Mine Support Facilities

The support facilities would include a maintenance shop complex, warehouse, fuel and lubricant storage facilities, a refuelling station, waste management facilities, maintenance offices and a back-up generator. The mine would also have a dry and open pit operations and technical offices (described in Section 18.3 of this report). The mine would utilize an automated truck monitoring and dispatch system.

Maintenance Shop

The mine maintenance shop would be responsible for maintaining all mining equipment and light vehicles. The maintenance shop building would consist of 4 service bays, offices, lunchroom and storage areas for tools and parts. One of the bays would be a wash bay. Two bays would generally be used for haul trucks maintenance and one for other support equipment servicing. A small light vehicle service area would also be provided. The building would be prefabricated from steel structural framing and metal cladding, with concrete floors.

Table 18.1-9. Selected Haul Truck Parameters.

Parameter	144 t capacity haul truck	327 t capacity haul truck
Loading shovel capacity	18-22 m^3	60 m^3
No. of passes to load truck	3-4	3-4
Truck manoeuvre / load time	205 seconds	210 seconds
Truck manoeuvre / dump time	100 seconds	140 seconds
Truck payload capacity	144 t	327 t
Estimated average payload	140 t	314 t
Effective working hours[1]	16.4 hours/day[1]	16.4 hours/day[1]

[1] After estimated fixed and variable delays.

Explosives Magazines

The secure explosives storage area for the mine would be located 1.5 kilometres from mining, processing and other facilities.

It is assumed that an electrical power line (described in Section 18.3) will be constructed to the mine site during the preproduction years -2 and -1. The mine electrical power distribution system would service the electric shovels and mine dewatering pumps. It is assumed that the mine dewatering pump power would increase from 150 kW (200 hp) initially to 750 kW (1,000 hp) in year 16.

18.1.8 Labour Force

The number of personnel on the mine payroll, in production years 1 to 16, is presented in Table 18.1-11. Table 18.1-11 excludes mine equipment maintenance personnel to be provided by equipment suppliers under a mine equipment, maintenance and repair contract (MARC).

Table 18.1-10. Estimated numbers of shovels and haulage trucks.

Year	Estimated numbers of haulage trucks and shovels required in each year.			
	144 t haulage truck fleet		327 t haulage truck fleet	
	Excavator (Komatsu PC4000)	Haul Truck (Komatsu HD1500)	Cable shovel (P&H 4100XPC)	Haul Truck (Komatsu 960E)
1	4	17		
2	4	19	1	3
3	3	18	2	5
4	2	11	2	9
5	1	9	2	11
6	1	4	3	15
7	1	4	3	17
8	1	4	3	18
9	1	4	3	20
10	1	2	3	22
11	1	1	3	24
12	1	1	3	26
13	1	4	3	29
14	1	4	2	12
15	1	4	1	7
16	1	5	1	3

Table 18.1-11. Mine Manpower Complement by Year.

Open Pit Operations Labour		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Pit supervision:	Pit Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	Pit Shift Supervisor	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3
	Pit clerk	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Drilling and blasting:	Blasting engineer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Drillers & drill mechanics	6	8	8	8	8	10	10	10	10	10	10	10	10	4	4	4
	Explosive supply / loading	10	10	10	10	10	12	12	12	12	12	12	12	12	8	8	8
Loading equipment:	PC4000 excavator operator	16	14	12	6	4	4	2	2	2	0	0	2	0	0	0	0
	P&H shovel operator	0	4	4	8	8	12	12	12	12	12	12	12	12	4	4	4
	Wheel loader operator	0	1	1	1	1	1	1	1	0	0	0	0	0	4	0	0
Haulage trucks:	Komatsu HD1500 truck drivers	56	64	60	36	28	12	10	12	12	4	0	4	12	12	12	12
	Komatsu 960E truck drivers	0	10	20	36	46	60	64	68	74	84	92	104	122	52	32	16
Ancillary equipment operators:	Bulldozer, wheel dozer operators.	12	12	12	12	12	12	12	12	12	12	12	12	8	6	4	4
	Grader, service truck operators.	8	8	8	8	8	10	8	8	8	8	8	8	8	4	4	2
	Subtotal pit operations	**117**	**140**	**144**	**134**	**134**	**142**	**140**	**146**	**151**	**151**	**155**	**173**	**193**	**103**	**72**	**57**
Pit maintenance[1]:	Pit mtce superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	Lead Mechanic	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
	Mechanics	4	8	8	8	8	8	8	8	8	8	8	8	8	6	2	0
	Electrician	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
	Subtotal pit mtce	**13**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**15**	**11**	**7**
	Geologist	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Geological technicians	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
	Mining engineer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
	Engineering technicans	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
	Clerk	0	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
	Subtotal tech services	**10**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**13**	**12**	**11**	**9**
	Total	**140**	**170**	**174**	**164**	**164**	**172**	**170**	**176**	**181**	**181**	**185**	**203**	**223**	**130**	**94**	**73**

[1] The number of personnel on the payroll excludes MARC (maintenance and repair contract) personnel.

Allowances were included for benefits and vacations in the wages (and 8.0% overtime for hourly employees).

18.2 PRODUCT PREPARATION

The product preparation plant will accept run-of-mine (ROM) lump chromite mineralization. Through a series of crushers and screens the material would be reduced in size to ensure all lump chromite mineralization is minus 50 mm (2") in size. The product preparation plant flowsheet is shown in Figure 18.2-1

The ROM lump chromite mineralization would be dumped from the haul trucks onto a grizzly and rockbreaker setup. This would remove oversize material and feed the primary gyratory crusher.

The crushing and screening facility will be constructed to utilize a combination of one gyratory and two (2) cone crushers to provide the final crushed product of minus 50mm. This also provides the mine with maximum flexibility, as both gyratory and cone crushers can be adjusted to increase or decrease flow or sizing as required.

The system would utilize a 42"x65" Superior II –type gyratory crusher, which is protected by a large opening scalping grizzly and rock breaker . The grizzly would have 762 mm x762 mm openings to eliminate oversize material and prevent plugging of the gyratory crusher. Material would be crushed to minus 152mm and then conveyed to an initial screen for separation.

The first screen would be set to 75mm passing, with any material smaller than 75mm falling through the grate for conveying to a second screen. Material that passes over the grate would be fed into the first cone crusher, for sizing to minus 75mm.

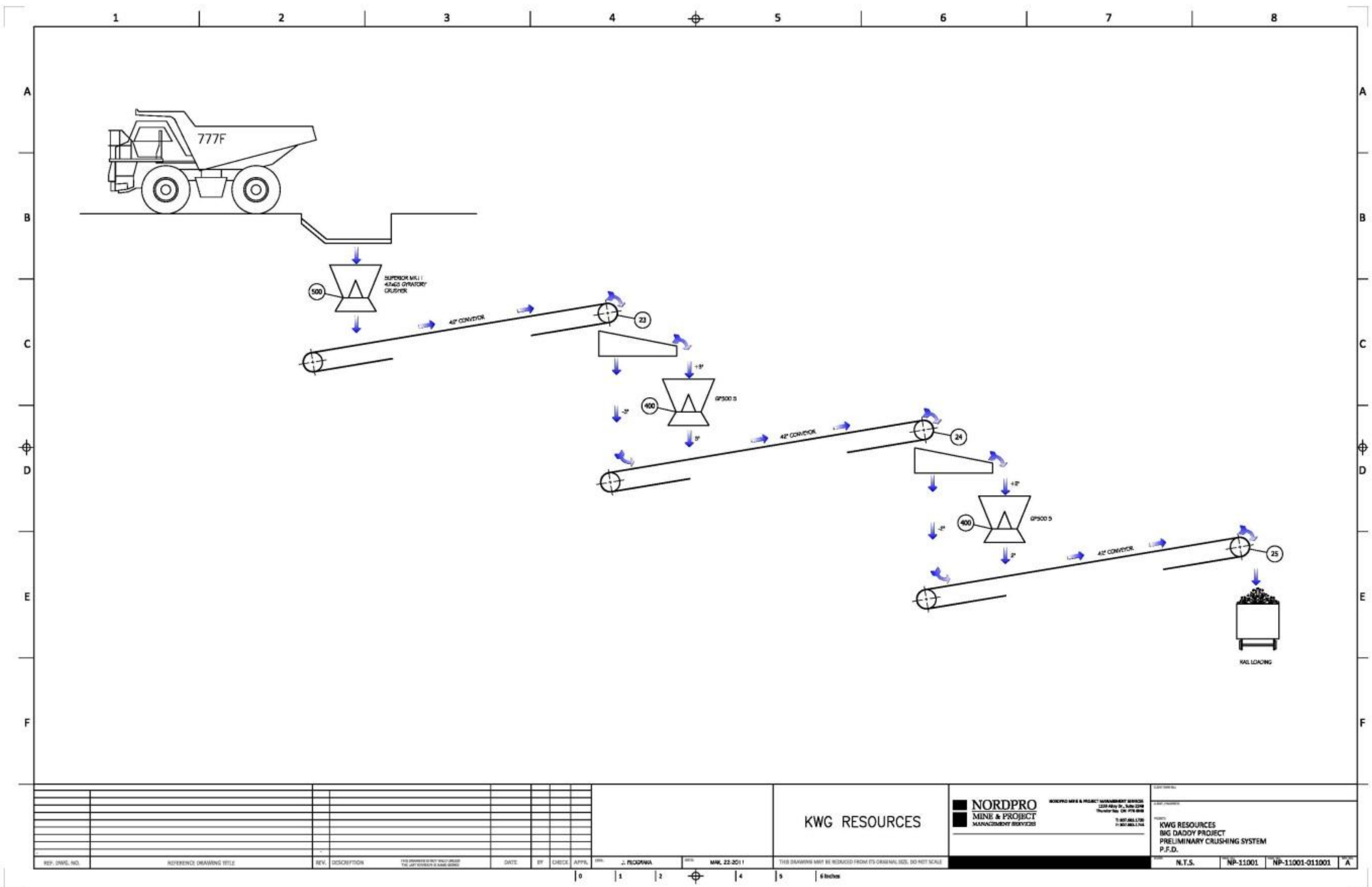
The 75mm material would then be conveyed to a second screen, with 50mm passing openings. Material smaller than 50 mm would pass through, while anything larger would pass over the screen and into a second cone crusher.

The second cone crusher would reduce the material to 50mm minus (or smaller if required). From here, the under size from the second screen and the material crushed through the second crusher would be sent via conveyor to a train loadout facility for loading into rail cars and shipping to customers.

All Conveyors would be from 914mm to 1067mm wide, using CEMA "E" class idlers (for durability and long life), and would use fluid couplings as part of their drives. All drives would be electric.

If it is found during operations that the material has a tendency to "shard", ie, crush into pieces that are much longer in one axis than the other two (3:1 or greater), it may be desirable to add a third screen to recirculate material back to the second crusher. This has not been included but can be added if necessary during initial operations.

Figure 18.2-1. Product Preparation Plant Flowsheet.



The overall horsepower in this system is 900 HP in crushing and approximately 300 HP in conveying, with less than 50 HP in screening. The system can be re-set to operate at as much as 800 tonnes per hour, if desired. In this way, the mine has expansion capacity, and can run at less than 80% load, thus reducing wear and tear on the system.

Manpower required to operate the product preparation plant would be 6 people per crew for a total of 24 people.

18.3 INFRASTRUCTURE

The Big Daddy Project location, not close to any major population centres, would require full service infrastructure and support facilities, including manpower accommodation and recreational facilities.

Because of the large product tonnage requiring transport from the mine to customers a rail line would be constructed from the property to connect to the main Canadian National Railway Company (CNR) rail line near Nakina.

An airstrip servicing the Ring of Fire area will be constructed in the near future and service all projects in the area. The air strip will be shared and it is assumed that the construction and related costs will not be incurred by the Big Daddy Project. Usage fees would be paid by the project and are included in the operating costs.

