Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report

FINAL REPORT

Prepared for

Lac Otelnuk Mining Ltd.

Prepared by

R. W. Risto, M.Sc., P. Geo.
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Effective Date : March 25th, 2015
Issue Date : April 23rd, 2015

Project Number : 2013-076

Prepared for

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Prepared by:

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Effective Date: March 25th, 2015

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IMPORTANT NOTICE

This Report was prepared as a National Instrument 43-101 Technical Report for Lac Otelnuk Mining Ltd. ("**LOM**") by Met-Chem Canada Inc. ("**Met-Chem**"). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in Met-Chem's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this Report. This Report can be filed as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under Canadian securities laws, any other uses of this Report by any third party are at that party's sole risk.

DATE AND SIGNATURE PAGE – CERTIFICATES

Effective Date: March 25th, 2015

Issue Date: April 23rd, 2015

CERTIFICATE

I, Richard W. Risto, do hereby certify that:

1. I reside at 22 Northridge Ave, Toronto, Ontario, Canada, M4J 4P2.

2. I am a Senior Associate Geologist with Watts, Griffis and McOuat Limited, a firm of consulting engineers and geologists, which has been authorized to practice professional engineering by Professional Engineers Ontario since 1969, and professional geoscience by the Association of Professional Geoscientists of Ontario.

3. This certificate accompanies the report titled "*Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report*", dated effective March 25, 2015.

4. I am a graduate from Brock University, St. Catherines, Ontario with an Honours B.Sc. Degree in Geology (1977), Queens University, Kingston, Ontario with a M.Sc. Degree in Mineral Exploration (1983), and I have practised my profession for over 30 years.

5. I am a licensed Professional Geoscientist of the Association of Professional Geoscientists of Ontario (Membership # 276); Association of Applied Geochemists; and, Prospectors and Developers Association of Canada.

6. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7. I visited the Property August 28 and August 29, 2007 and August 13 to August 16, 2008.

8. I am solely responsible for Sections 7 to 11. With co-author Michael W. Kociumbas, I am jointly responsible for Section 12.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101.

10. My relevant experience includes 30 years of field exploration and project evaluation for both precious and base metal projects including a number of iron deposits both in Canada and internationally. I have had prior involvement with the Property that is the subject of this technical report, including acting as co-author of the following reports: "*NI 43-101 Technical Report on the Preliminary Economic Assessment for 50 MTPY Otelnuk Lake Iron Project, Quebec – Canada*" by Met-Chem Canada Inc., Project Number 2010-082a, April 8, 2011; "*A Technical Report and Mineral Resource Estimate for the Lac Otelnuk Iron Property. Labrador Trough – Northeastern Québec for Adriana Resources Inc.*" May 7, 2009; "*A Technical Report and Mineral Resource Estimate for the Lac Otelnuk Iron Property, Labrador Trough – Northeastern Québec, for Lac Otelnuk Mining Ltd.*" dated August 3, 2012; and *"Technical Report and*

Updated Mineral Resource Estimate for the Lac Otelnuk Iron Property, Labrador Trough, Northeastern Québec for Lac Otelnuk Mining Ltd." effective date: October 31, 2013.

11. I have read NI 43-101, Form 43-101F1 and the technical report and have prepared the technical report in compliance with NI 43-101, Form 43-101F1 and generally accepted Canadian mining industry practice.

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

signed by
" *Richard W. Risto* "

Richard W. Risto, M.Sc., P.Geo.
April 23, 2015

CERTIFICATE

I, Michael W. Kociumbas, do hereby certify that:

1. I reside at 420 Searles Court, Mississauga, Ontario, Canada, L5R 2C6.

2. I am a Senior Geologist and Vice-President with Watts, Griffis and McOuat Limited, a firm of consulting geologists and engineers, which has been authorized to practice professional engineering by Professional Engineers Ontario since 1969, and professional geoscience by the Association of Professional Geoscientists of Ontario.

3. This certificate accompanies the report titled "*Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report*" dated effective March 25, 2015.

4. I am a graduate from the University of Waterloo, Waterloo, Ontario with an Honours B.Sc. Degree in Applied Earth Sciences, Geology Option (1985), and I have practised my profession continuously since that time.

5. I am a licensed Professional Geoscientist of the Association of Professional Geoscientists of Ontario (Membership # 0417), and Professional Engineers and Geoscientists of Newfoundland and Labrador. I am Member of: Canadian Institute of Mining, Metallurgy and Petroleum (Membership #94100); Prospectors and Developers Association of Canada (Membership #10463).

6. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7. I have not visited the Property.

8. I am solely responsible for Sections 14. With co-author Richard W. Risto, I am jointly responsible for Section 12.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101.

10. My relevant experience includes more than 25 years of field exploration and project management for both gold and base metal projects, including a number of iron deposits both in Canada and internationally. I have extensive experience with Mineral Resource estimation techniques and the preparation of technical reports. I have had prior involvement with the Property that is the subject of this technical report, including acting as co-author of the following reports: "*NI 43-101 Technical Report on the Preliminary Economic Assessment for 50 MTPY Otelnuk Lake Iron Project, Quebec – Canada"* by Met-Chem Canada Inc., Project Number 2010-082a, April 8, 2011; "*A Technical Report and Mineral Resource Estimate for the Lac Otelnuk Iron*

Property. Labrador Trough – Northeastern Québec for Adriana Resources Inc." May 7, 2009; "*A Technical Report and Mineral Resource Estimate for the Lac Otelnuk Iron Property. Labrador Trough – Northeastern Québec. for Lac Otelnuk Mining Ltd."* dated August 3, 2012; and *"Technical Report And Updated Mineral Resource Estimate For The Lac Otelnuk Iron Property, Labrador Trough, Northeastern Québec For Lac Otelnuk Mining Ltd."* effective date: October 31, 2013.

11. I have read NI 43-101, Form 43-101F1 and the technical report and have prepared the technical report in compliance with NI 43-101, Form 43-101F1 and generally accepted Canadian mining industry practice.

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

signed by
" *Michael Kociumbas* "

Michael Kociumbas, P.Geo.
April 23, 2015

CERTIFICATE OF AUTHOR

To Accompany the Report entitled "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report" prepared for Lac Otelnuk Mining Ltd. effective as of March 25th, 2015, issued on April 23rd, 2015

I, Roberto Onel Viera Martinez, Eng., do hereby certify that:

1) I am a Senior Process Engineer, presently employed by SNC-Lavalin Inc., in the Mining & Metallurgy Division, with an office situated at 1140 de Maisonneuve Blvd. West, Montreal, Quebec, Canada;

2) I am a graduate of, Higher Technical University "Julio A. Mella", Santiago de Cuba, Santiago de Cuba, Cuba in Chemical Engineer obtained in 1989;

3) I am a member in good standing of the "*Ordre des Ingénieurs du Québec*" (141437);

4) I have worked as a process engineer in metallurgical plant operations and design continuously since graduation from university in 1989 (year);

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 by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I am responsible for Section 13 and portions of Section 17 of the "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report" dated March 25, 2015 with an effective date of April 23, 2015 , 2015 ("**Technical Report**");

7) I have not visited the site property that is the subject of this Technical Report;

8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, Sections 13 and 17 of the Technical Report contain all scientific and technical information that is required to be disclosed to make these Sections of the Technical Report not misleading;

9) I have read NI 43-101 and believe that Sections 13 and 17 of the Technical Report have been prepared in accordance with NI 43-101;

10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

This 8th day of April 2015

Original signed and sealed

Roberto Onel Viera Martinez
Senior Process Engineer
SNC-Lavalin Inc.

INGÉNIEUR · ENGINEER
Roberto Onel Viera Martinez
141437
QUÉBEC
2015-04-08

CERTIFICATE OF AUTHOR

To Accompany the Report entitled "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report" prepared for Lac Otelnuk Mining Ltd. effective as of March 25th, 2015, issued on April 23rd, 2015

I, Julien Lord, Eng., do hereby certify that:

1) I am an Engineering Manager employed by SNC-Lavalin Inc. up until March 18, 2015, in the Mining & Metallurgy Division, with an office situated at 1140 de Maisonneuve Blvd. West, Montreal, Quebec, Canada;

2) I am a graduate of École Polytechnique with a Bachelor's in Mechanical Engineering obtained in 1986;

3) I am a member in good standing of the "*Ordre des Ingénieurs du Québec*" (101060);

4) I have worked as an Engineer continuously since graduation from university in 1986, and in mining since 1989;

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 by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I have participated in the preparation of the "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report", dated March 25, 2015 with an effective date of _ April 23, 2015 ("Technical Report") and am responsible for portions of Section 17 and all of Section 18;

7) I visited the site property that is the subject of this Technical Report in June 2013 for 2 days and made a subsequent visit in September 2013 for 2 days;

8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, Sections17 and 18 of the Technical Report contain all scientific and technical information that is required to be disclosed to make these Sections of the Report not misleading;

9) I have read NI 43-101 and believe that Sections 17 and 18 of the Technical Report have been prepared in accordance with NI 43-101;

10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

This 8th day of April 2015

Original signed and sealed

Julien Lord

Julien Lord
Engineering Manager
SNC-Lavalin Inc.

INGÉNIEUR · ING. Julien Lord 101060 QUÉBEC 2015-04-08

CERTIFICATE OF QUALIFIED PERSON

I, Eric Giroux of Québec city, Quebec do hereby certify:

- I am the Director of Environmental Studies in Quebec City with WSP Canada Inc. with a business address at 5355, des Gradins Blvd, Quebec (Quebec).
- This certificate applies to the technical report entitled Lac Otelnuk Project Feasibility Study NI-43-101 Report – Section 20 (the "Technical Report").
- I am a graduate of the University Laval, M.Sc., 1992. I am a member in good standing of *Ordre des ingénieurs du Québec* (#108694) My relevant experience includes 19 years of experience in environmental studies for mining development and operations. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I have not visited the Project site.
- I am responsible for Section 20 of the Technical Report.
- I am independent of Lac Otelnuk Mine.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of April, 2015 at Quebec City.

INGÉNIEUR - ENGINEER
Eric Giroux
108694
QUÉBEC

Eric Giroux, Eng., M.Sc.
Director – Environmental Studies - Quebec
WSP Canada Inc.