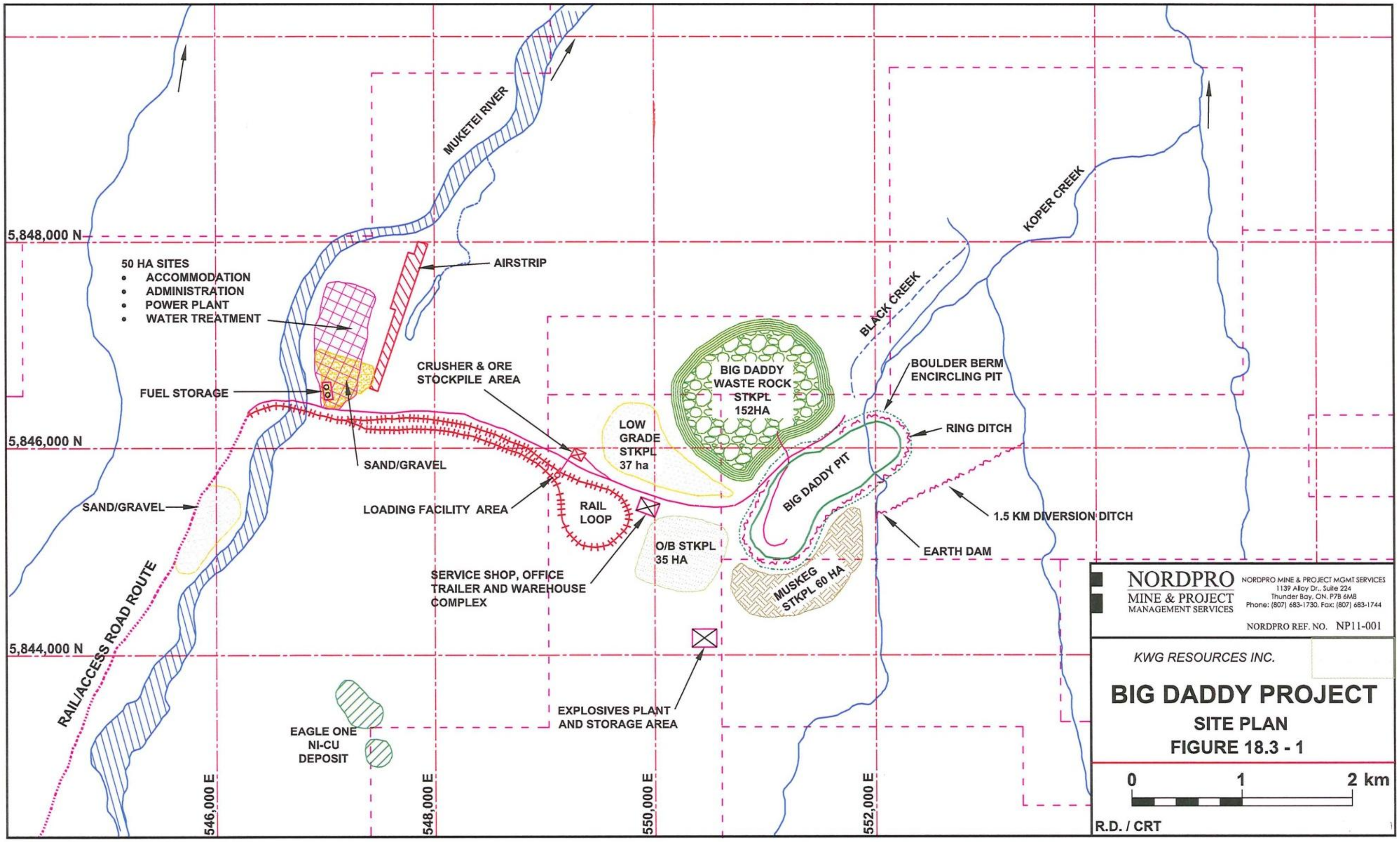
A permanent road to site would not be constructed as the railway would be capable of moving all materials and equipment to site.

The main site infrastructure requirements for the mine would be:

- Site Roads
- Haul Roads
- Explosives Magazines
- Mine Maintenance Shop
- Warehouse and Laydown Yard
- Services/Technical/Administration Office Building
- Camp and Recreational Facilities
- Electrical Substations and Distribution
- Water Supply System and Water Treatment Plant
- Landfill Site
- Sewage Disposal Site

A site plan showing all infrastructure is presented in Figure 18.3-1.

Figure 18.3-1. Big Daddy Site Plan.



18.3.1 Railway Line

A map showing the proposed railway line is presented in Figure 18.3-2.

Canada Chrome Corporation is considering construction of a railway line to serve proposed chromite projects, including the Big Daddy, in the "Ring of Fire" area. A railway route has been determined that would extend northwards from the CN transcontinental rail line near Nakina, Ontario to the district, a distance of 350 kilometres. Proposed most feasible routes for the railway line have been staked with claims to tie up the accesses and rights of way. Under mining regulations any use of the claims for a railway line is allowed, with compensation paid to the owners of the land.

Unit trains transporting lump chromite mineralization and backhauling materials and equipment, would operate on the rail line from the mine to Nakina, for interchange with CN. The railway would be operated by a railway operator, which could be either a private carrier or Owner managed.

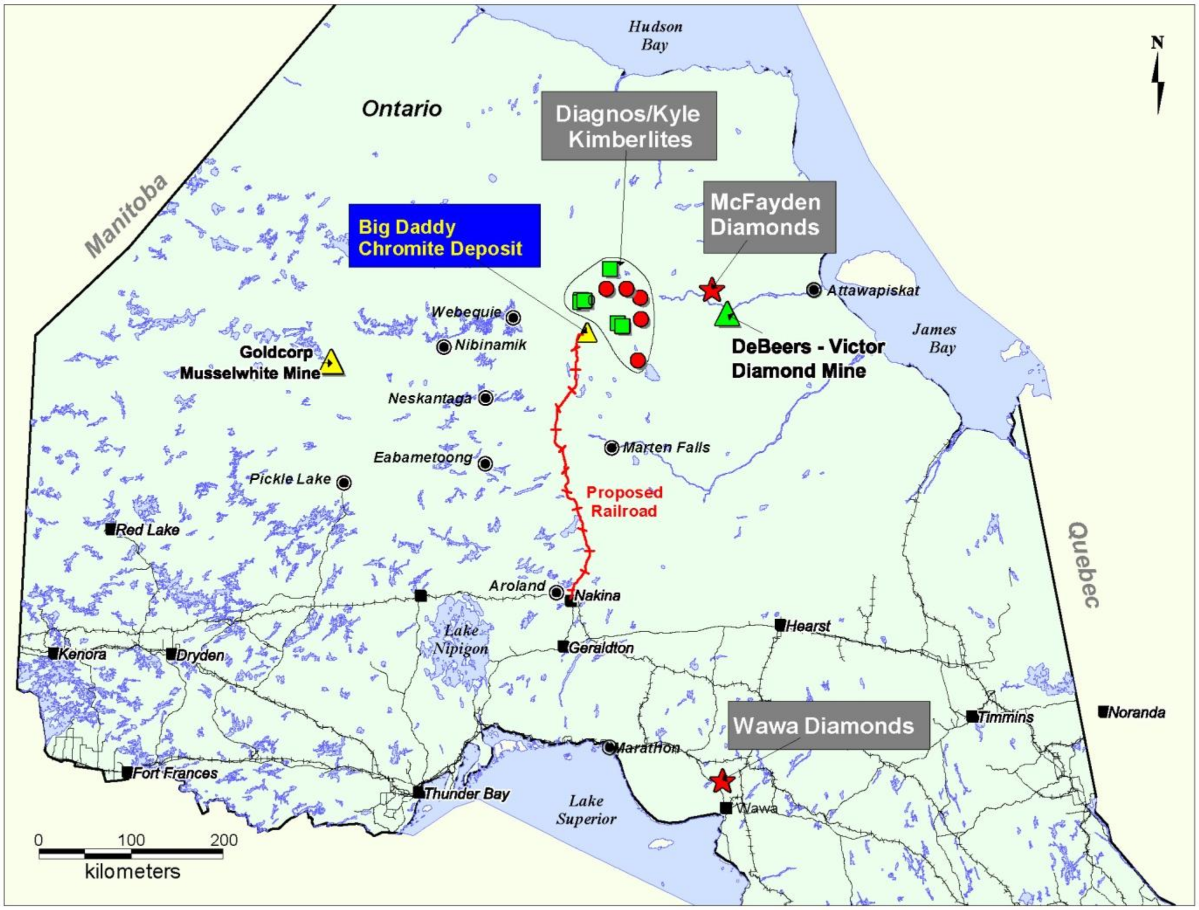
For purposes of the PEA it has been assumed that the railway would service the Big Daddy and the other projects in the area. Potential future capacity growth has been included in the designs to accommodate traffic where 3 or more mines are in production.

<u>Railroad Construction</u>

The geology of North western Ontario through which the railroad traverses consists of three main regions. The first 81 kilometres goes through the Canadian Shield consisting of exposed rock ridges that generally run in a north/south direction. The next 40 km is a transition zone of weaker glacial deposits that is a mixture of silts and aggregates. The final 282 km to the mine site is low lands essentially consisting of string bogs. There are 89 stream and river crossings along the length of the route. Fortunately a narrow band of acceptable soils left behind by a glacial deposit was found that essentially run from the edge of the Canadian Shield directly to the mine site. The proposed railroad site will utilize this sand ridge for the majority of the northern portion of the route.

The railroad will run along the edge of the rock ridges until it intersects with the glacial sand ridge which it will follow to the mine site. Aggregate and rock sources for the embankment and track construction have been identified along the route. Fifty-four bridges, some over 600-metres in length, will be constructed. There is a 150-metre difference in elevation between Nakina and the mine site with the maximum percent grade of 0.8%.

Figure 18.3-2. Proposed Railway Line Routing.



Because of the remoteness and poor soils surrounding the rail route all materials, supplies and construction camps will be staged during the winter using ice roads. Two years is expected for construction of the railroad.

Railroad Operations

Three basic scenarios exist for the operation of this railway:

1. Owner operated.
2. Contract operations and maintenance to a short line operator.
3. Contract operations and maintenance to CN.

In all cases equipment and rolling stock requirements and costs would be essentially the same. Dedicated trains, called unit trains, consisting of locomotives and one hundred - 100 ton capacity cars with a capacity of 8,000 to 10,000 tonnes would be used to transport the lump chromite mineralization. This translates into a loaded lump chromite mineralization train being dispatched every 24 hours. At an average speed of 48 kilometres per hour it would take approximately 7 to 8 hours to travel to the mine site, 4 hours to load the train and 7 to 8 hours to return to Nakina. Crews would interchange between loaded and unloaded trains at the mine site. At Nakina all lump chromite mineralization loaded cars would be interchanged to CN. A two track siding would be constructed at Nakina to accommodate the interchange between loaded and unloaded trains. If CN operates the mining railroad locomotive power would probably remain with the unit train. If a private operator operates this line it is likely that the locomotives would be changed between the loaded and unloaded trains.

It is possible that lump chromite mineralization would be shipped to Asia through CN's trans-shipment facility at Prince Rupert, British Columbia or through Sept Iles, Quebec. The CN would pick up a loaded train at the interchange point near Nakina and utilize their Trans-Canada route to deliver lump chromite mineralization to Prince Rupert or Sept Iles. A two week turn around for the unit trains could be expected.

If the mine railway is run by the owner or an operator other than CN, locomotive power would be required. If CN operates the railway it would leave the same locomotives on the unit trains for delivering the trains to the mine site after changing crews in Nakina.

Railroad Maintenance

Facilities would be required to accommodate and feed operations, locomotive and car maintenance and track equipment and maintenance.

Manpower

To operate the railway will require approximately 10 personnel including locomotive crews and supervision.

Due to its remoteness approximately 25 maintenance personnel will be required to maintain the railway.

18.3.2 Electrical Power Supply

Electrical power for the mining/product preparation/support facilities will be supplied by a powerline run along the railway corridor to site from Nakina.

Total site power consumption will breakdown approximately as follows:

Mine	5.0 mW
Processing Plant	1.3 mW
Surface Facilities	1.0 mW
Total	**7.3 mW**

18.3.3 Roads

Roads on site would be required to connect the mine, product preparation plant, support services facilities and camp/recreational facilities together and follow the water supply pipeline. A total site roads distance of 8 kilometres was assumed for this study based on site conditions.

A total haul roads distance of 3 kilometres connecting the open pit to the product preparation plant and waste stockpile is also included in the capital expenditures.

18.3.4 Support Facilities

The support services facilities would be located in proximity to the processing plant complex and open pit operation. The support services site would include the services/technical/administration office building and warehouse/laydown yard.

Services/Technical/Administration Building

The services/administration building would provide office and work space for the mine supervision, geology and engineering, support staff, administration and purchasing/accounting personnel.

The building would have a central open area, with partitioned office space, for engineering and geology personnel. This open area would have individual offices surrounding it for senior mine management, engineering, geology, and administration personnel, as well as a lunchroom, conference room and washrooms. A separate area for mine supervision offices and crew line up area would also be included. A network room would house the mines computer LAN and telephone communications systems. The building would be a prefabricated structure with steel structural framing and metal cladding. Tiled flooring on a concrete slab would constitute the flooring.

Work areas would be equipped with desks, filing cabinets, bookcases, computers and telephones. A separate area for photocopier, fax machine printers and plotter would be provided as well. All work areas would be heated and air conditioned.

Warehouse/Laydown Yard

The warehouse facility, connected to the mine maintenance shop, would have areas for pallet shelving storage of materials and parts, a lockup area for supplies and office space for purchasing and warehousing personnel. A laydown yard for large material and equipment which could be stored outdoors would be provided next to the warehouse building and a cold storage building to house large materials equipment which require cover. The warehouse building would be a prefabricated structure with steel structural framing and metal cladding, with concrete floors.

18.3.5 Camp and Recreational Facilities

The camp would consist of accommodation, catering and recreational facilities to house 300 people.

The accommodation would consist of a 2 storey pre-fabricated building with central corridors on each floor. Rooms would be off each side of the central corridors. Hourly personnel accommodation would consist of 2 beds per room one for each shift with communal toilets and washing facilities for approximately every 10 rooms. Staff accommodation areas would consist of rooms off a central corridor with one bed per room and a bathroom shared by every 2 rooms. All rooms would be equipped with satellite television feeds and internet.

The accommodation quarters would be connected to the cooking and catering facility which would consist of:

Food preparation areas & kitchens
Non-perishable food storage,
Perishable food fridges and freezers
Food distribution areas
Cafeteria style eating area
Garbage disposal.

Other facilities related to the camp would be laundry facilities for bedding and separate facilities for regular workers non-working clothes.

The recreational facilities would consist of television rooms, games rooms, indoor gymnasium with social area and outdoor facilities for such sports as basketball, hockey, etc.

The camp would consist of pre-fabricated steel structures with insulated cladding. Units would be placed on appropriate foundations for long term use.

18.3.6 Water Supply

Service and potable water for the operation would be supplied from a nearby river or lake. An HDPE or PVC pipeline laid on the ground would transport water approximately 3-4 kilometres. Wells could also be used if a suitable river or lake source is not identified. A pump would also be required to pump the water over the required distance and elevation changes.

18.3.7 Water Treatment Plant

A water treatment plant has been included to treat water from the open pit and stockpile drainage.

18.3.8 Waste Disposal

Sewage generated at the operation would be treated in septic tank and filtration bed systems. Sewage would be collected in septic tanks and overflow water sent to a filtration bed for treatment and release. Separate septic and filtration bed systems would be constructed for the services site and camp. Septic tank contents would be periodically pumped into rail cars and transported to the nearest community with sewage treatment facilities for disposal by a contractor.

All non-toxic garbage from the operation would be placed in a small landfill site on the property, or in a central landfill site if developed for the area prior to project completion.

18.3.9 Telecommunications and Computer Networking

Telephone, data links and Internet services infrastructure for the operation would be provided via a fibre optic cable link to the nearest main services infrastructure, most likely in Nakina. The fibre connection would be laid in the railway corridor as part of the railway line and powerline constructions.

<u>Computer LAN's and Networking</u>

The corporate computer systems of the mine would be based on Microsoft.NET Enterprise Servers. Network and office software would be installed on the network servers and local computers.

A mid-tier accounting package capable of general ledger, accounts payable and receivable, purchasing and inventory and mine maintenance planning would be implemented at site. The telephone system, would also provide data and internet services to the mine. It would provide the mine with worldwide internet access and systems to allow for sending electronic data to head office and also facilitate worldwide data transfers as required.

18.4 PROJECT MANAGEMENT

The Big Daddy project construction would be managed by an EPCM consulting team and/or company. The project team would be responsible for managing and supervising project contractors and undertaking inspection, acceptance and commissioning of contractor work.

EPCM costs associated with the project have been included in the capital estimates.

18.5 GENERAL & ADMINISTRATIVE

General and administrative (G&A) costs are those primarily associated with the general management and administration of the project. G&A is associated with surface facilities and personnel not included under the mining, product preparation or maintenance groups and in addition to the surface department and railway transport group comprise of: administration; procurement; human resources; camp operations; and security.

18.5.1 Administration

Administration comprises senior and general management, accounting, third party environmental support and information technology functions. In addition to employee salaries and benefits, other components include employee relocation, travel expenses for business away from the property, insurance (property and business interruption), permits and licences, fees for mining rights, professional fees, and operating surface vehicles for the personnel.

Accounting functions include payroll, accounts payable, accounts receivable, budgeting, forecasting and other corporate cost accounting.

Information technology comprises all components associated with operating and maintaining the telephone, computer network, internet, fax and radio systems for the mine site. Allowances for long distance telephone charges are also included.

Environmental costs are associated with monitoring of the mine's environmental performance and reclamation work.

18.5.2 Procurement

Procurement encompasses all functions associated with on and off site procurement of materials and supplies; warehousing and inventorying; transportation from point of origin to site and other associated support services. Actual freight costs for items required by the mine, processing plant and maintenance departments are included in those department's costs.

The main cost components are comprised of employee salaries and benefits and warehouse supplies (such as personal protective equipment). Also included is small equipment (pallet lifters, forklifts, etc.) and parts used for warehousing, purchasing and logistics. Surface support includes loading and unloading of trailers and shipping containers, movement of materials on site and maintenance of the warehouse and associated facilities.

18.5.3 Human Resources

Human resources encompass all functions associated with personnel, union relations, health and safety, training and community relations. Personnel and industrial relations costs include salaries and benefits for employees to recruit required personnel, manage Company salary and benefits policies, manage hourly employees and oversee the Company's policies and procedures. Health and safety includes salaries, benefits, on-site first aid personnel, first aid supplies and vehicles required by this group.

Community relations costs include funds to aid in supporting local community efforts and facilities.

18.5.4 Camp

The camp would be operated by a contractor. Facilities maintenance would be performed by the mine surface department personnel.

18.5.5 Security

Mine site security is provided on a contract basis by a third party security firm. Security surveillance equipment will be provided to the security firm by the mine. Other minor security equipment for the security personnel (such as metal detectors, etc.) would be provided by the contractor.

18.5.6 Manpower

The G&A manpower required for the mine after commercial production starts is estimated to be 31 employees with the cost structure based on expected salaries paid in the Canadian mining industry with a premium included for the isolated location. The G&A manpower is presented in Table 18.5-1.

Table 18.5-1 General & Administration Manpower Complement.

Position	Quantity	Annual Salary ($)	Benefits 40%	Camp and Travel	TOTAL COST
General Manager	1	$200,000	$80,000	$30,000	**$310,000**
Secretary/Receptionist	2	$65,000	$26,000	$37,000	**$256,000**
Head H&S, Surface & Security	1	$90,000	$36,000	$30,000	**$156,000**
Service Truck Operator	2	$60,000	$24,000	$37,000	**$242,000**
Labourer	2	$55,000	$22,000	$37,000	**$228,000**
Comptroller	1	$90,000	$36,000	$30,000	**$156,000**
Accountant	2	$75,000	$30,000	$37,000	**$284,000**
Personnel Officer	1	$75,000	$30,000	$30,000	**$135,000**
Environmental Engineer	1	$120,000	$48,000	$30,000	**$198,000**
Environmental Technician	2	$75,000	$30,000	$37,000	**$284,000**
H&S Coordinator	1	$75,000	$30,000	$30,000	**$135,000**
Purchasing Agent	1	$85,000	$34,000	$30,000	**$149,000**
Warehouseman	2	$70,000	$28,000	$37,000	**$270,000**
Warehouse Stocktaker	2	$60,000	$24,000	$37,000	**$242,000**
Loader Operator	2	$60,000	$24,000	$37,000	**$242,000**
Nurse	2	$85,000	$34,000	$37,000	**$312,000**
Security Officers	6	$65,000	$26,000	$37,000	**$768,000**
TOTAL COMPLEMENT	**31**				**$4,367,000**

18.6 PRODUCT TRANSPORT TO CUSTOMERS

Lump chromite product would be transported by rail to a deep sea port on the east or west coast of Canada, for shipping to overseas customers. At the port lump chromite mineralization would be loaded onto ocean going vessels. Potential routes for shipping would have lump chromite mineralization loaded onto Panamax vessels for transport through the Panama Canal or larger Capesize vessels for other routes.

18.7 ENVIRONMENTAL & SOCIO-ECONOMIC

A series of baseline studies has been started and reports prepared.

Baseline monitoring activities and areas of study are summarized below:

- Routine surface water monitoring in the Muketei River system, by AECOM (Parks Environmental Inc.) since 2009.
- Cultural heritage values assessment at the Property by Ross Archaeological Research Associates, February, 2010.
- Preliminary Life Science Environmental Assessment by Northern Bioscience, April, 2010.

Based on the studies a closure plan has been completed.

18.7.1 Current Land Use

The area surrounding the Property has been used historically for wilderness, trapping and preliminary mineral exploration.

18.7.2 Surface Waters

The Property drains to the Muketei River via Koper Creek, a tributary of the Attawapiskat River. The proposed Mine site is situated within the Muketei River watershed. Limited large body surface features are situated within the Koper Creek watershed.

Surface water monitoring was initiated by consultants AECOM Technology Corporation, USA in July of 2009 and has been performed seasonally. Stations MR-D and MR-M have been used to baseline water quality in the Muketei River at the proposed effluent discharge point (Parks, 2010).

General surface water quality within the study area identifies that total iron, total aluminum and total magnesium exceed PWQO, and is typical for northern waters with a clay underlying the watercourse in sporadic areas. Cr^6 was tested for during 2009 (July, October) and 2010 (March) and was non-detectable, based on CAEAL accredited laboratory detection limits.

18.7.3 Assimilative Capacity Assessment

Water for use in the mine and the crushing circuit will be recycled from a settling pond. It is proposed that excess water from the Settling Pond be discharged to the Muketei River on a continual basis in accordance with effluent limits that are intended to maintain PWQO in the Muketei River during a worst-case hydrologic condition.

After ore chemistry has been detailed, an assimilative capacity study will be undertaken to confirm proposed effluent treatment processes will be sufficient in meeting provincial PWQO and federal MMER criteria.

An application for an Industrial Sewage Certificate of Approval will be submitted to MOE and this approval will specify effluent limits for the proposed effluent discharge point.

18.7.4 Groundwater

The shallow groundwater flow regime is anticipated to follow surface water flow directions.
A zone of influence from the mine is expected in bedrock and overburden until the Mine is flooded at Close-out. The water table elevation would be drawn down within this zone of influence, resulting in a net movement of groundwater toward the open pit mine workings. The zone of influence is predicted to include the Mine yard. As there are no known users of groundwater in the host watershed, no conflicts are expected from Mine dewatering. No springs or upwellings were identified, so no impacts to fish habitat are anticipated as a result of Mine dewatering. Measures to maintain habitat volume and prevent a harmful alteration of fish habitat will be confirmed with the Department of Fisheries and Oceans ("DFO").

18.7.5 Assessment of Potential Groundwater Quality Impacts

The zone of influence created by the dewatering of the Mine during operation would result in a net movement of local bedrock groundwater toward the open pit from the vicinity of the project site. As the handling of development rock and ore would be limited to the Mine yard, this zone of influence would prevent migration of bedrock groundwater off-site during Mine operation.

Controlled ore, fuel, waste and reagent handling practices are expected to avoid an impact to groundwater quality due to these ancillary activities.

Due to the low sulphide content and self-buffering nature, of the relatively high levels of neutralizing potential for the host ultramafic rocks, there is no anticipated potential for acid rock drainage or metal leaching.

18.7.6 Soils

At the Big Daddy property overlaying the bedrock is an overburden of deep, calcareous glaciomarine clay, silt, and sand (Barnett, 1992). Glacial features in the Hudson Bay Lowlands also include till deposits that are overlain by these glaciomarine sediments. These tills are deeper at inland sites compared to the coast, and their depth, origin, and composition determine available nutrients and water holding capacity of the substrate, particularly where glaciomarine clays are thin or absent (Riley, 2003). Scattered eskers, deltas and other subglacial deposits also occur in the Hudson Bay Lowland and have often been reworked by the postglacial Tyrell Sea (Riley, 2003).

18.7.7 Terrestrial Plant and Animal Life

The Big Daddy site and its environs provide relatively poor environments for wildlife because of the vast expanses of low productivity fen and bog habitats. The three habitat types in the region, which do provide important wildlife habitat, are rich riverbank forests, creek margin forests, and northern ribbed fens with broad pools.

Caribou of the Big Daddy site area most commonly occur as individuals and in small to medium sized groups, as opposed to the larger herds typical of barren ground caribou. While believed to be non-migratory, local caribou populations are known to move around extensively within the general area. As far as it is known, there are no known specific local migration routes, calving areas or wintering areas.

Wolves and black bear are the largest predators in the region. Local furbearers include beaver, muskrat, snowshoe hare, marten, mink, otter, red fox and lynx, with marten and beaver being the most economically important. Furbearers, like moose, tend to be concentrated along the watercourses, either because they are directly associated with water habitats (beaver, muskrat, mink, and otter) or because they prefer forest and forest/scrubland habitats which border the creeks and rivers (marten, lynx and fox).

Waterfowl and shorebirds occur in extremely large numbers nearer to the James Bay coast, especially during the spring and fall migration periods. Numbers decrease further inland, such as at the Big Daddy site. A variety of raptors (eagles, osprey, hawks, and owls) also occur in the area, the most notable of which are bald eagles and osprey. These two species feed mainly on fish and are associated with fluvial habitats. Numerous other bird species also occur in the region, with the majority tending to be associated with forests and scrublands bordering the creeks and rivers. A number of migratory bird species, such as sand hill cranes, various shorebird species, and limited numbers of waterfowl, utilizes some of the open muskeg areas, where ponds are plentiful.

18.7.8 Aquatic Plant and Animal Life

Fisheries and aquatic resources of the Big Daddy site and its environs are provided principally by stream systems, and by scattered, comparatively small and shallow lakes and ponds. Most of the abundant smaller ponds, due to their shallow depths typical of muskeg habitats, are expected to periodically freeze to bottom during severe winters.