CERTIFICATE OF AUTHOR

To Accompany the Report entitled "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report" prepared for Lac Otelnuk Mining Ltd. effective as of March 25th, 2015, issued on April 23rd, 2015

I, Martial Côté, Eng., do hereby certify that :

1) I am a Project Manager, presently employed by SNC-Lavalin Inc., in the Mining & Metallurgy Division, with an office situated at 5500 des Galeries Blvd., Suite 200, Quebec, Quebec, Canada;

2) I am a graduate of École Polytechnique, with a B.Sc. in Mine Engineering, obtained in 1974;

3) I am a member in good standing of the "*Ordre des Ingénieurs du Québec*" (025686);

4) I have worked as an Engineer continuously since graduation from university in 1974;

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 by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I have participated in the preparation of the "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report", dated March 25,, 2015 with an effective date of April 23, 2015 ("Technical Report") and am responsible for Section 21, Operating Costs;

7) I have not visited the site property that is the subject of this Technical Report;

8) As at the effective date of the Technical Report, to the best of my knowledge, information and belief, Section 21 of the Technical Report contains all scientific and technical information that is required to be disclosed to make this Section of the Report not misleading;

9) I have read NI 43-101 and believe that this Section of the Technical Report has been prepared in accordance with NI 43-101;

10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policy.

This 22nd day of April 2015

Original signed and sealed

INGÉNIEUR · ENGINEER
Martial Côté
25686
QUÉBEC

04-22-15

Martial Côté
Project Manager
SNC-Lavalin Inc.

CERTIFICATE OF AUTHOR

To Accompany the Report entitled "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report" prepared for Lac Otelnuk Mining Ltd. effective as of March 25th, 2015, issued on April 23rd, 2015

I, Sam Buccitelli, Eng., do hereby certify that :

1) I am a Project Manager presently employed by SNC-Lavalin Inc., in the Mining & Metallurgy Division, with an office situated at 1140 de Maisonneuve Blvd. West, Montreal, Quebec, Canada;

2) I am a graduate of the University of Waterloo with a Bachelor of Applied Science in Civil Engineering obtained in 1979;

3) I am a member in good standing of the "*Ordre des Ingénieurs du Québec*" (36265);

4) I have worked as an Engineer continuously since graduation from university in 1979 and in mining since 2000;

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 by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

6) I have participated in the preparation of the "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report", dated March 25, 2015 with an effective date of April 23, 2015 ("Technical Report") and am responsible for supervision of Section 21 Capital Cost prepared by SNC.Lavalin Lead Estimator, Mr. James Alarcon, and Section 22 prepared by by SNC.Lavalin Financial Analyst, Mr. Abhinav Pathak;

7) I visited the site property that is the subject of this Technical Report in June 2013 for 2 days and did not make any subsequent visits;

8) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, Sections 21 and 22 of the Technical Report contain all scientific and technical information that is required to be disclosed to make these Sections of the Report not misleading;

9) I have read NI 43-101 and believe that Sections 21 and 22 of the Technical Report have been prepared in accordance with NI 43-101;

10) I have read and understand NI 43-101 and I am considered independent of the issuer as defined in section 1.5 of NI 43-101 Rules and Policies.

This 17th day of April 2015

Original signed and sealed

Sam Buccitelli
Project Manager
SNC-Lavalin Inc.

INGÉNIEUR · ENGINEER
Samuel Buccitelli
36265
QUÉBEC

17/04/2015

CERTIFICATE OF AUTHOR

To Accompany the Report entitled "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report" prepared for Lac Otelnuk Mining Ltd. effective as of March 25th, 2015, issued on April 23rd, 2015.

I, Jeffrey Cassoff, Eng., do hereby certify that:

1) I am the Lead Mining Engineer with Met-Chem Canada Inc. with an office at suite 300, 555 René-Lévesque Blvd. West, Montréal, Canada;
2) I am a graduate of McGill University in Montréal with a Bachelor's degree in Mining Engineering obtained in 1999;
3) I am a registered member of "*Ordre des Ingénieurs du Québec*" (5002252);
4) I have worked as a Mining Engineer continuously since my graduation from university;
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101;
6) I have participated in the preparation of this Technical Report and I am responsible for sections 15 and 16 and part of sections 1, 25 and 26;
7) I have visited the site on June 19th, 2013;
8) I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report;
9) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Lac Otelnuk Mining Ltd., or any associated or affiliated entities;
10) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Lac Otelnuk Mining Ltd., or any associated or affiliated companies;
11) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from Lac Otelnuk Mining Ltd., or any associated or affiliated companies;

12) I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the Report in conformity with generally accepted Canadian mining industry practice, and as of the date of the 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.

This 23rd day of April 2015.

Jeffrey Cassoff, Eng. (signed and sealed)

____

Jeffrey Cassoff, Eng.
Lead Mining Engineer
Met-Chem Canada Inc.

CERTIFICATE OF AUTHOR

To Accompany the Report entitled "Lac Otelnuk Project Feasibility Study - NI 43-101 Technical Report" prepared for Lac Otelnuk Mining Ltd. effective as of March 25th, 2015, issued on April 23rd, 2015.

I, André Boilard, Eng., do hereby certify that:

1) I am Project Manager with Met-Chem Canada with an office at suite 300, 555 René-Lévesque Blvd. West, Montréal, Canada;

2) I am a graduate from "*Université Laval*", Quebec with B.Eng. in Mechanical Engineering in 1977;

3) I am a registered member of "*Ordre des Ingénieurs du Québec*" (31060);

4) I have worked as a Mechanical Engineer and Project Manager continuously since my graduation from university;

5) I have read the definition of "qualified person" set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101;

6) I have participated in the preparation of this Technical Report and am responsible for parts of the sections 1, 2, 3, 4, 5, 6, 24, 25, 26 and 27 and responsible to putting together all the section of the report;

7) I have not visited the project site;

8) I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report;

9) Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Lac Otelnuk Mining Ltd., or any associated or affiliated entities;

10) Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Lac Otelnuk Mining Ltd., or any associated or affiliated companies;

11) Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from Lac Otelnuk Mining Ltd., or any associated or affiliated companies;

12) I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the Report in conformity with generally accepted Canadian mining industry practice, and as of the date of the 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.

This 23rd day of April 2015.

André Boilard, PMP, Eng. (signed and sealed)

____

André Boilard, Eng., PMP,
General Manager, Projects
Met-Chem Canada Inc.

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List of Abbreviations

Abbreviation	Description
µm	Microns, Micrometre
'	Feet
"	Inch
$	Dollar Sign
$/l	Dollar per Liter
$/t	Dollar per Metric Tonne
%	Percent Sign
°	Degree
°C	Degree Celsius
3D	Three Dimensional
A	Ampere
ACSR	Aluminium Conductor Steel-Reinforced
ADI or Adriana	Adriana Resources Inc.
AG	Autogenous Grinding
AIS	Air-Insulated Switchgear
Al_2O_3	Aluminium Oxide
API	American Petroleum Institute
ARD	Acid Rock Drainage
ASL	Above Sea Level
ATV	All-Terrain Vehicles
Avg.	Average
BF	Blast Furnace
BIF	Banded Iron Formation
BOF	Basic Oxygen Furnace
BQ	Drill Core Size (3.65 cm diameter)
BTW	Drill Core Size (4.20 cm diameter)
BWI	Bond Ball Mill Work Index
CAD	Canadian Dollar
CAGR	Compounded Average Growth Rate
CaO	Calcium Oxide
CAPEX	Capital Expenditures
CCR	Central Control Room
CDE	Canadian Development Expenses
CEAA	Canadian Environmental Assessment Act
CEET	Comminution Economic Evaluation Tool
CFR	Cost and Freight
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIS	Commonwealth Independent States

Abbreviation	Description
cm	Centimetre
CNY	Chinese Yuan
DGPS	Differential Global Positioning System
dmt	Dry Metric Tonne
DR	Direct Reduction
DRI	Direct Reduced Iron
DSO	Direct Shipping Ore
DT	Davis Tube
DTM	Digital Terrain Model
DTWR	Davis Tube Weight Recovery
DWT	Deadweight tonnage
EA	Environmental Assessment
EAF	Electric Arc Furnace
EGL	Equivalent Grinding Length
eH	Redox Potential
EIA	Environmental Impact Assessment
EL	Elevation
EMP	Environmental Management Plan
EPCM	Engineering, Procurement and Construction Management
EQA	Environmental Quality Act
ER	Electrical Room
ESIA	Environmental and Social Impact Assessment
FAG	Fully Autogenous Grinding
FBLK	Field-Inserted Blank
FDUP	Field Duplicates
Fe	Iron
FMG	Fortescue Metals Group Ltd.
FOB	Free on Board
g	Gram
G&A	General and Administration
g/l	Gram per Litre
g/t	Gram per Tonne
g/cm^3	Gram per cubic centimeter
Ga	Million Year
GDP	Gross Domestic Product
Gemcom™	Gemcom Software International Inc.

Abbreviation	Description
GESTIM	Public Register of Mining Rights in Quebec
GIS	Gas Insulated Switchgear
GPS	Global Positioning System
H	Head
h	Hour
h/y	Hour per Year
HBI	Hot Briquetted Iron
HDPE	High Density Polyethylene
HF	Hydrofluoric Acid
HPGR	High Pressure Grinding Rolls
HPi	high-pressure indices
HQ	Drill Core Size (6.4 cm Diameter)
HVAC	Heating Ventilation and Air Conditioning
I/O	Input / Output
IBA	Impact Benefit Agreement
ICP	Inductively Coupled Plasma
ID	Identification
IDW	Inverse Distance Method
IDW^2	Inverse Distance Squared Method
IDW^{10}	Inverse Distance to the 10^{th} Method
IOC	Iron Ore Company
IOS	IOS Services Géoscientifiques Inc.
IP	Internet Protocol
IRR	Internal Rate of Return
JBNQA	James Bay and Northern Quebec Agreement
JV	Joint Venture
JVA	Joint Venture Agreement
K_2O	Potassium Oxide
kg	Kilogram
kg/l	Kilogram per Litre
Kg/t	Kilogram per Metric Tonne
$kgDS/m^2h$	Kilogram Dry Solid per square meter hour (Filtration Rate)
km	Kilometre
km/h	Kilometre per Hour
kPa	Kilopascal

Abbreviation	Description
KRG	Kativik Regional Government
kt	Kilotonne
kV	Kilovolt
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per Metric Tonne
LAN	Local Area Network
LCR	Local Control Room
LG	La Grande
LiDAR	Light Detection and Ranging
LIMS	Low Intensity Magnetic Separator
LoI	Letter of Intent
LOI	Loss On Ignition
LoM	Life Of Mine
LOM	Lac Otelnuk Mining Ltd.
LV	Low Voltage
LTZ	Lithotectonic Zone
m	Metre
m/h	Metre per Hour
m^2	Square Metre
m^3	Cubic Metre
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/y	Cubic Metre per Year
MagFe	Magnetic Iron
Makivik	Makivik Corporation
MBIOI	Metal Bulletin Iron Ore Index
MCC	Met-Chem Canada Inc.
MCC	Motor Control Center
MD&A	Management Discussion & Analysis
MDDEP	*Ministère du Développement durable, de l'Environnement et des Parcs*
MENA	Middle East and North Africa
mg/l	Milligram per Litre
MgO	Magnesium Oxide
Micron	Micrometre
min	Minute
min/h	Minute per Hour
mm	Millimetre
Mm^3	Million Cubic Metres

Abbreviation	Description
MMER	Metal Mining Effluent Regulation
MN	Manganese
MOU	Memorandum of Understanding
MPH	Name of a Company from Toronto
MRB	MRB and Associates Inc.
MRC	Midland Research Center
MRNF	Ministère des Ressources Naturelles et de la Faune
Mt	Million Metric Tonnes
Mt/y	Millions of Metric Tonnes per year
MV	Medium Voltage
MVA	Mega Volt-Ampere
MVAR	Mega Volt-Ampere Reactive
MW	Megawatts
N	North
Na_2O	Sodium Oxide, Soda
NEQA	North Eastern Quebec Agreement
NI	National Instrument
Nm^3/h	Normal Cubic Metre per Hour
No.	Number
NPV	Net Present Value
NQ	Drill Core Size (4.8 cm diameter)
NTS	National Topographic System
NW	North West
OPEX	Operating Expenditures
P	Phosphor
PCS	Programmable Control System
PD	Positive Displacement
PDCS	Power Distribution Control System
PDS	Product Delivery System
PEA	Preliminary Economic Assessment
pH	Potential Hydrogen
PLC	Programmable Logic Controllers
ppm	Part per Million
PPP	Purchasing Power Parity
PQ	Drill Core Size (8.5 cm diameter)
QA/QC	Quality Assurance/Quality Control
QC	Quebec Province

Abbreviation	Description
QNS&LR	Quebec North Shore & Labrador Railway
QP	Qualified Person
ROM	Run of Mine
RQD	Rock Quality Index
S	South
S	Sulphur
SAG	Semi-Autogenous Grinding
SCADA	Supervisory Control and Data Acquisition
SE	South East
sec	Second
SEDAR	System for Electronic Document Analysis and Retrieval
SFe_H	Soluble Iron Head
SG	Specific Gravity
SGS or SGS-Lakefield	SGS Lakefield Research Limited of Canada
SIA	Social Impact Assessment
SiO_2	Silica
SIPA	Sept-Îles Port Authorities
SLI	SNC Lavalin Inc.
SMC	SAG Mill Comminution
SNL	SNL Metals & Mining
SPI	SAG Power Index
SPT	Static Pressure Test
SVC	Static VAR Compensation
t	Metric Tonne
t/h	Metric Tonne per Hour
$t/h/m^2$	Metric Tonne per Hour per Square Metre
t/m^3	Metric Tonne per Cubic Metre
t/y	Metric Tonne per Year
TCP/IP	Transmission Control Protocol/Internet Protocol
TFe	Total Iron
TiO2	Titanium Dioxide
TMF	Tailings Management Facility
ton	Short Ton
tonne	Metric Tonne
TSS	Total Suspended Solids

Abbreviation	Description
U.A.E.	United Arab Emirates
U/F	Under Flow
ULC	Underwriters Laboratories of Canada
UPS	Uninterruptible Power Supply
USD	United States Dollar
UTM	Universal Transverse Mercator
W	West
WISCO	Wuhan Iron and Steel (Group) Corp
WHIMS	Wet High Intensity Magnetic Separation
WGM	Watts, Griffis and McOuat Limited
WR	Weight Recovery
WRA	Whole Rock Analysis Method
WSP	Water Supply Pond
WSP	William Sale Partnership
wt	Weight

Abbreviation	Description
w/w	Solid by Weight
X	X Coordinate (E-W)
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
y	Year
Y	Y coordinate (N-S)
Z	Z coordinate (depth or elevation)
ZEC	Zone d’Exploitation Contrôlée
Zn	Zinc

1.0 SUMMARY

1.1 Introduction and Terms of Reference

In September 2005, Adriana Resources Inc. ("Adriana") acquired the right to earn a 100 % interest in claims, notwithstanding certain royalties held by Bedford Resource Partners Inc. ("Bedford") encompassing its Lac Otelnuk Iron Property in the Labrador Trough, Nunavik, Quebec. Additional contiguous claims were staked by Adriana in 2005 through 2013. These claims comprise Adriana's Lac Otelnuk Iron Property (the "Property"). In January 2012 Adriana with a wholly owned subsidiary of WISCO International Resources Development & Investment Limited ("WISCO") formed a joint venture company: Lac Otelnuk Mining Ltd. ("LOM"). Pursuant to this agreement, the Property was transferred into LOM which is held 60 % by WISCO and 40 % by Adriana.

The Property includes an undeveloped surface exposed, gently dipping taconite iron deposit, known as the Lac Otelnuk iron deposit, first recognized and mapped in 1948.

In 2013, Watts, Griffis and McOuat Limited ("WGM") was retained by LOM to provide an updated Mineral Resource Estimate based on all drilling and exploration results to date and document its findings in a Technical Report compliant with NI 43-101 guidelines and standards and Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definitions. The technical report concluded that 20.64 billion tonnes averaging 29.8 % Total Fe head grade at 25.4 % DTWR using a cut-off at 18 % DTWR of resources are contained in the Lac Otelnuk ore body of which 16.21 billion tonnes are classified as measured and 4.43 billion tonnes as indicated. Additionally, 6.84 billion tonnes bearing 29.8 % TFe head grade at 26.3 % DTWR have also been identified and classified as inferred.

In parallel, LOM has mandated SNC Lavalin Inc. ("SLI") to produce a feasibility study on the Lac Otelnuk property based on the assumption that a 50 million tonnes of product per year open pit mine and concentrator operation will be constructed at Lac Otelnuk together with the required tailings disposal works and site infrastructure. Trade-off studies to decide on the final product (pellets or concentrate), product delivery system and power supply type and logistics were part of this Study. A new port facility to be constructed at the Sept-Îles Port, capable of servicing + 400,000 DWT vessels, was also assumed.

Then, in January 2015, Met-Chem Canada Inc. ("Met-Chem") was mandated to assemble the present NI 43-101 compliant Technical Report of the Lake Otelnuk Feasibility Study based on the WGM resource estimate and the SLI study.

All prices and costs in this report are expressed in United States of America dollars (USD or $) unless otherwise specifically stated.

1.2 Property Description and Location

The Lac Otelnuk Iron Property is located in Nunavik, approximately 165 km by air northwest of the village of Schefferville. Schefferville, which lies in Quebec, almost on the border with Labrador, is located approximately 1,200 km by air northeast of Montréal.

The Property consists of 1,398 contiguous mineral claims for a total of approximately 673 km² registered 100 % to LOM. On certain claims a 1.25 % gross revenue royalty in favour of the original claim holder is maintained. The royalty applies to 328 claims aggregating 158.09 km².

Adriana and Bedford executed an Option Agreement November 30th, 2005, following a Memorandum of Understanding ("MOU"), amended by an Amending Agreement dated July 31st, 2006. The agreements provide Adriana with the option to earn a 100 % interest in the original Bedford Lac Otelnuk Property and also define an "Area of Common Interest" surrounding the original Bedford claims.

On January 12th, 2012 Adriana announced that it has successfully closed the Joint Venture Agreement (the "JVA"), with WISCO to engage in the development and operation of Adriana's Lac Otelnuk and December Lake iron ore properties in Nunavik, Quebec (together, the "Lac Otelnuk Project").

Pursuant to the JVA, WISCO provided funding and a proportion of this was injected into a joint venture company, LOM. Adriana has transferred its interest in the Lac Otelnuk Project into LOM. WISCO has acquired a 60 % interest in LOM while Adriana holds the remaining 40 % interest. WISCO has agreed to use commercial best efforts to assist LOM to obtain project financing for 70 % of the development and construction costs for the Lac Otelnuk Project, the size and scope of which will be determined by a bankable Feasibility Study. Adriana and WISCO have agreed to purchase from LOM all the production from the Lac Otelnuk Project at fair market value in proportion to their respective equity interests.

In 2010, Adriana filed an application with the Quebec Superior Court for a judicial interpretation of certain provisions of the Lac Otelnuk Option Agreement. In 2011, the defendants to the application served a plea and cross demand. On August 19th, 2011 the parties entered into a conditional settlement agreement pursuant to which the litigation in the Quebec Superior Court was adjourned pending the satisfaction of the settlement's conditions.

As a result of the closing of the JVA, all the settlement conditions have been satisfied and the litigation is at an end. As part of the settlement, Adriana exercised the option agreement relating to certain claims and all the related titles have been transferred to LOM; half of the royalty in the Lac Otelnuk Option Agreement has been acquired and extinguished leaving a residual 1.25 % gross revenue royalty on certain claims being the original Bedford claims and the claims in the area of Common Interest.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

There is no road access to the Property. Several lakes on the north and south parts of the Property are accessible from Schefferville and Kuujjuaq via fixed-wing float or ski-equipped aircraft. Access to Schefferville and Kuujjuaq is provided by daily scheduled air service from Quebec City, Montreal and Sept-Îles. There is also once-a-week round-trip passenger and freight train service between Schefferville and Sept-Îles.

The village of Caniapiscau, situated about 160 km southwest of the Property, provides an alternative access route. The village is connected by road from Val d'Or via Matagami and Radisson.

The Property area has a sub-Arctic climate with temperatures averaging 12.6 °C in July and -23.2 °C in December. The average annual temperature is -4.7 °C. Average annual rainfall is approximately 355 mm and snowfall 340 cm. Winters are harsh and often lead to poor flying conditions. Exploration programs are normally carried out from June through September.

The Inuit village of Kuujjuaq located north of the Property is the largest nearby community and has a population of approximately 2,200. Kuujjuaq is the administration center of Nunavik and head offices for Makivik Corporation, Kativik Regional Government, Kativik development Council, and Nunavik Board of Health and Social Services. Kuujjuaq has a modern hospital, schools several stores and banking facilities. First Air and Air Inuit provide daily service to the modern airport in Kuujjuaq and charter fixed wing and helicopter services are available. Kuujjuaq and other Inuit villages in Nunavik are a potential source for employment.