Stream habitats in the general Project area are dominated by very large systems such as the Attawapiskat River and its tributaries, such as the Muketei River; and by smaller creek systems.

The majority of the watercourses exhibit cool summer temperatures (<24ºC). These temperatures are typical of cold water habitats, and support cool to cold water species.
There are no known active commercial fisheries in the Project area, although fishing is a valued activity by the local community of Webequie.

The Attawapiskat River, including its tributary, the Muketei River, is a very large river system. From the west to a point approximately 50 km upstream of the community of Attawapiskat it is predominantly bedrock controlled, with numerous areas of bedrock exposure along the riverbanks, together with several areas of rapids. From this point forward to James Bay, the character of the river changes to one of exposed clay and silt banks. Bedrock exposures through this area are very rare. Riverbed materials throughout the river system consist mainly of mixtures of gravel, cobble, and boulder. Larger fish species inhabiting the river system include walleye, pike, sturgeon, whitefish, suckers and burbot. Brook trout are also common in many of the feeder creeks.

18.7.9 Historic Activities and Mine Hazards

While there is no documentation of past human activity in the area an examination of area landforms and relevant site records indicates that five locations on the Muketei River and all eskers and moraines in the area have a high potential for archaeological resources (Ross, 2010).

The Property has been subject to only preliminary exploration since the late 1980s, as outlined in earlier sections of this report.

There are no known mine hazards within the Property.

18.7.10 Geochemistry

An analytical program was completed to define the geochemical characteristics of the rock that may be extracted either as ore, low grade ore, or mine rock at the Big Daddy site. Other than whole rock analyses there has yet to be any acid base accounting, US EPA 1312 extraction tests, process water analyses, mineralogical examinations and saturated column tests completed on mine rock, and ore (coarse and fine) samples.

Acid Base Accounting

The mine rock is composed primarily of serpentinized ultramafic, which has a high neutralizing potential due to its high matrix alkalinity (high Mg content). Of 1,485 samples analyzed 99.6% of the samples have sulphide sulphur contents of less than 0.6%. This low sulphide level indicates that the AP too is very low. The data set has a mean of 0.09% S with half of the samples having less than 0.06% S, the median. Such low sulphur indicates a low Acid Producing Potential (AP). A large number of samples representing the ultramafic-hosted ore (the samples were raw rock samples) were analyzed.

The results of the testing show that samples had consistently low sulphur concentrations and therefore a low AP. In addition the high Lost on Ignition (LOI) levels associated with a strong linear relationship with MgO suggest high magnesite content, a form of reactive carbonate mineral, and therefore a high Neutralization Potential (NP). As a result these rocks are considered to have negligible acid generation potential. Due to the high NP any seepage contacting the mine rock will be buffered in the neutral to slightly alkaline pH range.

Low grade ultramafic mineralization would also be extracted according to the current mine plan. The low grade samples had consistently low sulphur concentrations and consistently high NP and could therefore be classified as having a negligible potential for acid generation. The majority of the sulphur is present as finely disseminated sulphide.

A second consideration from the environmental perspective is the concentration of certain metals and metal-like substances in the chromite mineralization and mine rock. Except for iron, chromium and magnesium, metal concentrations in the mine rock are generally very low. Metals concentrations in the chromite mineralization are much higher than that of the mine rock, with levels of iron, chromium, and magnesium above the Ontario typical ranges.

The geochemical sampling program and data interpretation, as generally required under Section 58 of Schedule 1 of O. Regulation 240/00 (as amended) will be provided via a Form 2 (Notice of Material Change) prior to excavating rock at the Mine site, in accordance with MNDMF requirements. Should the geochemical characterization indicate that there are chemical stability risks that need to be mitigated in accordance with Section 59(1) of Schedule 1 f O. Regulation 240/00 (as amended), a management plan will be developed and included in the Form 2 submission with any additional financial assurance that may be required.

18.7.11 Consultation With Aboriginal Peoples

The Mine site is understood to be within the traditional land of the Nishnawbe Aski Nation (Webequie and Martin Falls First Nations). Introductory meetings were held with Nishnawbe Aski Nation and communications regarding the Project are on-going.

18.7.12 Permitting

Preliminary Exploration Phase

The current field exploration (geophysics, geological mapping, limited surface stripping, surface diamond drilling, <1000 tonne bulk sample) is considered to be preliminary exploration and may currently be executed without triggering the requirement for a Closure Plan, pursuant to Part VII of the *Mining Act*. Approvals under the *Public Lands Act*, *Ontario Water Resources Act* and the *Mining Act* may be required for specific preliminary exploration activities, as described at

http://www.ontario.ca/en/information_bundle/mineral/STEL01_033476

KWG has adopted a progressive approach during their preliminary exploration phase and has initiated consultation with stakeholders and the First Nations that have been identified as having an interest in the Project. Engaging the affected First Nations will identify issues of mutual interest (*e.g.* protection of cultural heritage values; protection of the environment; sourcing local labour, goods and services; developing a local labour pool for potential future employment opportunities, economic development initiatives of mutual benefit, etc.) as well as constraints to development (*e.g.* cultural heritage values; incompatible development plans, etc.) so that these may be taken into consideration during the early design stages of the Project. In addition, the identification of KWG's contact people establishes credibility with stakeholders and facilitates the timely resolution of potentially contentious issues.

18.8 MANPOWER COMPLEMENT SUMMARY

Once in operation, the total workforce for the Big Daddy Project, including mining, processing, surface facilities and general and administrative, will total approximately 264 employees. A summary of the workforce is presented in Table 18.8-1. The departmental workforce listings have already been presented in each appropriate chapter.

The operations will be managed by a senior management team led by the General Manager. The other senior management will include:

Mine Superintendent
Surface Department Superintendent
Maintenance Superintendent
Chief Engineer
Chief Geologist
Head of Health, Safety and Security
Controller

All of the employees, contractors and lower level supervision will report to or through these positions.

18.9 PROJECT SCHEDULE

The Big Daddy pre-production period will take approximately 3 years. The critical project activities to reach commercial production are:

- Railway line construction;
- Port facilities infrastructure (if required);
- Power line construction;
- Crushing plant and chromite product storage construction; and
- Open pit site pre-stripping.

18.10 CAPITAL EXPENDITURES

The capital expenditures estimates are based on budget pricing from suppliers, consultants, contractors and a review of other Canadian projects. Smaller equipment and facilities

Table 18.8-1. Operations Workforce.

Department	Complement
Mine (Average)	174
Product Preparation Pant	24
General & Administration	31
Rail Department	35
TOTAL	**264**

component costs were factored based on industry norms for the type of facility being constructed and, where possible, adjusted to reflect local conditions.

Labour rates are based on contractor costs in the region, and country, for similar types of work. Where costs were either not available or irrelevant, costs from other similar projects in Canada were used. The rates used include all cost and profit components payable to contractors.

All cost estimates are in 2011 constant Canadian Dollars.

18.10.1 Mining

Mine capital expenditures are primarily related to mining and support equipment for the open pit operation. The capital expenditures are presented in Table 18.10-1. The overburden stripping carried out in pre-production years -2 and -1 is also capitalized. The total mine pre-production expenditures are estimated to be $156.2 million. It is assumed that the overburden would be stripped by a contractor. The estimated pre-production stripping costs, at $10 per cubic metre of overburden are:

- Year -2 $28 million
- Year -1 $29 million

The projected mine sustaining capital expenditures are shown in Table 18.10-1 and total $421.5 million.

18.10.2 Product Preparation Plant

Capital expenditures for the product preparation plant are presented in Table 18.10-2 and total $15.7 million. This expenditure includes a contingency of 25%.

18.10.3 Infrastructure & Support Facilities

Total pre-production capital expenditures for project infrastructure are approximately $138.8 million. Table 18.10-3 provides the infrastructure capital costs breakdown, excluding all expenditures related to the railway line from the site to Nakina (which are provided in Table 18.10-4).

Table 18.10-1. Pit Equipment Capital Costs.

Year	Estimated Mine Equipment Capital Costs [1]
-1	$99.2 M
1	$29.9 M
2	$64.4 M
3	$15.6 M
4	$43.5 M
5	$17.0 M
6	$59.6 M
7	$18.8 M
8	$7.3 M
9	$23.5 M
10	$25.1 M
11	$25.2 M
12	$33.9 M
13	$43.4 M
14	($29.7 M) Salvage Value
15	$0.8 M
16	($12.1M) Salvage Value

[1] The above costs include the cost of the mine shop complex, mine offices and dry, and are net equipment salvage values.

Table 18.10-2. Product Preparation Plant Capital Expenditures ($).

DESCRIPTION	QUANTITY	UNIT	TOTAL COST ($)	Year 1	2	3	Total Cost ($)
CRUSHING FACILITY							
CIVIL/STRUCTURAL ISSUES							
Civil Systems							
Formwork	5000	m2	$156,000		156,000		$156,000
Reinforcing Steel	500	tonnes	$340,000		340,000		$340,000
Concrete, installed	1600	m3	$1,112,000		1,112,000		$1,112,000
Floor Slabs	250	m3	$174,000		174,000		$174,000
Grouting for columns, baseplates, machinery	25	m3	$28,000		28,000		$28,000
Excavation	2500	m3	$61,000		61,000		$61,000
Backfill	2000	m3	$94,000		94,000		$94,000
Geotextile	800	m2	$82,000		82,000		$82,000
Structural Systems							
Cladding	16000	m2	$1,508,000		1,508,000		$1,508,000
Roofing	9600	m2	$1,241,000		1,241,000		$1,241,000
Framing	450	tonnes	$2,493,000		2,493,000		$2,493,000
Platforms/Stairs/Interior Framing	60	tonnes	$392,000			392,000	$392,000
Mandoors	12	ea	$42,000			42,000	$42,000
Overhead Doors	6	ea	$102,000			102,000	$102,000
SUB-TOTAL			**$7,825,000**		7,289,000	536,000	$7,825,000
MECHANICAL ITEMS							
Gyratory Crusher	1	ea	$1,873,000			1,873,000	$1,873,000
Cone Crushers	2	ea	$1,417,000			1,417,000	$1,417,000
Screens	3	ea	$342,000			342,000	$342,000
Conveyors	400	m	$2,057,000			2,057,000	$2,057,000
Transfer Chutes/Conveyor Supports	25	tonnes	$160,000			160,000	$160,000
SUB-TOTAL			$5,849,000			5,849,000	$5,849,000
ELECTRICAL SYSTEMS							
Distribution electrics and cabling	1	ea.	$865,000			865,000	$865,000
Lighting and panels	1	ea.	$96,000			96,000	$96,000
Grounding	1	lot	$39,000			39,000	$39,000
SUB-TOTAL			$1,000,000			1,000,000	$1,000,000
BUILDING SERVICES							
Crusher Building Heating	1	lot	$106,000			106,000	$106,000
Dust Control	1	lot	$226,000			226,000	$226,000
Fire suppression system - hoist house/headframe	1	lot	$171,000			171,000	$171,000
SUB-TOTAL			$503,000			503,000	$503,000
MISCELLANEOUS							
Crusher Maintenance - lower floor 20 tonne crane	1	ea.	$167,000			167,000	$167,000
Gyratory service crane (40 ton)	1	ea.	$343,000			343,000	$343,000
SUB-TOTAL			**$510,000**			510,000	$510,000
TOTALS			**$15,687,000**	**$0**	**$7,289,000**	**$8,398,000**	**$15,687,000**

Table 18.10-3. Infrastructure and Support Services Capital Expenditures ($).