Schefferville, in the Province of Quebec, is the closest centre and has a population of approximately 300. The Matimekosh (Montagnais) Indian Reserve is contiguous with the town. The total Schefferville area population, including that of the Kawawachikamach (Naskapi) Indian Reserve, a few kilometers east of Schefferville, is approximately 1,500. The town is served by a reliable source of electricity. A very small pool of skilled labour exists in Schefferville. Several stores, accommodation, restaurant, a health clinic, some services are available, as well as primary and secondary schools.

Schefferville lies at the northern terminus of the Quebec North Shore & Labrador Railway ("QNS&LR"). The town is serviced by once-a-week rail trips to Sept-Îles via Ross Bay Junction (the Wabush corner, 228 km to the south) and on to Sept-Îles, a further 360.5 km.

Topography in the property area is generally flat to gently rolling. A northwest-southeast trending, several km long, 5-10 m high cliff face representing the surface exposure of the iron formation occurs on the northern half of the Property. The elevation varies from 260 to 380 m above sea level. The Property is poorly drained, has extensive swampy areas, and is covered by sparse northern boreal forest consisting of stunted spruce, alders and willows.

1.4 History

The first recorded work on the Property was in 1948 when a Noranda/Conwest joint venture, carried out a regional iron exploration program over a large area, including the Property. The Lac Otelnuk iron formation was recognized and reconnaissance-mapped at that time. No more activity was reported until 1970 when the Property was staked by King, a Canadian subsidiary of a Denver-based company with the same name. MPH of Toronto was engaged to manage the field work, metallurgical test work, "mineral resource" estimates and economic studies, which were carried out between 1970 and 1977. Between 1970 and 1973, 31 vertical diamond drillholes aggregating 1,349 m were completed on the North Zone and in 1976, five (5) drillholes aggregating approximately 308 m were completed on the South Zone. All assay and test work was done at SGS-Lakefield Research of Canada Limited ("SGS-Lakefield"), Lakefield, Ontario.

In 1973, a "mineral resource" estimate for the North Zone was completed. Classified as "open pit" or "mineable reserves", the estimate totalled 613,600,000 long tons grading

25.08 % magnetic iron (33.92 % soluble iron) to a down-dip depth of 125 feet (38 m) vertical and covered by a maximum of 75 feet (23 m) of overburden and/or cap rock. Average thickness of mineralization was estimated at approximately 50 feet (15 m). Using the northern-four of the 1976 holes, a very speculative South Zone "mineral resource" estimate was prepared in 1976. A total of 1,126,600,000 long tons grading 25.76 % magnetic iron (33.06 % soluble iron) was classified as "reserves".

These estimates are regarded by Met-Chem as historical resources and are mentioned in this Report for general interest only.

Adriana's 2007 drilling program was the first ground exploration reported after King's 1976 field program. Adriana's activities are described more fully under Exploration and Drilling sections of this Report.

1.5 Geological Setting and Mineralization

The Property is situated in the Churchill Province, of the Labrador Trough ("Trough") adjacent to Archean basement gneiss.

The Trough, otherwise known as the Labrador-Quebec Fold Belt, extends for more than 1,100 km along the eastern margin of the Superior Craton from Ungava Bay to Lake Pletipi, Quebec. The belt is about 100 km wide in its central part and narrows considerably to the north and south. Adriana's Property is located north of the Grenville Front in the Churchill Province where the Trough rocks have been subject to greenschist or sub-greenschist grade metamorphism and the principal iron formation unit is known as the Sokoman Formation. The lithological units of interest on the Property due to their iron content are members of the Sokoman.

Towards the western edge of the Trough the older, lower units of the sequence are successively exposed as the upper younger units have been removed by erosion. To the northeast, the Sokoman rocks are overlain by the Menihek Formation shales and mudstones. The total thickness of the iron-bearing stratigraphic package, where all is preserved and capped by Menihek, from the top of Unit 2 to top of bottom of Subunit 4b (top of Unit 5) is in the order of 100 m to 120 m.

Within the Property, the structure is very simple with the exception of the far northern portion. The iron formation is generally northwest-southeast striking, very flat-lying, monoclinic to gently inclined and rolling, with an average easterly dip of 5°. The individual members of the sedimentary succession are exposed as a series of benches or mesas in the west-central portion of the north half of the Property.

Within oxide iron formation units, the most distinguishable compositional feature through the local stratigraphic column is the rather abrupt changes from dominantly magnetite to dominantly hematite, and corresponding change of the silica from chert over to jasper. These oxidation potential variations and changes in iron grade define the sub-unit or member lithology units. The iron carbonate minerals, principally siderite and ferro-dolomite, are widespread but are more abundant in the upper and middle iron formation units. These features all appear to be related to primary deposition. Units are named 2, 3 and 4 and subunits are designated with a letter suffix. Unit 5 forms the basal unit and is the Ruth Formation which directly underlies the Sokoman Formation.

1.6 Deposit Types

The Lac Otelnuk deposits are composed of iron formations of the Lake Superior-type. This type of iron formation consists of banded sedimentary rocks composed principally of bands of magnetite and hematite within quartz (chert)-rich rock, with variable amounts of silicate, carbonate and sulphide lithofacies. Such iron formations have been the principal sources of iron throughout the world. Lithofacies that are not highly metamorphosed or altered by weathering are referred to as taconite and the Lac Otelnuk deposits are examples of this type. Mineralization in the iron formation consists mainly of magnetite (Fe_3O_4) and hematite (Fe_2O_3), however, some iron also occurs in siderite and ferro-ankerite. Iron oxide bands containing concentrations of magnetite and/or hematite alternate with grey chert of jasper.

1.7 Exploration

Adriana and LOM exploration programs on the Property started with its 2007 program. Its 2007, 2008, 2010, 2011 and 2012 programs consisted mostly of diamond drilling. In September, 2008, Eagle Mapping Limited ("Eagle") was retained to conduct an aerial photographic survey of the Property. In summer 2010, a reconnaissance geological mapping program covering a part of the north part of the Property was carried out. Adriana also carried out a search for a survey to locate historic drill collars.

1.8 Drilling

The initial aim of the programs was to drill test an area of the South Zone 250 m wide by 9 km along strike with vertical drillholes centred on a 600 m by 500 m grid aligned to the historic MPH cut grid. The first few Adriana drillholes were designed to test the entire iron formation stratigraphy, but most of the 2007 drilling targeted only the upper iron formation Sub-units 2a and 2b. The purpose of the 2008 program was to complete the grid drilling of the designated area of the South Zone and to target the entire Sokoman stratigraphy to the Ruth Formation.

The following Table 1.1 summarizes Adriana and LOM drilling to date.

Table 1.1 – Summary of Adriana and LOM Drilling Programs

Program	Area	Number of Holes	Aggregate Meterage
2007	South Zone	27	2,195
2008	South Zone	41	5,203
2010	South Zone	41	5,874
2011 – Phase I	South and North Zones	29	3,665
2011 - Phase II	South and North Zones	83	11,696
2012	South and North Zones	196	22,249
Total[1]		**414**	**50,229**

Notes: [1] Totals excludes three (3) geotechnical holes drilled in 2011 for water pressure assessment.

For the 2007 program, core size was BTW (4.20 cm) diameter. For the 2008 through 2011 programs, the core size was BQ (3.65 cm diameter). The drilling in 2012 comprised BQ, NQ (4.76 cm) and PQ (8.5 cm) sized holes. All drill and crew moves, except at the very beginning of the 2007 program, were facilitated using a helicopter. All Adriana drillholes were vertical hence no down-hole altitude surveys were completed. Upon completion, drillhole collars were staked with a marker and labelled with an aluminum tag. A certified Land Surveyor conducted DGPS surveys of all drillhole collars, triangulation targets established for aerial photography and other significant surface features.

In 2010, Adriana purchased a second rig and drilling again was focussed on the South Zone. Most of the holes were infill holes on a staggered grid pattern covering the central portion of the South Zone grid.

Drilling in 2011 expanded the drilling area into the North Zone and renamed the area of concentration the Main Zone. The 2011 diamond drill program comprised two (2) phases. Phase II of the 2011 drill campaign was essentially delineation drilling designed to test the extension of the Main Zone on strike to the northwest and southeast over an additional strike length of approximately 26 km.

The 2012 program continued the delineation drilling. The program consisted of 157 BQ, 21 NQ and 18 PQ diamond drillholes. The objectives of the 2012 drill program were to:

- Further expand and upgrade the Lac Otelnuk Mineral Resource;
- Conduct hydrogeology tests and establish hydrogeology monitoring wells in the area of the proposed initial open pit mine;
- Investigate sub-surface soil and bedrock conditions for the proposed tailings facility and;
- Collect large diameter PQ core samples for bench scale and pilot plant metallurgical testing.

Delineation drilling has now been carried out over a total strike length of 35 km. Infill drilling, if warranted or required will be done in subsequent drilling campaigns.

1.9 Sample Preparation, Analysis and Security

LOM and Adriana to date have operated five (5) field drilling programs on the Property (2007, 2008, 2010 and 2011 Phase I and Phase II and 2012) and procedures have remained much the same. Julien Hélou, Geologist, has logged core since Adriana program inception and provided continuity to the process.

Drill core is delivered to the campsite by helicopter on a daily basis where it is unpacked and ordered. Core trays were labelled with aluminum tags denoting drillhole identification and box number. Core logging software has changed through the years but descriptive core logging lithology codes developed by MPH from the King programs have been used in all programs. Core logging included Rock Quality Index ("RQD"), magnetic susceptibility measurements on 0.25 m to 1.0 m intervals down the core and core photography.

Sample intervals are marked on the core by the logging geologists and then recorded in 3-part sample books. The entire iron-rich section of the drill core was sampled leaving no gaps.

Sample lengths were based on geological criteria and sample lengths have averaged approximately four (4) metres. These sample lengths are similar to what was done by MPH for the King programs. The LOM-Adriana protocol included shoulder samples bordering mineralized intervals, and for certain programs, blanks, standards and duplicates. No standards were used during the 2012 program but a number of sample rejects prepared at SGS-Lakefield were sent to a secondary laboratory, Midland Research Center ("MRC") for check assaying). One (1) portion of the 3-part sample tickets are stapled into the core trays at the beginning of each sample interval. Aluminum tags recording the sample identification information were also stapled into the trays accompany the paper tags.

Split core samples were placed into plastic sample bags with the second portion of the 3-part sample tickets and stapled shut. Samples were packed into steel pails and labelled. Samples were sent as batches from the Property by aircraft to Schefferville. From there, the samples went by rail and truck to SGS-Lakefield, Lakefield, Ontario.

Dedicated core storage buildings were constructed at the camp site in 2008 and all historic and Adriana core is stored securely on racks in these two (2) buildings.

WGM made two (2) site visits to the Property to review field program procedures and monitor results. Only one (1) of these visits (September 2007) was made during a period when logging and sampling was in progress. On the basis of their observations, WGM is satisfied that the core handling and core splitting was done to an adequate standard.

Adriana's standard analysis protocol from inception in 2007 through 2010 included Davis Tube tests. For the 2011 program, Davis Tube tests were partially discontinued and replaced by Satmagan determinations. This protocol was continued for the 2012 program. Sulphur content determinations were done on some samples and phosphorous content was also determined on some samples by Inductively Coupled Plasma ("ICP"). Specific gravity on selected pulps was completed using a gas comparison pycnometer. WGM understands that these samples were selected on the basis of trying to be representative of all rock types.

For the 2011 and 2012 programs, Satmagan determinations of magnetic Fe ("MagFe") were completed on most samples. Many were still subjected to Davis Tube tests. A selection of samples had both Davis Tube tests and Satmagan determinations completed. Similar to the previous programs, heads were all analysed by X-Ray Fluorescence spectroscopy ("XRF") for major elements.

WGM is satisfied that sampling and assaying for Adriana's programs since 2007 have been performed well and have been effective but improvements can certainly be made to Adriana's follow-up procedures.

1.10 Data Verification

WGM geologists have made three (3) visits to the Property, but none recently. A former-WGM geologist visited the Property in September 2005 and viewed the Property and historic drill core in storage. Mr. Richard Risto visited the Property from August 28th to

August 31st, 2007 and again from September 13th to 16th, 2008. Mr. Risto's first visit was made during Adriana's first drill program, and drilling and core logging were in progress. At the time of Mr. Risto's 2008 site visit, drilling and core logging was finished for the season.

Some core sampling was still to be completed, but at the time of the visit was in hiatus. WGM reviewed field procedures including core logging and sampling and checked drillhole locations. WGM validated logs and found that sampling records accurately reflected geology and mineralization in the core. WGM recommended more description in the core logs and better qualification of contacts between units. Coordinates for drillhole collars were found to reasonably match existing records.

WGM independently collected second half split core samples on each visit for independent assaying and results of this work validate Adriana's results. The second half core samples independently collected by WGM were "blind" to Adriana and any other of its contractors.

The samples were then sent on to SGS-Lakefield for preparation, assay and test work following the flow sheet for routine samples. Analytical results for original and WGM independent second half core samples were found to correlate well. Results indicate that Adriana sampling is reliable and no sample sequencing errors are apparent. The results also provide a measure of field sampling variance.

The check assaying programs completed at MRC in 2007 and 2008 were both a part of WGM's corroboration work and also were a component of the general QA/QC program.

1.11 Mineral Processing and Metallurgical Testing

The Lac Otelnuk mineralization has been subject to a series of metallurgical testing programs beginning in 1971. This work was documented by WGM in Technical Reports in 2005 and 2009 and these reports are available on SEDAR. The work involved bench scale test work, as well as two (2) pilot plant runs on bulk samples. The reports indicated that the Lac Otelnuk deposit could be processed into a saleable concentrate of approximately 68 % Fe and 4 % silica at a weight recovery of 30 %. Preliminary testing also demonstrated that the concentrate could be pelletized.

Metallurgical test work campaigns by Adriana starting in 2007 utilized drill core to define variations in the metallurgical response based on recognized variations in mineralogy and ore work indices. In April 2011, a PEA was carried out by Met-Chem with a proposed flow sheet based on the metallurgical studies and documented in study entitled "Preliminary Economic Assessment for 50 MTPY – Otelnuk Lake Iron Ore Deposit".

Test work on drill core has continued in 2011, 2012, and 2013 with studies of ore type variability, mineralogy and tailings characterization to further refine the process flow sheet and the requirements to sustain a concentrator operation at Lac Otelnuk. In April 2013, two composites, representing the first 10 and 30 years pit, as well as three PQ composites from the Lac Otelnuk deposit, were prepared for a beneficiation testing program. A final magnetite concentrate quality grading 3.11 % of SiO_2 and 68.8 % Fe on average resulted from the five composites tested. The average overall weight and iron recoveries were 27.5 % and 63.7 % respectively, while the magnetite recovery was 94.7 % on average.

At the end of 2013 a pilot plant was conducted with 80 tonnes of bulk sample in SGS-Lakefield. Five (5) bulk samples, representing the five (5) main ore types from the Lac Otelnuk deposit and totaling about 80 tonnes of material, were received at the SGS-Lakefield facilities to be tested in a grinding and beneficiation pilot plant. The five (5) bulk samples were combined into a single composite for the pilot plant test work. The average overall weight recovery was 24.8 %, while the magnetite recovery was 96.1 % on average.

1.12 Mineral Resources Estimates

1.12.1 Resources Estimates of 2009-2012

a) General

The resources estimates of 2009-2012 were completed by WGM in compliance with the NI 43-101 Standards. However, a Met-Chem's QP has not attempted to verify or classify these estimates. These resources are no longer current and should not be relied upon. They are superseded by the current resources estimate completed by WGM in 2013 that incorporate the most recent data from the 157 delineation holes drilled in 2012. The current resources are documented in an NI 43-101 technical report dated October 31st, 2013.

Met-Chem's QP reviewed the 2013 resources by WGM and prepared a technical note dated March 24, 2014 (Internal document).

b) 2009 Resources Estimate

WGM estimated the resources, using the data from the drill holes completed by Adriana in 2007-2008 that covered approximately 9 km of strike length of the South Zone. The estimate is documented by an NI 43-101 report dated May 7th, 2009.

This estimate defined 4.29 billion tonnes of Indicated Mineral Resources averaging 29.08 % TFe and 1.97 billion tonnes of Inferred Resources averaging 29.24 % TFe.

In November 2010, Met-Chem prepared an NI 43-101 Preliminary Economic Assessment of the Property using WGM's 2009 Mineral Resource estimate.

c) 2011 Resources Estimate

Further drilling of primarily infill holes was conducted on the Property in 2010 and 2011. WGM updated the Mineral Resources including the new data from the 2010 and 2011 (43 holes of Phase I only) drilling program on the South Zone, renamed the Main Zone.

WGM estimated 4.89 billion tonnes of Measured and Indicated Mineral Resources and an additional 1.56 billion tonnes of Inferred Mineral Resources based on a cut-off grade of 18 % Davis Tube Weight Recovery ("DTWR"). No Technical Report was required in support of this Mineral Resource estimate as it was not deemed to be a material change.

d) 2012 Resources Estimate

In 2012, WGM provided an updated Mineral Resource Estimate including the drilling results from the 213 holes (28,232 m) completed from 2007 to 2011 (Phases I & II) and covering approximately 36 km of strike length.

WGM estimated 11.35 billion tonnes of Measured and Indicated Mineral Resources and an additional 12.39 billion tonnes of Inferred Mineral Resources based on a cut-off grade of 18 % DTWR.

WGM documented its estimate in a Technical Report dated August 3rd, 2012 and compliant with the NI 43-101 guidelines and standards.

1.12.2 Current Resources Estimate (2013)

The previous Mineral Resource estimates from 2009 to 2012 are no longer current and should not be relied upon. WGM has prepared a new Mineral Resource estimate for the mineralized zones that have sufficient data to allow for continuity of geology and grades. The current Gemcom™ drillhole database consists of 370 drillholes and covers the same strike length as the 2012 estimate. An additional 157 holes were drilled for the 2013 Mineral Resource estimate and these holes were completed primarily as infill drilling in the north part of the previously defined Mineral Resource area to upgrade the categorization of the resources and to extend the up-dip mineralization to surface along the western margin of the mineralization.

WGM re-modeled the upper geological sub-units of the Lac Otelnuk iron formation that were previously defined (2a, 2b, 2c, 3a and 3b), retaining the transitional 2b-c sub-unit identified for the 2012 estimate. WGM also added an internal shale waste unit north of the old Main Zone, starting at about Line 30 S. This waste unit has become better defined with additional drilling and is more prominent and thicker to the north. It directly underlies sub-unit 2c.

There is some confusion on whether to identify this unit as shale or 3c, so these were used almost interchangeably to define this internal waste unit in the north part of the Property. It is not uncommon for this waste unit to reach thicknesses of 30 to 50 m to the northwest, but it thins and pinches out down-dip and to the east the further south one goes until about Line 30 S where it disappears completely.

The current Mineral Resource estimate was completed using an Inverse Distance to the power of one method. Measured Resources are defined as blocks being within 400 m of a drillhole intercept, Indicated Mineral Resources are defined as blocks from 400 to 600 m from a drillhole intercept and Inferred Mineral Resources are defined as blocks more than 600 m distance from a drillhole intercept and interpolated out to a maximum of approximately 1,000 m where the drilling is more sparse, predominantly in the deeper parts of the deposit.

This categorization was used specifically in the previously named "Main Zone" area of the deposit and directly to the north of this area where more infill drilling was completed during 2012. Mineralization defined by more widely spaced drilling north of Line 270 N has been classified as Indicated and Mineral Resources south of Line 490 S were classified as Inferred, due to even more widely spaced drilling. The deeper intersections of mineralization, predominantly on the northeastern down-dip extension of the deposit, generally lie beneath 70 m or more of cover rock and this mineralization was re-categorized as Inferred.

As with the 2012 Mineral Resource estimate, specific gravities for the 2013 Mineral Resource estimation of tonnage were completed using a variable density model based on the relationship generated by WGM between % TFe and measured densities (pycnometer and bulk density). Significantly more density information was collected during the most recent drilling programs and WGM determined that a variable density model would more accurately define the local variations based on grade than the "per sub-unit basis" used for previous Mineral Resource estimates.

Internally, the continuity of geology/geometry and grade of the sub-units was excellent, however, there appears to be some structural complexity to the northeast of the deposit where possible thrusting has occurred, but it was not followed up from the previous Mineral Resource estimate as this was not the focus of the 2012 drilling program. In general, the recent drilling program was successful in upgrading the categorization of the existing Mineral Resources and expanding the resources where continuity was not certain due to lack of drilling.

A summary of the 2013 Mineral Resources is provided in the Table 1.2 below.