Infrastructure Capital	Quantity	Units	Unit Cost ($)	Total Cost ($)	Year -3	Year -2	Year -1	Total ($)
Site Roads	8	km	$75,000	$ 600,000	400,000	200,000		$600,000
Haul Roads	3	km	$60,000	$ 180,000		120,000	60,000	$180,000
Plantsite Preparation	25	ha	$50,000	$ 1,250,000	1,250,000			$1,250,000
Waste Rock Storage Facility	1	L.S.	$2,000,000	$ 2,000,000		2,000,000		$2,000,000
Shop/Warehouse	3,580	sq.m.	$1,000	$ 3,580,000		3,580,000		$3,580,000
Shop Equipment and Tools	1	L.S.	$750,000	$ 750,000		750,000		$750,000
Warehousing Equipping	1	L.S.	$200,000	$ 200,000			200,000	$200,000
Office	900	sq.m.	$2,000	$ 1,800,000			1,800,000	$1,800,000
Ofice Furniture, Equipment, Computers et	1	L.S.	$1,700,000	$ 1,700,000			1,700,000	$1,700,000
Environmental Department Equipment	1	L.S.	$100,000	$ 100,000			100,000	$100,000
Dry	750	sq.m.	$2,000	$ 1,500,000			1,500,000	$1,500,000
Dry Equipping	1	L.S.	$180,000	$ 180,000			180,000	$180,000
Metallurgical Laboratory	900	sq.m.	$1,900	$ 1,710,000			1,710,000	$1,710,000
Metallurgical Laboratory Equipping	1	L.S.	$200,000	$ 200,000			200,000	$200,000
Miscellaneous Buildings	900	sq.m.	$1,500	$ 1,350,000			1,350,000	$1,350,000
Surface Parking Areas	9,600	sq.m.	$30	$ 288,000			288,000	$288,000
Laydown Yard	900	sq.m.	$30	$ 27,000		27,000		$27,000
Camp & Catering Facilities	6,200	sq.m.	$1,500	$ 9,300,000		9,300,000		$9,300,000
Recreational Facilities	900	sq.m.	$1,500	$ 1,350,000		1,350,000		$1,350,000
Camp Equipment	1	L.S.	$1,000,000	$ 1,000,000		1,000,000		$1,000,000
Recreational Facilities Equipping	1	L.S.	$400,000	$ 400,000		400,000		$400,000
Main Power Stepdown Substation	1	L.S.	$4,000,000	$ 4,000,000		4,000,000		$ 4,000,000
Powerline	300	km	$100,000	$ 30,000,000	12,000,000	12,000,000	6,000,000	$30,000,000
Services Substation	1	L.S.	$1,000,000	$ 1,000,000		1,000,000		$1,000,000
Electrical Distribution	1	L.S.	$4,000,000	$ 4,000,000		2,000,000	2,000,000	$4,000,000
Communication & Data Link	30,000	metres	$200	$ 6,000,000	2,500,000	2,500,000	1,000,000	$6,000,000
Communications	1	L.S.	$500,000	$ 500,000		500,000		$500,000
Fuel Storage	1	L.S.	$1,000,000	$ 1,000,000		1,000,000		$1,000,000
Explosives Magazines	1	L.S.	$100,000	$ 100,000		100,000		$100,000
Fresh Water Pipeline	1	L.S.	$2,000,000	$ 2,000,000		2,000,000		$2,000,000
Water Treatment Plant	1	L.S.	$5,000,000	$ 5,000,000			5,000,000	$5,000,000
Sewage Disposal	1	L.S.	$500,000	$ 500,000			500,000	$500,000
Staff Pickup Trucks	15	each	$50,000	$ 750,000			750,000	$750,000
Garbage Truck	1	each	$250,000	$ 250,000			250,000	$250,000
Subtotal Infrastructure Capital				**$ 84,565,000**	**16,150,000**	**43,827,000**	**24,588,000**	**$84,565,000**
EPCM	15	%		$ 12,685,000	2,423,000	6,574,000	3,688,000	$ 12,685,000
Contractors Overhead	12	%		$ 10,148,000	1,938,000	5,259,000	2,951,000	$10,148,000
First Fills, Commissions, Vendor Reps	1	L.S.		$ 1,522,000			1,522,000	$ 1,522,000
Spare Parts	1	L.S.		$ 2,114,000			2,114,000	$ 2,114,000
Contingency	25%			$ 27,759,000	$5,128,000	$13,915,000	$8,716,000	$27,759,000
Total Infrastructure Expenditures				**$138,793,000**	**$25,639,000**	**$69,575,000**	**$43,579,000**	**$138,793,000**

Table 18.10-4. Railway Capital Expenditures ($).

Item	Quantity	Units	Unit Cost ($)	Total Cost ($)	Year 1	Year 2	Year 3	Total
Rail & Bed	365	km	781,644	285,300,000	100,007,000	100,255,000	85,038,000	**$285,300,000**
Bridges & Structures	10,300	metres	32,000	329,600,000	125,248,000	104,000,000	100,352,000	**$329,600,000**
Communications & Misc. Infrastructure	1	L.S.	5,000,000	5,000,000	1,750,000	1,750,000	1,500,000	**$5,000,000**
Rolling Stock	7	L.S.	10,600,000	74,200,000		25,000,000	49,200,000	**$74,200,000**
Transfer Yards	2	L.S.	7,000,000	14,000,000	10,500,000	3,500,000		**$14,000,000**
Facilities	1	L.S.	6,200,000	6,200,000		3,000,000	3,200,000	**$6,200,000**
EPCM	5%	percent		35,700,000	12,495,000	12,495,000	10,710,000	**$35,700,000**
Contingency	20	%		150,000,000	50,000,000	50,000,000	50,000,000	**$150,000,000**
Total Railway Expenditures				**$900,000,000**	**$300,000,000**	**$300,000,000**	**$300,000,000**	**$900,000,000**

The largest single infrastructure expenditure requirement is $900 million for the railway line, rolling stock and other railway facilities. At lump chromite shipping rates of approximately 3 million tonnes per year 6 trains are required, at a total cost of $74.2 million. Camp accommodations for operations, locomotive and car maintenance and track equipment and maintenance personnel at a total expenditure of $6.1 million has been budgeted, as well as $6.2 million for track maintenance equipment.

Other major costs are related to the power line ($35 million), camp ($12 million) and the fibre optic cable link to Nakina ($6 million).

The capital expenditures also include EPCM and supply of the first fills such as fuel.

A 20% contingency is included in the total expenditures.

18.10.4 Project Total Expenditures

The estimated project total pre-production capital expenditure, inclusive of contingencies, is approximately $784 million. A summary of project pre-production capital expenditures is presented in Table 18.10-5. A working capital allowance of $40 million has also been allocated to the project.

For the project capital expenditures estimate the cost of the railway has been assumed to be borne between the Project and other area projects presently under development or study for development and thus allocates 50% of the cost to the Project. The total cost for the railway included in the project capital expenditure is $450 million.

Sustaining Capital

Sustaining capital expenditures are estimated to be $438 million. The major mine sustaining capital expenditures are associated with equipment rebuilds and replacement. Other significant sustaining capital expenditures include periodic costs associated with maintaining operations at existing levels. Closure costs estimates are included at a total cost of $10 million at the end of the project life.

18.11 OPERATING COSTS

Operating costs are based on Canadian norm prices from suppliers and other similar type Canadian projects, for consumables and parts. The cost of power is based on rates charged by Hydro One for similar sized power consumers in the province.

Labour costs for the operating period are based on the manpower schedules presented for each department and the associated labour costs. The costs include a benefits component of approximately 40%, as well as costs for travel to site on work rotations.

Table 18.10-5. Project Pre-Production Capital Expenditures ($).

Component	Total Expenditure ($)
Mine	$156,190,000
Product Preparation Plant	$ 15,687,000
Railway	$450,000,000
Infrastructure	$138,793,000
Project Management Infrastructure & Mine	$ 18,500,000
Engineering Studies	$ 5,000,000
TOTAL EXPENDITURES	**$784,170,000**

All costs are quoted in constant Q1 2011 Canadian Dollars.

18.11.1 Mining

The mine operating cost estimates were developed from first principles. Projected average haulage profiles were used to estimate the pit loading and haulage equipment fleet requirements. This information provided a basis for estimating the annual operating costs including direct labour, supervision, maintenance labour, technical services, and mine consumables and mine indirect costs such as pit dewatering costs. Critical operating cost components are based on the following input costs:

- The diesel fuel price is assumed to be $1.20 / litre. The electrical power cost is assumed to be $0.12 per kWh.
- A preliminary budget estimate of the cost of bulk emulsion explosive was obtained from a supplier.
- Preliminary estimates of the operating costs for the shovels and haul trucks were developed with input from a Komatsu dealer and P&H MinePro.

The open pit is scheduled to produce 69.4 Mt of lump chromite mineralization and waste rock in year 2 of production. The estimated mine operating costs for drilling, blasting, and haul truck loading and hauling in year 2 are summarized in Table 18.11-1. A more detailed breakdown of the haul truck loading and hauling costs in year 2 is presented in Table 18.11-2.

Yearly mining costs were derived from the unit mining costs and production schedule. The estimated annual operating costs breakdown are shown in Table 18.11-3. The yearly costs vary in relation to the production schedule, but life of mine average mining costs are $1.90 per tonne for lump chromite mineralization and $1.70 per tonne for waste.

18.11.2 Product Preparation

The product preparation plant total operating cost for crushing, screening, stockpiling and loadout into rail cars would be $1.96 per tonne of lump chromite mineralization. The component cost breakdown is shown in Table 18.11-4.

18.11.3 General & Administration Operating Costs

Infrastructure operating costs have been included in G&A costs. Surface ½ ton pickup trucks will be utilized by staff to travel around site and their operating costs have been included in G&A costs.

Table 18.11-1. Estimated Mine Operating Costs in Year 2.

Estimated Mine Operating Costs in Year 2		
Activity	**Direct Shipping and Low Grade Mineralization**	**Waste Rock**
Production Drilling [1]	$ 0.22 / t	$ 0.14 / t
Grade Control [2]	$ 0.02 / t	-
Blasting [3]	$ 0.26 / t	$ 0.24 / t
Subtotal	$ 0.50 / t	$ 0.38 / t
Loading and Haulage	$ 1.05 / t	$ 1.05 / t
Total	$ 1.55 / t	$ 1.43 / t

1 Includes driller and maintenance labour costs, diesel fuel and drilling consumable costs.

2 Cost allowance for the analysis of grade control samples collected from blast holes drilled in direct shipping mineralization and in adjacent blast holes near the contact.

3 Cost of blasting consumables including secondary blasting consumables. Blasting design and explosive supplier supervision, labour and rental costs are included elsewhere and are projected to be $2.3 M in year 2.

Table 18.11-2. Estimated Load and Haul Operating Cost in Year 2.

Estimated Load and Haul Operating Cost in Year 2	
Item	**$ / day**
Open Pit Operations Labour [1]	$ 61,093
Open Pit Maintenance Labour [2]	$ 9,035
Technical Services Labour [3]	$ 6,254
Direct Supplies:	
Electrical Power	$ 2,504
Diesel Fuel	$ 63,970
Lubricants	$ 1,919
Parts, Ground Engagement Tools, and Tires [2]	$ 63,396
Subtotal	$ 208,171
Scheduled tonnes mined per day	198,440
Direct load and haulage cost	$ 1.05/t mined

1. Estimated labour cost for open pit operations supervision and equipment operators. As an example, the estimated cost of a haul truck driver is $ 765/12-hour shift based on a $ 36.00/hour base rate plus overtime and shift premium allowances and 40% payroll burdens. Room & board and travel costs are additional and are included in the $ 61,093/day labour cost.

2. It is assumed most of the pit equipment will be maintained by the equipment suppliers as part of maintenance and repair contracts (MARC). The $ 9,035/day maintenance labour cost is for mine maintenance labour not included in a MARC.

3. Technical services personnel including geologists, mine engineers, and technicians.

Table 18.11-3. Estimated Annual Mine Operating Costs.

Year	Estimated Annual Mine Operating Costs	
	$/t Lump Chromite Mineralization	$/t Waste Rock
1	$1.50	$1.38
2	$1.55	$1.43
3	$1.64	$1.52
4	$1.72	$1.60
5	$1.75	$1.63
6	$1.69	$1.57
7	$1.63	$1.51
8	$1.67	$1.55
9	$1.73	$1.61
10	$1.79	$1.67
11	$1.90	$1.78
12	$2.06	$1.94
13	$2.25	$2.13
14	$2.61	$2.49
15	$2.61	$2.49
16	$2.61	$2.49
LOM	**$1.90**	**$1.70**

Table 18.11-4. Product Preparation Plant Operating Cost Components ($).

Area	Cost ($/t)
Maintenance	$0.55
Power	$0.18
Manpower	$0.73
Rail Loadout and Support	$0.50
TOTAL	**$1.96**

Operating costs for surface mobile equipment required to maintain the surface infrastructure and provide surface services have been included in the mine department.

Administration operating costs include costs and taxes for maintaining the property in good standing, land taxes, and resource usage fees (water, etc.). The G&A operating costs encompass all operating costs associated with operating the offices and providing materials and supplies for staff functions. The total yearly operating costs are estimated to be approximately $7 million (presented in Table 18.11-5), of which approximately $4.4 million is for salaries and benefits. The total G&A equates to an average of $2.88 per tonne of potentially economic mineralization processed.

Manpower costs represent approximately 60% of G&A operating costs. Employee benefits component accounts for approximately 35% the total salary for each employee.

18.11.4 Product Transport to Customers

Operating costs for the minesite to Nakina railway (350 kilometres approximately) were estimated and total approximately $10 per ton of lump chromite mineralization. An additional annual maintenance cost of $12,420 to $18,630 per kilometre of track, or $1.51 to $2.26 per tonne of lump chromite mineralization, is estimated for the railway.

Shipping brokers have indicated that use of CN or a third party such as Ontario Northland could transport lump chromite mineralization from Nakina to sea ports. The lump chromite mineralization would at the ports be loaded into bulk ocean going vessels, probably Capesize, for delivery to China or elsewhere. The cost for these components is estimated to be between $60 and $70 per tonne of lump chromite mineralization.

18.11.5 Project Total Operating Costs

The total average to smelter operating cost over the mine life is $130.37 per tonne of potentially economic lump chromite mineralization. Table 18.11-6 presents a summary table showing the life of mine average operating costs for each department on a cost per tonne of potentially economic lump chromite mineralization produced basis. The average on site operating costs per tonne of potentially economic lump chromite mineralization mined are estimated to be $47.28 for mining, $1.96 for product preparation and $2.88 for G&A. The remaining costs are for transportation of product to customers and the 1% NSR Royalty.