Table 1.2 – 2013 Categorized Mineral Resource Estimate for Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

Resource Classification	**Tonnes (in billions)**	**TFe Head (%)**	**DTWR (%)**	**Magnetic Fe (%)**
Measured	16.21	29.3	25.8	17.8
Indicated	4.43	31.5	24.1	16.7
Total M&I	**20.64**	**29.8**	**25.4**	**17.6**
Inferred	6.84	29.8	26.3	17.8

Notes:
1. Interpretation of the mineralized zones was created as 3D wireframes/solids based on logged geology and a nominal 10 % DTWR when required.
2. Mineral Resources were estimated using a block model with a block size of 50 m x 50 m x 5 m.
3. No grade capping was done. Tonnages and grades reported above are undiluted.
4. Assumed Fe price was US$ 110/dmt.
5. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues.
6. The quantity and grade of reported Inferred Mineral Resources in this estimation 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 upgrading them to an Indicated or Measured Mineral Resource category.
7. The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.

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.

1.13 Mineral Reserve Estimates

The Mineral Reserves are the portion of the Measured and Indicated Mineral Resources that have been identified as being economically extractable and which incorporate mining losses and the addition of waste dilution. The Mineral Reserves for the Lac Otelnuk deposit have been developed using best practices in accordance with CIM guidelines and NI 43-101 reporting.

The first step in the Mineral Reserve estimate is to carry out a pit optimization analysis to determine the parts of the Mineral Resources that are economical to mine. Pit optimization takes place at the start of the study and uses initial assumed operational costs and product selling prices to estimate the economics of mining and processing each block in the model. The pit optimization identifies the limits of the pit and the depth at which the mining costs outweigh the benefits of processing and selling the concentrate.

Since the Lac Otelnuk deposit is close to the surface and contains very little waste rock, the pit optimization analysis showed that the entire resource is economical to mine and to process. It should be noted that pit optimization analyses are strictly based on operating costs and do not consider the capital cost component of a project.

Although the 20,640 Mt of Measured and Indicated Mineral Resources are sufficient for a 105-year mine life at an annual production rate of 50 Mt of concentrate, it was decided early in the Project that the Feasibility Study would be limited to a 30-year mine life. The mine life was limited since a market study cannot be reliably conducted for the period of 105 years. Furthermore, the cash flows generated beyond 30 years have little impact on the Internal Rate of Return ("IRR"), and payback period of a project.

The open pit for the Feasibility Study was therefore designed to contain enough Mineral Reserves which when processed would produce 1,325 Mt of concentrate (30 years of production, considering the phased ramp up).

The location for the 30-year open pit was determined to achieve the following objectives:

- Mine the Mineral Resources that are closest to the concentrator;
- Mine the Mineral Resources that have a low waste-to-ore stripping ratio;
- Mine the Mineral Resources that have a high weight recovery;
- Mine the Mineral Resources that were estimated based on a high density of exploration drilling;
- Avoid disturbing as many water bodies as possible to limit the hydrological footprint of the open pit.

The Mineral Reserves within the 30-year open pit, which account for mining dilution and ore losses, have been estimated to include 4,943 Mt of Proven Mineral Reserves and 50 Mt of Probable Mineral Reserves for a total of 4,993 Mt at an average DTWR of 26.5 %. In order to access these Mineral Reserves, 180 Mt of overburden, 142 Mt of waste rock and 1,052 Mt of low grade material must be mined. This total waste quantity of 1,374 Mt results in a stripping ratio of 0.28 to 1. The low grade material includes all of the Measured and Indicated Mineral Resources within the 30-year open pit that are below the cut-off grade for the Mineral Reserves of 20.65 % DTWR. Table 1.3 presents the Mineral Reserves for the Lac Otelnuk deposit.

Table 1.3 – Mineral Reserves

Category	Tonnage (Mt)	Total Fe Head (%)	DTWR (%)	Magnetic Fe (%)
Proven	4,943	28.7	26.5	18.3
Probable	50	27.5	26.6	18.3
Proven and Probable	4,993	28.7	26.5	18.3

1.14 Mining Methods

The mining method selected for the Project is a conventional open pit consisting of drilling and blasting and a truck and shovel operation with a designed bench height of 15 m. Trees will be cleared prior to the start of mining. Next, a mining contractor will remove the topsoil and overburden using a fleet of dozers, excavators and haul trucks. The ore and waste rock will be drilled, blasted, and then loaded with large mining shovels into a fleet of rigid frame trucks which will haul the material either to the primary crushers, the low grade stockpiles, or the mine rock piles (for the waste rock).

The mining operations for the Project will be 365 days per year, operating around the clock on two (2) 12-hour shifts. A total of five (5) days per year have been accounted for when the mine will be shut down due to severe weather conditions. During these periods, the primary crushers will be fed from the run of mine ore stockpiles using front end wheel loaders.

A mine plan was developed for the 30-year life of the open pit which follows the phased approach of the Feasibility Study, producing 30 Mt/y of concentrate in Phase 1 and 50 Mt/y of concentrate in Phase 2. One of the goals of the mine plan is to ensure that several different ore types are always in production in order to avoid fluctuations in grade and hardness. Blending of the different ores will occur at the primary crushers as trucks arrive from different shovels that are mining the different ore types. Blending will also happen directly at the shovel face since several ore types may be present within a given 15 m high bench.

A 6-month period of pre-production has been included in the mine plan to prepare the pit for operations. During pre-production, 2.1 Mt of overburden will be stripped and four (4) Mt of ore will be stockpiled.

In order to improve the economics of the Project, mining will begin in the southeastern corner of the open pit where there is a considerable area of high grade ore. This high grade zone is limited to the 2a and 2b ore types at the top of the iron formation. In order to ensure the proper blending of ore types, a second area will be developed at the start of the operation on the western side of the deposit, adjacent to the primary crushers. This area will provide the 2c, 3a and 3b ore types. Both of these mining areas will be developed at an even pace for the first seven (7) years of the operation.

Mining will begin in the northern part of the open pit in Year 7 to prepare for Phase 2, when the additional two (2) primary crushers will be installed. During Phase 2, $^{1}/_{3}$ of the production will come from the northern part of the open pit and $^{2}/_{3}$ from the southern part. The purpose of this split in production is to optimize the use of the five (5) primary crushers.

The DTWR throughout the mine plan averages 26.5 % and varies from a high of 33.0 % during year 1 to a low of 25.7 % in year 10. The total material mined ranges from 34.8 Mt in Year 1 to a peak of 260 Mt/y for the years 11 to 15.

During peak production, the total number of 363-tonne haul trucks is expected to reach 50, along with ten (10) cable shovels, two (2) hydraulic shovels, four (4) front end wheel loaders, 16 production drills and a large fleet of support and service equipment. The total mine workforce during peak production is expected to reach 1,002 employees.

1.15 Recovery Methods

The process plant will treat magnetite ore extracted from the Lac Otelnuk open pit mine located in the Nunavik region of the province of Quebec, about midway north in the Labrador Trough iron range, to produce a concentrate. The Project will be developed in two (2) phases: Phase 1 will produce 30 Mt/y of iron ore concentrate and Phase 2 will produce 20 Mt/y to bring the plant to 50 Mt/y capacity. The process plant involves multiple grinding and milling stages followed by a conventional magnetite recovery circuit with desliming thickeners and magnetic separators to produce a pellet feed iron concentrate.

The process plant is designed to treat approximately 188.7 Mt/y of taconite ore with an average magnetite content of 18.2 % and Fe grade of approximately 28.7 % that will permit production of concentrate with a Fe content of about 68 % and less than 4 % silica. Operating 365 days per year, the process plant will recover a nominal 50 Mt/y of pellet feed iron ore concentrate. The plant design is based on a 30-year mine life.

The location and layout of the process plant, shown in Figure 1.1 below, has been chosen to minimize environmental impact, provide a safe working facility to suit subarctic conditions and ensure that no permanent infrastructure interferes with potential mineral resources. The equipment and process have been selected in view of their known and proven technology. The primary crushers and process plant have been designed on the basis of 70 % and 90 % availability respectively and integrate operability, maintainability, and constructability elements.

Figure 1.1 – Plant Location and Layout

Careful consideration has been given to natural topography and prevailing wind directions to minimize surface preparations (earthworks), facilitate building and system erection and construction during harsh weather conditions and limit dust carry-over from tailings and mine site, snow accumulation and excessive cold air infiltrations. The distance between the living camp and work areas have been minimized while ensuring adequate safety and providing adequate nuisance (noise, dust, etc.) separation. Material handling of ore has been optimized to minimize length of conveyors, number of transfer towers and maximize energy conservation.

The location of main equipment inside the plant has been done to minimize pumping requirement and make maximum use of gravity for slurry flows. The plant complex has also been designed such as to avoid unnecessary circulation and minimize ground use.

The process flow sheets, refer to Figure 17.4 and Figure 17.5 in Section 17.0, were developed on the bench-scale test work results and complemented by supplier tests for equipment sizing. The bench-scale test work was performed on the 30 Y composite samples since these samples are representative of the average feed from the mine plan. The SAG process design has been defined with the pilot plant test results. The process plant is designed to treat approximately 188.7 Mt/y of taconite ore with an average magnetite content of 18.2 % and Fe grade of approximately 28.7 % that will permit production of concentrate with a Fe content of about 68 % and less than 4 % silica.

From the mine, the ore is first crushed by five (5) large-capacity gyratory crushers (3 in Phase 1 + 2 in Phase 2) located at a minimum of approximately 0.5 km from the mine pit for safety reasons. The crushed ore is then process plant stockpiles using overland conveyors. At the process plant, the overland conveyors discharge the ore onto belt tripping conveyors installed on the top of the plant stockpile designed for 16 hours (equivalent to approximately 380,000 t for both phases combined) of live ore storage at nominal capacity. Ore reclaiming is achieved through an enclosed steel structure at the bottom of the stockpile housing three (3) apron feeders for each ore-reclaiming lines installed in individual tunnels. At the exit of each ore reclaiming tunnel, a main ore belt conveyor is installed in a gallery that extends into the process plant and discharges ore into its respective SAG mill feed hopper.

The concentration plant consists of five (5) independent ore processing trains (3 in Phase 1 + 2 in Phase 2). Each train produces nominally 10 Mt/y of iron ore concentrate (dry basis).

Each train consists of three (3) SAG mills, one (1) common steel ball feeding system for all SAG mills, vibrating screens, oversized recirculation conveyors, three (3) hydrocyclones, two (2) first-stage ball mills, steel-ball feeding system for all first-stage ball mills, one (1) second-stage ball mill, steel ball feeding system for all second-stage ball mills, three (3) stages of magnetic separators (cobber, rougher, and cleaner LIMS), three (3) hydro-separators (desliming thickeners), two (2) concentrate thickeners, one (1) tailings thickener, coarse tailings hydrocyclone, process water pond and pumping station. Overhead cranes are provided inside the plant building to handle pieces of equipment and for removal of liners and for handling tools.

The concentrate, at 67 % solid content will be pumped to slurry concentrate storage tanks, then through the Product Delivery System ("PDS"), one common (PDS) for all three (3) Phase 1 process trains and another PDS for the two (2) additional Phase 2 trains, will be transported from the concentrator to the port facilities located in Port of Sept-Îles, where the slurry will be filtered and ship loaded.

For a more detailed description of the processing plant, please refer to Section 17.0.

1.16 Project Infrastructure

The necessary infrastructure and logistic requirements for the Project have been considered. They include power transmission and distribution, access roads, communication and automation systems, support infrastructures and utilities, camp site accommodations, airstrip, tailings management, dams, water management, stockpiles, concentrate transport (PDS) and port facilities, required for the Lac Otelnuk deposit located in the Nunavik region of the province of Quebec, about midway north in the Labrador Trough iron range.

It is expected that three (3) incoming power transmission lines will be supplied at each of the main Project locations these being the mine and process plant, PDS pump station PS3 and the Port Facilities in Sept-Îles.

The main access road linking the process plant site to the town of Schefferville, Quebec is designed for delivery of consumables during operations as well as to provide access during construction for delivery of construction material and equipment. The road is approximately 185 km long, 10.3 m wide (including shoulders), and constructed of granular materials or crushed rocks suitable for traffic travelling at a nominal speed of up to 60 km/h. The plant and secondary roads are 8 m and 4 m wide respectively, and constructed of granular materials suitable for traffic traveling at a nominal speed of up to 30 km/h. The mine haul-truck roads required for off-highway mine 400-ton (short ton) haul trucks will be designed to be 30 m wide, and to feature a 2 m high rock safety berm on each side.

Blasting will be carried out with bulk emulsion which will be manufactured in a facility that will be built and operated on site by a properly licensed explosives supplier. The explosives plant and the magazines to store the accessories will be located at the south end of the open pit.

The telecommunication systems will need to be available 24 hours per day, 365 days per year. All active core components to be installed in each telecommunication systems will be designed in order to achieve 99.999% availability.

Diesel, gasoline and jet fuel will be delivered to Schefferville by train and then transferred to tanker trucks and transported to the plant site where it will be stored in dedicated storage tanks.

Electrical steam boilers are used for heating the process plant, workshops and the warehouse. Eleven boiler trains for Phase 1 and five (5) for Phase 2 are necessary to fulfill the heating requirements of the buildings.

A central compressed air station provides all the necessary compressed air for the plant air distribution system adjacent to the boiler building.

The maintenance workshop is used for the maintenance of small vehicles, rubber lining, and process equipment and can be accessed by utilidors. The mine equipment maintenance complex includes all necessary facilities and equipment required for mine equipment maintenance. It includes several mine truck wash bay stations, repair stations, welding shop, tire press shop, as well as a storage and a handling systems for all new and used oil and lubricants. A main warehouse used for storage for consumables, spare parts, tools is located next to the maintenance workshop building.

The administration complex located adjacent to the process plant substation and next to the SAG mills area will house all management offices, meeting and control room on a two (2) story building. A laboratory, required for the process plant and environmental analysis, is included within the same complex.

The domestic waste water treatment plant will be built in modular units and located to the north of the permanent camp. Waste water from the process plant, the camp, and the maintenance areas is collected and transferred to the waste water treatment plant.

Non-hazardous solid waste from the mine, the process plant, and the accommodation complexes and the residue from the wastewater treatment plant are collected by truck and sent to waste disposal cells.

A potable water treatment plant will be built using modular units each consisting of two (2) pre-assembled skids. The source of potable water at the process plant is from the Water

Supply Pond ("WSP"). Water treated by the potable water system is first pre-filtered by the raw water filtration package.

A portion of the water from the raw water filtration plant is sent to the firewater tank. Three (3) firewater pumps are connected to the firewater network: one (1) jockey pump and two (2) pumps (one electrical and one diesel) which supply water to hydrants, sprinkler systems, and other fire protection systems.

Acting as a waste management facility on site, a landfill will receive trash, sludge, and garbage which consist mostly of food scraps.

The permanent accommodations are located approximately 0.5 km from the process plant. They consist of: living units (dormitories), kitchen / dining room / recreational facilities building, gym and fitness center, and security / first aid / emergency services building.

The emergency and security building is a single level building consisting of offices and an emergency response centre. The security building includes a reception area, arrival and departure areas with luggage storage and distribution facilities, medical facilities fitted with a nurses'/doctor's office, examination rooms, and washroom and a helipad for emergency evacuation.

The kitchen / dining room / recreational facility has two (2) floors, with the kitchen and dining area on the ground floor, and recreational facilities on the first floor. The kitchen has a capacity of serving approximately 500 meals/hour. The dining area can accommodate a minimum of 750 persons in one seating, with additional space planned for vending machines, lunch room area, coat room, and washrooms for men and women.

The living modules (dormitories) are arranged in nine (9) separate wings of three (3) floors each. Altogether, the complex comprises 1,134 one-person rooms, each with a surface area of approximately 17.4 m^2.

The aerodrome area includes a pre-fabricated terminal building, maintenance garage, helicopter pad, and an apron area. The buildings will be heated by an electrical heating system. The apron sizing is designed for a Boeing B737-200C.

The final tailings footprint covers an area of approximately 45 to 50 km^2 and is shown in Figure 18.13. The average (struck level) tailings surface at about elevation 353 m provides the required tailings storage volume of about 2,300 Mm^3. This results in a maximum tailings thickness at the end of operations of about 150 m at the West Dam.

The product delivery system refers to a pair of slurry pipelines (one with a 30 Mt/y capacity for Phase 1 and the other having a 20 Mt/y capacity for Phase 2) required to bring iron ore concentrate in the form of a slurry from the Lac Otelnuk Iron Ore Project mine site to the Pointe Noire Terminal at the Port of Sept-Îles. The pipeline route is approximately 755 km and the pipelines are mostly shallow buried except when crossing water bodies, where they are on piles or on top of culverts for small crossings. The design has adopted a "No-Freezing, No-Plugging philosophy", which means that the slurry transportation system will have the necessary reliability and emergency back-up equipment to maintain no-freeze conditions during an emergency stoppage and the slurry transportation system slope will be restricted to avoid plugs even if the line is shut down full of slurry.

The concept for the port area which includes product dewatering, storage, reclaiming, and shipping of the concentrate is comprised of a product dewatering facility for dewatering of the slurry in order to achieve a dry concentrate at 8 % moisture content, transferred to the concentrate storage building, and ultimately reclaimed and conveyed to the ship loader on the wharf to be constructed and operated by the Sept-Îles Port Authorities ("SIPA").

The shipping concept for the dry concentrate is based on the use of the future Phase II deep-water wharf at the Port of Sept-Îles for the future potential use by LOM where the ship loaders will be installed. The installation is designed to load bulk carriers ranging from 180,000 DWT to 400,000 DWT in capacity.

1.17 Market Studies

Lac Otelnuk Mining Ltd. has requested a review of the iron ore market in general and more specifically for high grade iron ore concentrates until the year 2050.

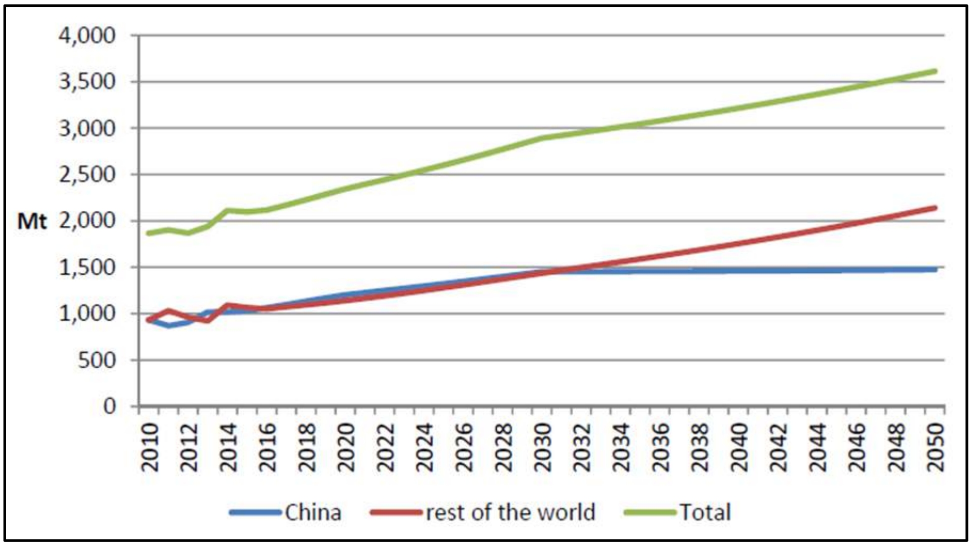
SNL Metals & Minings ("SNL") demand projections for iron ore give particular emphasis to expected Chinese developments, since China accounts for such a large portion of world demand and since its share is still growing. The projection of Chinese demand takes as a starting point the reduction of the share of investment in Chinese GDP from about 55 % now to 41 % in 2020, 30 % in 2030 and 25 % in 2040. We also assume that the overall rate of growth in Chinese GDP will fall from its very high present level to an annual rate of 7 % from 2014 to 2015 and that it will fall further to 6 % from 2016 to 2020, 4 % from 2021 to 2030 and 3 % from 2031 to 2050. With respect to the rest of the world, the assumptions are conservative and the growth rates are generally lower than the ones achieved historically.

Under these assumptions, world iron ore demand will reach 2,098 Mt in 2015, 2,345 Mt in 2020, 2,601 Mt in 2025 and 2,892 Mt in 2030 (refer to Figure 1.2 below). The average annual increase over the entire period 2013 to 2030 is 2.5 %, which is below the rate of growth achieved in the early 2000s, but higher than the growth rates in the 1980s and 1990s.

According to this projection, some 233 Mt of production would need to be added until 2020 and about 256 Mt more from 2020 to 2025. These are impressive figures, in spite of being derived from very conservative assumptions regarding overall world economic development. From 2000 to 2007, output rose by 765 Mt, or slightly more in terms of annual additions than what we are projecting for the period 2014 to 2020.