18.12 FINANCIAL ANALYSIS

The expected cashflows are calculated using a lump chromite mineralization price of $US 325 per tonne.

18.12.1 Market Analysis

Although many minerals contain chromium in low concentrations, the only commercial ore mineral is chromite, an iron chromium oxide ($FeCr_2O_4$). Chromite is found in peridotite

Table 18.11-5. General & Administration Operating Costs ($).

	Per Year
Salaries & Overhead	$4,367,000
Communications/IT	$120,000
Equipment Rental & Maintenance	$35,000
Computer Supplies & Software	$40,000
Office Supplies	$25,000
Warehouse Supplies	$10,000
Dry Supplies	$30,000
Surface Buildings Maintenance	$100,000
Shop Tools Replacements/Repairs	$50,000
Electrical Distribution Repairs	$75,000
Road Maintenance Materials	$50,000
Water Treatment Plant Supplies	$1,000,000
Postage, Courier & Light Freight	$50,000
Insurance	$250,000
Permits & Licences	$10,000
Bank Charges	$7,000
Professional Fees - Accounting	$30,000
Professional Fees - Legal	$15,000
Recruitment/Relocation	$50,000
Security	$7,000
Safety, Clothing and Training	$75,000
First Aid	$15,000
Dues & Subscriptions	$3,000
Public Relations	$50,000
Power	$241,000
Surface Transportation - Pickups	$75,000
Professional Fees - General	$60,000
Travel & Accommodation - Business	$60,000
Freight	$50,000
Miscellaneous	$50,000
TOTAL G&A COSTS	**$7,000,000**

Table 18.11-6. Life of Mine Average Operating Costs per Tonne of Potentially Economic Mineralization ($).

Department	Cost ($/t)
Mine	$ 47.28
Product Preparation	$ 1.96
G&A	$ 2.88
Rail Transport to Port	$ 60.00
Overseas Shipping	$ 15.00
NSR Royalty	$ 3.25
TOTAL	**$130.37**

from the Earth's mantle. It also occurs in layered ultramafic intrusive rocks and in metamorphic rocks such as serpentine, and corundum.

Because of its corrosion resistance and hardness, chromium is combined with iron and nickel to form stainless steel and/or super steel alloys. Its other main uses are in chrome electro-plating and refractory applications. Some chromium is still used in paint as it changes colours in combination with other elements. It is sometimes used for leather tanning, and as a dietary supplement, although these uses are falling into increased disrepute because of the toxicity of their by-products.

The principal end-use is in stainless steel and non-ferrous alloys with stainless steel production accounting for approximately 94% of chromite demand. The foundry sands, chromium chemicals and refractories applications sectors account for most of the remaining demand.

The majority of chromite used in metallurgical applications is smelted to ferrochromium before it is added to the steel melt. The principal ferrochromium alloys are high-carbon ferrochromium (HCFeCr) for which the chromite ores should have a Cr:Fe ratio of 2.0-3.6, and charge chrome which is produced from lower grade ores with Cr:Fe ratio of 1.3-2.0. Direct shipping, or lumpy ore, has a grain size over 6 mm and Cr_2O_3 grades of approximately 40%. This is a premium product as it can be fed directly to the ferroalloy smelter. Fine grained chromite (less than 6 mm) must be pelletized before use.

About 45% of the mined chromite ores in the world are produced in South Africa. Kazakhstan, India, Russia and Turkey are also substantial producers, and Finland, Iran and Brazil produce smaller amounts.

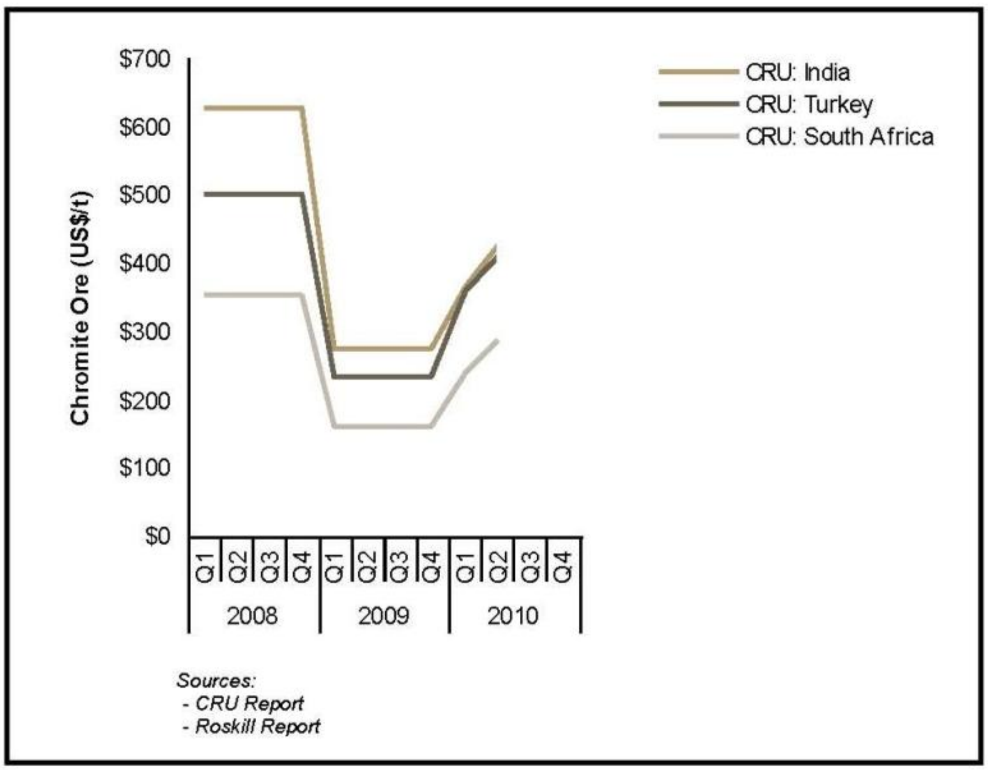
Prices

There is no terminal market, such as the London Metal Exchange, for chromite and ferrochromium prices are negotiated between buyers and sellers, either on the spot market or under contract.

Prices for chromite are quoted monthly by Industrial Minerals journal based on data from industry participants (producers, traders and consumers). It should be noted that such prices are indicative of market activity and do not represent actual transactions. M Creamer in Mining Weekly of March 2010, reported that "China bought the 2.9-million tons [of raw chromite ore] from South Africa at the comparatively low average price of $215/t including cost, insurance and freight (CIF), compared with the $360/t CIF it paid for raw ore from India - 67% more. China also paid 35% more for the raw ore it bought from Turkey.

A price chart for lump chromite ore is shown in Figure 18.12-1 with prices in Q1 2011 still within the ranges shown in the chart.

Figure 18-12-1. Chromite Ore Price Chart.



The expected price used in the financial analysis is an approximate mid-point of the lower and upper price ranges though Big Daddy lump chromite mineralization could be expected to be priced closer to the non-South African sources of supply pricing.

18.12.2 Cashflow Analysis

The cash flow model is included in the Appendix for the expected lump chromite price of $US325 per tonne.

A summary of the expected parameters used for the financial analysis is presented in Table 18.12-1.

For the purposes of this PEA lump chromite mineralization is material which has sufficient grade (i.e. greater than 35% Cr_2O_3) that smelting of the rock can be directly performed. Lower grade material (greater than 3.83% and less than 35% Cr_2O_3) would be stockpiled for possible processing in the future.

The cashflow analysis excludes any element or impact of financing arrangements. All exploration and acquisition costs incurred prior to the production decision are also excluded from the cashflows.

Capital expenditures, as shown in the capital section, will be incurred over a 3 year period, which is reflected in the discounted cash flow calculations. The cash flows include sustaining capital and capital expenditures contingency of approximately 25%.

Revenue is based on lump chromite shipped to overseas customers.

Costs for metal sales and shipping are included in the deductions that the refiner makes.

A maximum NSR royalty of 1% is included.

The expected cash flow analysis used the lump chromite price indicated above and a C$:US$ exchange rate of 1:1.

The discounted cashflow analysis uses 2011 Constant Canadian Dollar values.

18.12.3 Financial Returns

The overall level of accuracy of this study is approximately +/- 40 percent for all aspects of the project. This Preliminary Economic Assessment includes mainly the use of Indicated Resources (25.4 Mt) but also uses Inferred Resources (13.5Mt) which are considered too speculative geologically to have economic considerations applied to them, that would enable them to be categorized as Mineral Reserves. Therefore, there is no certainty that the results predicted by this Technical Report and the Preliminary Economic Assessment will be realized.

Table 18.12-1. Expected Project Parameters.

Potentially Mineable Resource *(based on the estimate dated March 30, 2010 by Micon International Limited)* after mining dilution & recovery	25.35 million tonnes Indicated Resources @ a grade of 38.02% Cr_2O_3 and 13.54 million tonnes Inferred Resources @ a grade of 37.03% Cr_2O_3
Overall Stripping Ratio (waste tonnes: lump chromite mineralization)	27:1
Estimated Mining Dilution	10 percent @ 0% grade
Projected Mining Recovery	97 percent
Lump Chromite Mineralization Produced[1]	8,000 tonnes per day typically
Pre-Production Capital Expenditures[2]	$784 million
Working Capital	$ 40 million
Total Sustaining Capital Expenditures	$438 million
Closure Cost	$ 10 million
Estimated Operating Costs ($/tonne): Mining Product Preparation General & Administration Transportation to Customers NSR Royalty Total Operating Costs	 $ 47.28 $ 1.96 $ 2.88 $ 75.00 $ 3.25 $130.37
Life of Mine (lump chromite mineralization)	16 Years

1 Includes the Project's 50% portion of estimated $900 million cost for railway construction.

The expected cashflow scenario reflects the Big Daddy project funding 50% of the full capital expenditures for the railway from the minesite to Nakina. There are several large projects being studied in the "Ring of Fire" and a possible scenario would be that the railway would be utilized to ship chromite products from and materials and equipment into the area. It is assumed that these operations would help fund construction of the railway.

The project expected investment and returns based on the expected cashflow parameters for the Big Daddy project are shown in Table 18.12-2.

Table 18.12-2. Expected Project Returns.

	Pre-Tax	After-Tax
Undiscounted Gross Revenue	$12.64 billion	$12.64 billion
Undiscounted Cashflow	$ 6.30 billion	$ 4.30 billion
NPV (8%)	$ 2.48 billion	$ 1.58 billion
NPV (10%)	$ 2.01 billion	$ 1.25 billion
IRR	42.0%	31.8%
Payback Period	2.5 years	2.5 years
NSR Royalty(undiscounted)	$126 million	$126 million

Free cashflow from the project is shown in Figure 18.12-2 and turns positive in Year 3 of operations. The project payback period is approximately 2.5 years.

18.12.4 Sensitivity Analysis

The project returns are very sensitive to the capital expenditures required to build the railway line from the property to Nakina, chromite prices and product transport costs. The railway expenditure allocation to the project requires a $450 million (50% of the total $900 million expenditure) investment in the pre-production period. Table 18.12-2 shows the project returns sensitivities to the critical cost components. Figures 18.12-3 and 18.12-4 present the sensitivity analysis graphs for pre-tax and after-tax financial results.

Further opportunities to share the expenditures for construction of the railway and if the project was required to fund the full construction expenditure for the railway were assessed. Using 25% and 100% allocations of the railway expenditures to the project the pre-tax IRR and NPV would range from approximately 57.2% to 30.2% and $2.87 billion to $2.29 billion.

Changes in the price of lump chromite were assessed within ranges experienced from the 2010 to expected future long term prices. At prices from $US300 to $US 425 lump chromite the pre-tax NPV and IRR range between $2.05 billion and $4.21 billion and 37.5% and 57.6% , respectively.

Table 18.12-1. Project Returns Sensitivity Analysis.

Variable	Pre-Tax NPV @ 8% ($billions)	After-Tax NPV @ 8% ($billions)	Pre-Tax IRR (%)	After-Tax IRR (%)
Chromite Price - $350	2.92	1.89	46.2	35.3
Chromite Price - $400	3.78	2.49	54.0	41.7
Chromite Price - $425	4.21	2.79	57.6	44.7
Chromite Price - $300	2.05	1.28	37.5	28.1
Product Transport Cost +15%	2.29	1.45	40.0	30.1
Product Transport Cost -15%	2.68	1.72	43.9	33.4
Capital Expenditure - 15%	2.57	1.66	46.7	35.6
Capital Expenditure +15%	2.39	1.51	38.1	28.7
Capital Expenditure +25%	2.34	1.45	35.9	26.9
Capital Expenditure +50%	2.19	1.32	31.3	23.3

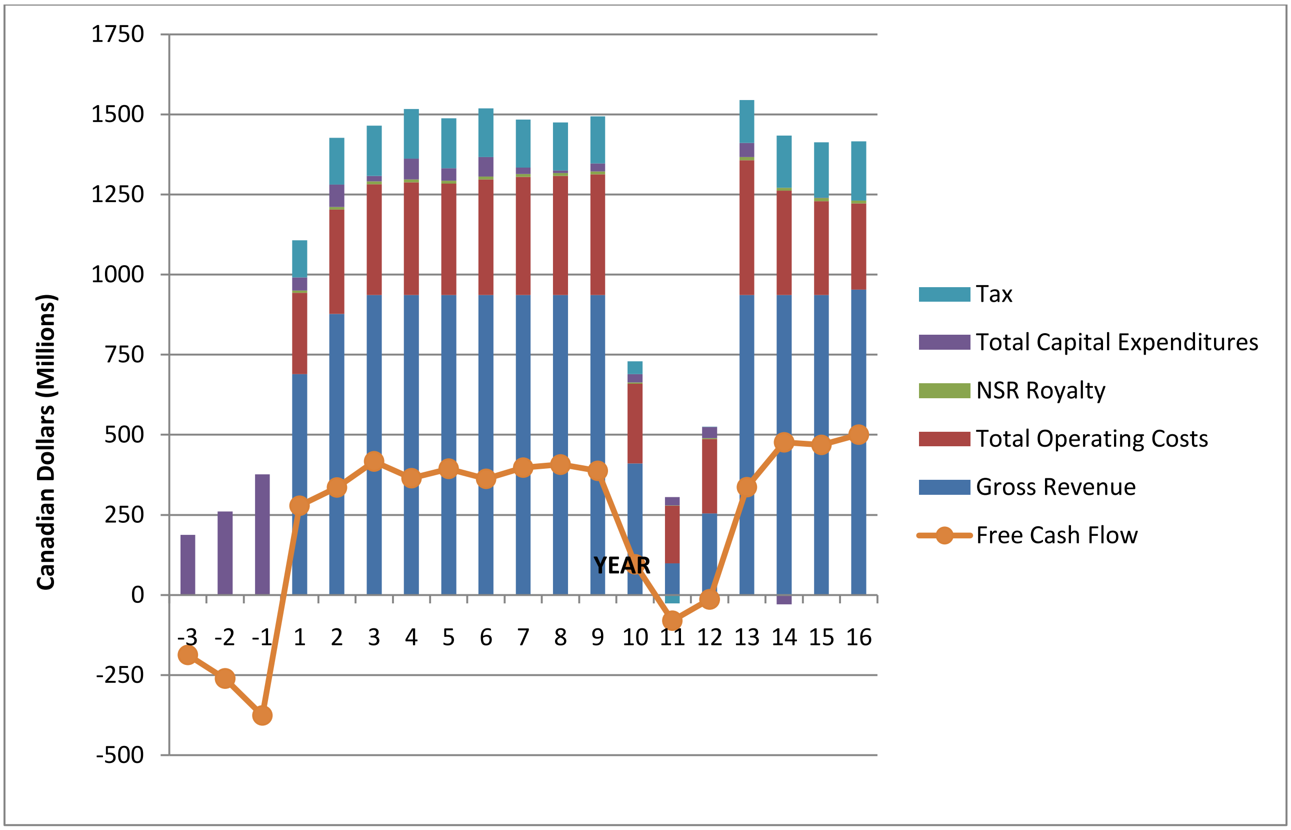


Figure 18.12-2. Big Daddy Project Base Case Free Cashflow.

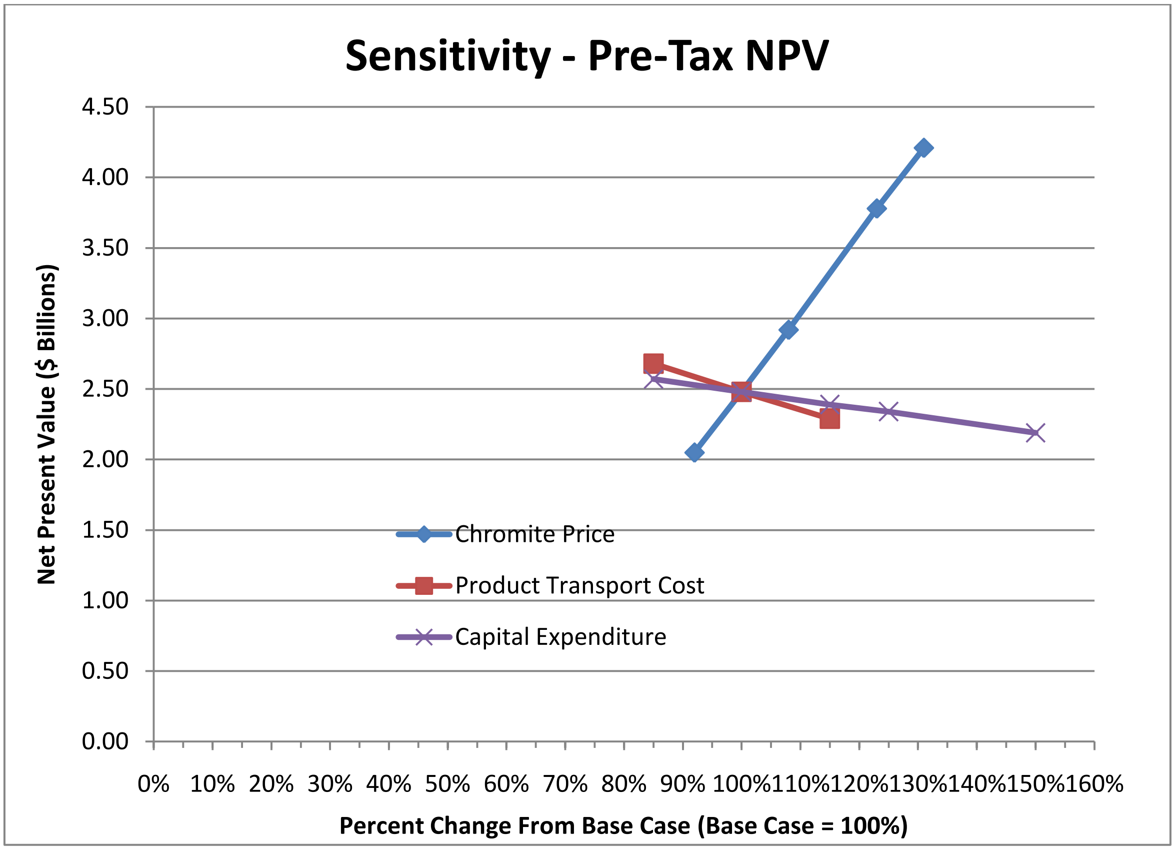


Figure 18.12-3. Big Daddy Project Base Case Pre-Tax NPV Sensitivities.

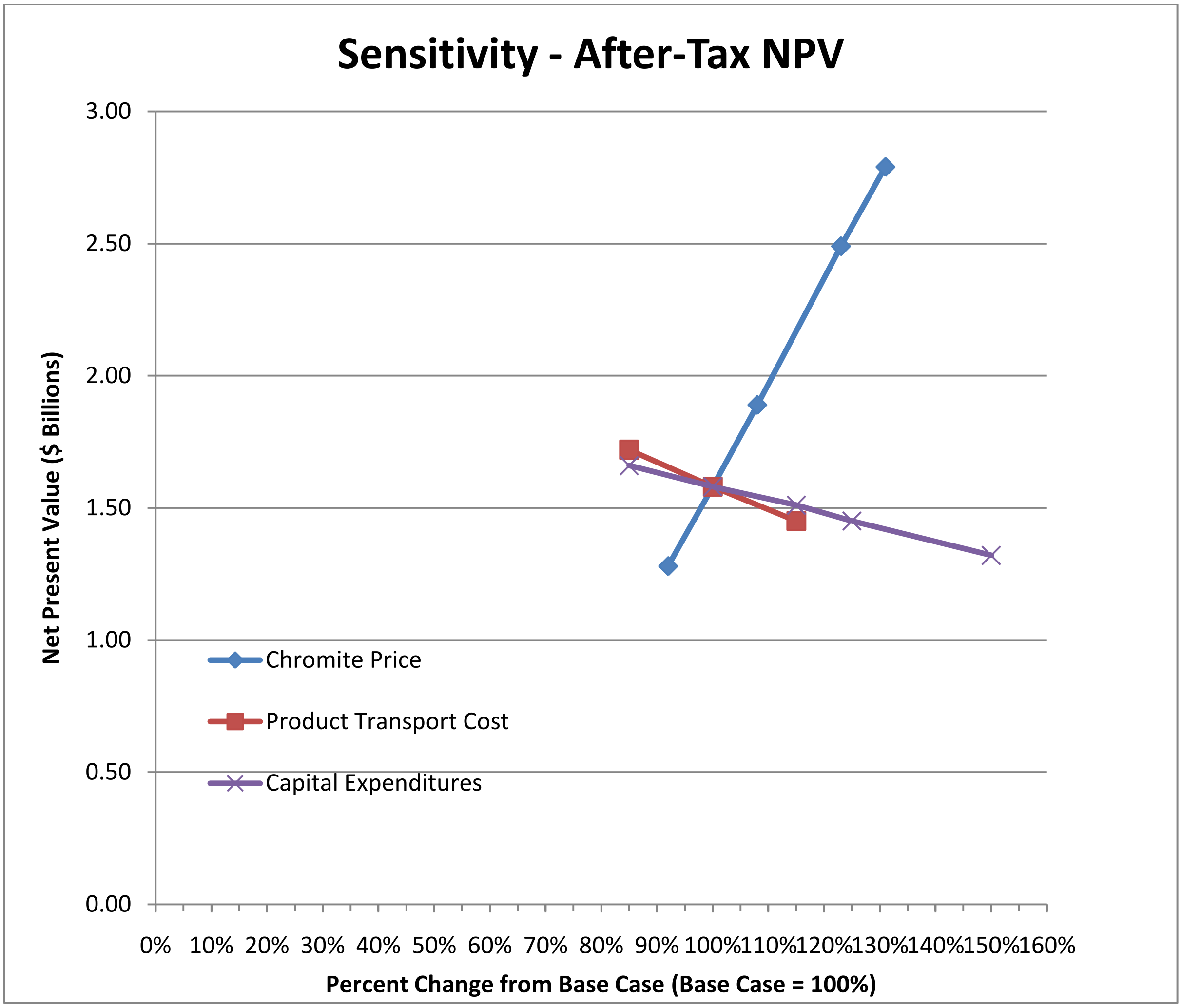


Figure 18.12-4. Big Daddy Project Base Case Sensitivities.

Chromite product transport cost has a lesser sensitivity effect on the project financial returns but is still a significant factor in project success. For +or -15% changes in the transport costs the pre-tax NPV and IRR range between $2.29 billion and $2.68 billion and 40.0% and 43.9% , respectively.

18.13 INTERPRETATIONS AND CONCLUSIONS

This study has identified a potentially mineable open pit resource estimated to be 25.35 million tonnes of Indicated Resources at a grade of 38.02% Cr_2O_3 and 13.54 million tonnes of Inferred Resources at a grade of 37.03% Cr_2O_3, to an ultimate open pit depth of 570 metres. The resource will be mined at 8,000 tpd of potentially economic mineralization with a mine design providing an overall stripping ratio of approximately 27:1.

The product preparation plant will crush all material to minus 50 mm size and the final product stored in railway load out silos. Material will be taken from the silos and loaded into rail cars for transport to overseas shipping points.

Main infrastructure will include a railway line from the minesite to Nakina and powerline and fibre communications link along the railway corridor. On site infrastructure will include site and haul roads, site services, office buildings and camp and recreational facilities.

The financial analysis including estimated capital expenditures and operating costs for the project indicates positive payback, even including $900 million in capital expenditures required for a railway line.

The financial analysis for the base case using the potentially mineable resource stated above is shown in Table 18.13-1.

Table 18.13-1. Expected Project Returns.

	Pre-Tax	After-Tax
Undiscounted Gross Revenue	$12.64 billion	$12.64 billion
Undiscounted Cashflow	$ 6.30 billion	$ 4.30 billion
NPV (8%)	$ 2.48 billion	$ 1.58 billion
NPV (10%)	$ 2.01 billion	$ 1.25 billion
IRR	42.0%	31.8%
Payback Period	2.5 years	2.5 years
NSR Royalty(undiscounted)	$126 million	$126 million

The project payback period is approximately 2.5 years.

18.14 RECOMMENDATIONS

Readers of this report (Preliminary Economic Assessment) should be cautioned that the current owner of a majority interest (Cliffs Natural Resources Inc.) in the Big Daddy Deposit has indicated that it does not support the development of the project as described in this report. Therefore this report should not be relied upon as an indication or representation that the Project will be developed in the manner described herein or at all.

Based on the results of this Preliminary Economic Assessment, recommendations are:

1. Advance the Big Daddy Project to the Feasibility Study phase, subject to the joint venture partners reaching agreement on moving the process forward..

2. Undertake diamond drilling to confirm the high grade lump chromite core, which is amenable to shipping for smelting. Also complete diamond drilling to, where possible, upgrade the inferred resources to at least indicated resources, with particular attention to Phase 3 mining resources which is materially comprised of inferred resource.

3. Complete the following technical studies to a feasibility level of accuracy:

    a. Nakina to minesite railway alignment and railway line design, capital expenditures and operating costs. This would require geotechnical investigations and approximately 10 to 15% of the relevant engineering to be completed.

    b. Main line railway transportation and overseas shipping costs studies.

    c. Chromite marketing and pricing studies.

4. While the overburden, low grade and waste stockpiles have been included in the mine plan a geotechnical study should be completed to assess the stability of the stockpiles and provide geotechnical designs for the overburden and waste stockpiles.

5. Further assess the potential for acid rock and metal leaching from the mine materials which will be exposed.

6. Evaluate environmental assessment requirements in consultation with regulatory authorities and continue social consultation and environmental baseline work.

7. Evaluate overburden stripping options.

8. Perform geotechnical drilling and slope stability analysis.

9. On completion of items 3 to 7, to a level providing sufficient confidence in the estimates to support continuation to a Feasibility Study, update the PEA or prepare a Pre-Feasibility Study (including cost estimate results generated by the technical studies), to continue justification of a Feasibility Study.

10. First priorities in the Feasibility Study Phase should be:

    Updating and improving the geology block model.

    Complete transportation, market and processing trade-off studies.

11. Assess the railway system configuration and capital expenditures to develop the optimum system and investigate options for financing the railway expenditures.

12. The Feasibility Study Phase will require expenditures of approximately $33 million (See Table 18.14-1). Prior to commencing a Feasibility Study approximately 2.5 years of studies are required. The Feasibility Study would take approximately 2-3 years to complete.

Table 18.14-1. Feasibility Phase Budget ($).

Component	Total Cost ($)
Trade Off Studies Upgrading	
Product Transport	$ 500,000
Railway	$ 4,000,000
Market Analysis	$ 500,000
Power	$ 20,000
Feasibility Components	
Geology & Resources	$11,700000
Rock Mechanics Studies	$ 150,000
Mine Design & Optimization	$ 200,000
Geotechnical Drilling	$ 2,000,000
Metallurgical Testwork	$ 500,000
Processing	$ 100,000
Infrastructure & Surface Facilities	
Onsite Facilities (Shops, Offices, Camp, etc.)	$ 500,000
Roads & Power	$ 125,000
Environmental & Socio-Economic	
Baseline Studies	$ 1,625,000
Camp & Logistics	$ 4,000,000
General & Administration	$ 150,000
Project Indirects (Project Management, etc.)	$ 50,000
Financial Analysis	$ 50,000
FS Report Preparation	$ 100,000
Expenses - Review Meetings, Site Visit, etc.	$ 100,000
Estimated Total Cost	**$26,370,000**
Contingency	**$ 6,592,000**
Total Cost	**$32,962,000**

19.0 REFERENCES

Micon International Limited., `` Spider Resources Inc. and KWG Resources Inc. Technical Report on the Mineral Resource Estimate for the Big Daddy Chromite Deposit McFaulds Lake Area, James Bay Lowlands, Northern Ontario, Canada ", March 30, 2010.

The Tennessee Valley Authority and the Center for Business and Economic Research, Lewis College of Business, Marshall University, "The Incremental cost of Transportation Capacity in Freight Railroading: An Application to the Snake River Basin", July 1998, prepared for the U.S. Army Corps of Engineers.

20.0 CERTIFICATES

CERTIFICATE of QUALIFIED PERSON
MALCOLM BUCK, P.ENG.

I, Malcolm Buck, M. Eng., P. Eng., residing at 164 Castle Crescent, Oakville, Ontario, Canada do hereby certify that:

1. I am a Senior Associate Mining Engineer at NordPro Mine & Project Management Services Ltd.

2. This certificate applies to the technical report titled "NI 43-101 Technical Report on the Preliminary Economic Assessment of the Big Daddy Chromium Project, McFaulds Lake Area, James Bay Lowlands, Northern Ontario, Canada" (the "Technical Report"), with an effective date of April 12, 2011.

3. I am a graduate of The Technical University of Nova Scotia, with a Bachelor of Engineering in Mining Engineering (1983). I have also obtained a Masters of Engineering, in Mining Engineering (Mineral Economics) from McGill University (1986).

4. I am licensed by the Professional Engineers Ontario (License No. 5881503). In addition, I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum.

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. My relevant experience for the purpose of the Technical Report is:

   - Practiced my profession continuously since 1983.
   - Extensive and progressively more senior engineering and operational duties at base metals, gold and uranium mining operations and development projects.
   - 15 years experience performing all types of feasibility studies and due diligence and strategic planning studies for mines and mining companies.

7. I authored Sections 2 and 3 and co-authored the Executive Summary and Sections 4, 5 and 18 of the technical report.

8. I have not visited the Property that is the subject of this Technical Report.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

10. I am independent of the issuer applying all of the tests in sect 1.4 of NI 43-101.

11. I have not had prior involvement with the Property that is the subject of this Technical Report.

12. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.

Effective date: April 12, 2011
Signing Date: May 26, 2011

{SIGNED AND SEALED}
[Malcolm Buck]

Malcolm Buck, P.Eng.

CERTIFICATE of QUALIFIED PERSON
DAVID A. ORAVA, P. ENG.

I, David A. Orava, M. Eng., P. Eng., residing at 19 Boulding Drive, Aurora, Ontario, L4G 2V9, do hereby certify that:

1. I am an Associate Mining Engineer at NordPro Mine & Project Management Services Ltd. and President of Orava Mine Projects Ltd.

2. This certificate applies to the technical report titled "NI 43-101 Technical Report on the Preliminary Economic Assessment of the Big Daddy Chromium Project, McFaulds Lake Area, James Bay Lowlands, Northern Ontario, Canada" (the "Technical Report"), with an effective date of April 30, 2011.

3. I am a graduate of McGill University located in Montreal, Quebec, Canada at which I earned my Bachelor Degree in Mining Engineering (B.Eng. 1979) and Masters in Engineering (Mining - Mineral Economics Option B) in 1981. I have practiced my profession continuously since graduation. I am licensed by the Professional Engineers of Ontario (License No. 34834119).

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.

5. My summarized career experience is as follows:

   - Mining Engineer – Iron Ore Company of Canada. 1979-1980
   - Mining Engineer – J.S Redpath Limited / J.S. Redpath Engineering. 1981-1986
   - Mining Engineer & Manager Contract Development – Dynatec Mining Ltd. 1986-1990
   - Vice President – Eagle Mine Contractors 1990
   - Senior Mining Engineer – UMA Engineering Ltd. 1991
   - General Manager - Dennis Netherton Engineering 1992-1993
   - Senior Mining Engineer – SENES Consultants Ltd. 1993-2003
   - President – Orava Mine Projects Ltd. .... 2003 to present
   - Associate Mining Engineer – NordPro Mine & Project Management Services Ltd..................2011

6. I am responsible for co-authoring Sections 1 and 18 of the Technical Report.

7. I have not visited the Property that is the subject of this report.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am an independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.

10. I have had no prior involvement with the Property that is the subject of this Technical Report.

11. I have read NI 43-101 and Form 43-101F1 and the Report has been prepared in compliance therewith.

Effective Date: April 12, 2011
Signed Date: May 26, 2011

{SIGNED AND SEALED}
[David Orava]

David Orava, M. Eng., P. Eng.

CERTIFICATE of QUALIFIED PERSON
Christopher Sharpe, P.ENG.

As a co-author of portions of this report entitled "NI-43-101 Technical Report on the Preliminary Economic Assessment of the Big Daddy Chromium Project, McFaulds Lake Area, James Bay Lowlands, Northern Ontario, Canada" (the "Technical Report") dated May 26, 2011, I, Christopher Sharpe, do hereby certify that:

1. I am an Associate Mining Engineer at NordPro Mine & Project Management Services Ltd.

2. I am a graduate of Dalhousie University, with a Bachelor of Engineering in Mining Engineering (2001).

3. I am a Professional Engineer registered with the Association of Professional Engineers Ontario (License No. 100053512). In addition, I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I have worked in the minerals industry for 8 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI-43-101. My work experience includes over 8 years in design and evaluation of mineral properties.

6. I have not visited the Property that is the subject of this Technical Report.

7. I am responsible for the preparation of Sections 18.1.2 to 18.1.6.

8. I am independent of the issuer applying all of the tests in section 1.4 of NI 43-101.

9. I have not had prior involvement with the Property that is the subject of this Technical Report.

10. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective date: April 12, 2011
Signing Date: May 26, 2011

{SIGNED AND SEALED}
[Christopher Sharpe]

____
Christopher Sharpe, P.Eng.

CERTIFICATE of QUALIFIED PERSON
BRIAN LEBLANC, P.ENG.

I, Brian LeBlanc, B.Sc., P. Eng., residing at 781 Community Hall Road, Thunder Bay, Ontario, Canada do hereby certify that:

1. I am Vice President & General Manager of NordPro Mine & Project Management Services Ltd.

2. This certificate applies to the technical report titled "NI 43-101 Technical Report on the Preliminary Economic Assessment of the Big Daddy Chromium Project, McFaulds Lake Area, James Bay Lowlands, Northern Ontario, Canada" (the "Technical Report"), with an effective date of April 12, 2011.

3. I am a graduate of the Haileybury School of Mines as a Mining Technician (1981). I have also obtained a Bachelor of Science degree in Mining Engineering from Michigan Technological University (1986).

4. I am licensed by the Professional Engineers Ontario (License No. 90427972).

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. My relevant experience for the purpose of the Technical Report is:

   - Extensive and progressively more senior engineering and operational duties at base metals, gold and nickel mining operations and development projects.
   - 6.5 years experience directing and overseeing several scoping level, pre-feasibility level and feasibility level studies for mines and mining companies.

   - Mill Operator – Giant Yellowknife Mines..1974 – 1975
   - Crusher Operator/Screening Plant Operator/Loadout Operator/Surveyor – Steep Rock Iron Mines Ltd..1976 - 1979
   - Mine Planner/Chief Surveyor – Nanisivik Mines Ltd....................................1981 - 1984
   - Mining Engineer/Underground Supervisor/General Foreman/ Technical Services Superintendent/ Mine Superintendent – Williams Mine...............................1986 – 2003
   - Manager of Mining – Kinross Kubaka Mine (Russia).....................................2003 – 2004
   - Technical Services Superintendent – Lac Des Isles Mines...............................2004 – 2006
   - Project Superintendent – Redpath Indonesia...2006 - 2007
   - Project Manager for Ontario – North American Palladium Ltd.....................2007 - 2010
   - Vice President and General Manager – NordPro Mine and Project Management Services Ltd...2010 - Present

7. I supervised preparation of the Technical Report and co-authored Sections 1 and 18 of the Technical Report.

8. I have not visited the Property that is the subject of this Technical Report.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

10. I am independent of the issuer applying all of the tests in sect 1.4 of NI 43-101.

11. I have not had prior involvement with the Property that is the subject of this Technical Report.

12. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.

Effective date: April 12, 2011
Signing Date: May 26, 2011

{SIGNED AND SEALED}
[Brian LeBlanc]

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Brian LeBlanc, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

CHARLEY Z. MURAHWI

As a co-author of this report entitled "NI 43-101 Technical Report on the Preliminary Economic Assessment of the Big Daddy Chromite Project, McFaulds Lake Area, James Bay Lowlands, Northern Ontario, Canada" dated May 27, 2011, I, Charley Z. Murahwi do hereby certify that:

1) I am employed as a Senior Geologist by, and carried out this assignment for, Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, telephone 416 362 5135, fax 416 362 5763, e-mail cmurahwi@micon-international.com.

2) I hold the following academic qualifications:

   B.Sc. (Geology) University of Rhodesia, Zimbabwe, 1979;

   Diplome d'Ingénieur Expert en Techniques Minières, Nancy, France, 1987;

   M.Sc. (Economic Geology), Rhodes University, South Africa, 1996.

3) I am a registered Professional Geoscientist in Ontario (membership #1618) and in Newfoundland and Labrador (membership #05662); I am also a member of the Australasian Institute of Mining & Metallurgy (AusIMM) (membership # 300395) and a registered Professional Natural Scientist with the SACNASP (membership #400133/09).

4) I have worked as a mining and exploration geologist in the minerals industry for over 28 years;

5) I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 12 years on Cr-Ni-Cu-PGE deposits (on and off- mine), and the balance on a wide variety of other mineral commodities including gold, silver, copper, tin, and tantalite.

6) I visited the Activation Laboratory in Thunder Bay on 10 January, 2009 and the Big Daddy mineral property, between 11 and 13 January, 2009 and on October 22, 2009.

7) I have had no prior involvement with the mineral property in question, other than that I was a co-author of the March 31, 2009 and March 30, 2010 Technical Reports.

8) As of the date of this certificate to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading;

9) I am independent of the parties involved in the Big Daddy property as described in Section 1.4 of NI 43-101.

10) I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with this Instrument.

11) I am responsible for the preparation of Sections 6 through 15 and Section 17 of this report.

Effective Date: May 27, 2011

Signing Date: May 27, 2011

"Charley Z. Murahwi" {signed and sealed}

Charley Z. Murahwi, M.Sc., P. Geo. Pr.Sci.Nat., MAusIMM