The "Big 4" (Vale, BHP Billiton, Rio Tinto and FMG) will add 229 Mt of capacity in 2014 and 2015. Other Australian and Brazilian producers are expected to add about 40 Mt. Projects elsewhere in the world may add another 20-40 Mt, for a total of about 290-310 Mt. Compared to the 150 Mt of additional demand that SNL expect to see, this clearly would point to a massive oversupply developing. For later periods, the number of confirmed projects is much lower and the large producers are reportedly reducing their planned capital expenditure.

Figure 1.2 – Actual and Projected World Iron Ore Demand 2010-2050, Mt



Source: SNL Metals & Mining

Despite the cancelling of many projects, the mismatch between planned capacity additions and expected demand will have to be resolved by some producers leaving the market.

Chinese iron ore mines have assumed the role of swing producers and are likely to remain in this position. A large part of the Chinese iron ore industry had to shut down following the slower demand growth and lower prices in 2008. The mines used to be protected by high freight rates but as freight rates came down and production costs went up, only the unexpectedly strong recovery of the Chinese steel industry and the consequent high prices for iron ore have saved them. With the prospects for steel production in China now looking decidedly less positive, the time of reckoning has clearly come.

Chinese ore production fell from 344 Mt (gross run of mine production converted to comparable grades) in 2010 to 269 Mt in 2013. Thus, about 80 Mts of capacity had already disappeared at the beginning of 2014. Reports of further closures have surfaced in early 2015 and it remains to be seen how many mines stay closed after the cold months of January and February.

We believe that Chinese iron ore production will decline by a further 100-150 Mt to about 120-170 Mt in 2020. In September 2014, iron ore prices have just fallen below 80 USD/dry tonne CFR China. In view of the very large additions to supply in the pipeline, prices are unlikely to rise more than marginally over the next couple of years. This means that the decline in Chinese production could take place during this year and next.

Even with the expected Chinese production decline, however, the next couple of years are likely to be characterized by lower prices than producers have become accustomed to over the past five (5) years. Hundred (100) USD per dry tonne is likely to become a ceiling rather than a floor for prices.

During the period 2015 to 2020, the market would be expected to rebalance at a higher price level. The reductions in capital expenditure by the large producers together with cancellations of projects by other producers will serve to eliminate the excess supply. However, a number of projects will be ready to come on stream at relatively short notice, so any price upturn is unlikely to be of more than short duration.

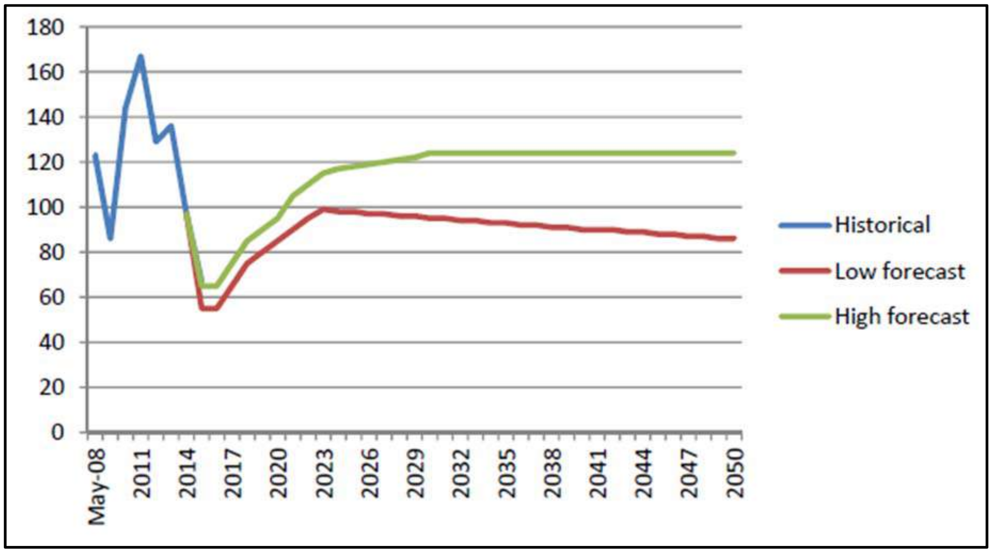
From 2020 to 2030, the need for capacity additions will remain and, given the experience of the complications associated with increasing capacity in Australia and Brazil at present, it is possible that these regions will find it difficult to grow production further. As a result, the 2020s may see the emergence of Africa as a major producing region. Many of the projects now being planned in Africa will not become fully operational until the early 2020s, but they will be followed by others. However the growth seen in the last couple of years in iron ore production from Africa have been stalled as most new producers are in administration or is struggling with financing.

We believe that the Chinese market will be the main outlet for Lac Otelnuk concentrate, with some of it going to blast furnace pellets and some to Direct Reduction ("DR") pellets. North America and the Middle East are the other two (2) possibly important destinations, where new Direct Reduction Iron ("DRI") plants will provide good opportunities for high grade Lac Otelnuk concentrate. The European market will in all likelihood be of only marginal importance. As for prices, SNL believe that it is appropriate to assume that Lac Otelnuk concentrate will attract a premium.

The factors that have been described appear to support the following scenario in four (4) stages for iron ore prices (see Figure 1.3):

1. Over the next three (3) years, prices will be determined by the oversupply with a floor being set by the need to ensure that enough mines break even.
2. From about 2017 to 2020, the situation will resemble that of the 1980s and 1990s, when production increased in an orderly fashion and iron ore price movements were moderate. While price spikes may occur temporarily, particularly if producers encounter unexpected problems with capacity additions, prices will rise somewhat at the beginning of the period and then remain more or less constant.
3. From 2020 to 2030, prices will be set by costs in new projects and could be expected to decline gradually as a function of productivity improvements. However, increases in costs resulting from significantly higher energy prices and/or rising extraction costs due to the need to mine lower grade ore bodies could set a floor for this decline.
4. From 2030 to 2050, productivity improvements are expected to lead to continuing declines in prices, to some extent offset by rising extraction costs in new mines.

Figure 1.3 – Prices for 62 % Fe Iron Ore Delivered in China, USD/Dry Tonne



Source: Metal Bulletin (historical prices), NSL forecast

Table 1.4 shows the result of the netback calculations for Lac Otelnuk. The calculations are based on assumptions for freight rates that are somewhat higher than those prevailing today. It also includes a Fe grade premium which is based on historical values.

Table 1.4 – Netback Calculations for Lac Otelnuk

	2015	2020	2025	2030	2040	2050
Low, MBIOI 62 %	55	85	98	95	90	86
High, MBIOI 62 %	65	95	118	124	124	124
Theoretical comparator values						
Low						
Alexandria	43.47	73.47	86.47	83.47	78.47	74.47
New Orleans	41.62	71.62	84.62	81.62	76.62	72.62
High						
Alexandria	53.47	83.47	106.47	112.47	112.47	112.47
New Orleans	51.62	81.62	104.62	110.62	110.62	110.62
Lac Otelnuk						
Freight Pointe Noire-Alexandria, $/t wet	11.46	11.46	11.46	11.46	11.46	11.46
Freight Pointe Noire-New Orleans, $/t wet	7.35	7.35	7.35	7.35	7.35	7.35
Freight Pointe Noire-Qingdao, $/t wet	26.17	26.17	26.17	26.17	26.17	26.17
Humidity	8 %	8 %	8 %	8 %	8 %	8 %
Freight Pointe Noire-Alexandria, $/t dry	12.45	12.45	12.45	12.45	12.45	12.45
Freight Pointe Noire-New Orleans, $/t dry	7.99	7.99	7.99	7.99	7.99	7.99
Freight Pointe Noire-Qingdao, $/t dry	28.45	28.45	28.45	28.45	28.45	28.45
Netback Lac Otelnuk						
Fe grade premium, $/dry tonne	26	26	26	26	26	26
Low price alternative, FOB Pointe Noire						
Pointe Noire-Alexandria	57.02	87.02	100.02	97.02	92.02	88.02
Pointe Noire-New Orleans	59.63	89.63	102.63	99.63	94.63	90.63
Pointe Noire-Qingdao	52.55	82.55	95.55	92.55	87.55	83.55
High price alternative, FOB Pointe Noire						
Pointe Noire-Alexandria	67.02	97.02	120.02	126.02	126.01	126.01
Pointe Noire-New Orleans	69.63	99.63	122.63	128.63	128.63	128.63
Pointe Noire-Qingdao	62.55	92.55	115.55	121.55	121.55	121.55

1.18 Environmental Studies, Permitting and Social or Community Impact

Golder Associates was retained by LOM to review the existing government reports, data bases and publications and to complete the various engineering and environmental studies in support of the Environmental Impact Assessment ("EIA") and Feasibility Study and for the mine site. In addition a baseline study was conducted by AECOM for the mine site access trail corridor from Schefferville to the proposed mine site. No other environmental baseline studies have been conducted yet on other potential components of the Project such as a power line, a slurry pipeline, access roads and an airstrip.

Table 1.5 summarizes the environmental baseline studies and fieldwork that has already been completed on the proposed mine site and site access trail. This environmental baseline information serves as the basis for preparing the EIA that will be required.

Table 1.5 – Summary of the Environmental Baseline Studies and Fieldwork Completed on the Proposed Mine Site and Site Access Trail

Environmental Sector	Baseline Field Work and Studies
Climate Data	LOM has operated a meteorological station since July 2010 with Annual Climate reports prepared by Golder: Annual reporting (years 2010, 2011, 2012, 2013) of recorded climatic data, Golder (2013); Weather Report 2013, Golder (2013); Weather Report 2011-2012, Golder (2013).
Physical Environment	General description, Golder (2011); Geomorphology – Field Report 2012, Golder (2013); Baseline Study – Geomorphology and Soils – 2012 Fieldwork Report, Golder (2013).
Hydrogeology	Ground Water Sampling at LOM Site – Technical Memorandum – Fieldwork Result Compilation, Golder (2010); Initial Hydrogeology Study, Golder (2012); Complementary Hydrogeology Study – Summer 2012, Golder (2013).
Hydrology	Hydrology Field Report – 2010 to 2012, Golder (2013).
Baseline Water Quality	Water Quality Surface Water and Sediments Baseline – Fieldwork, Golder (2013).
Vegetation and Wetlands	Vegetation and Wetlands – Baseline Study, Golder (2013).
Bathymetry	Technical Memorandum – Release of Bathymetry Data, Golder (2013).
Fish Habitat	Fish Habitat Baseline Study – Field Report 2011 and 2012, Golder (2013); Fish Inventory Baseline Study – Field Report, Golder (2013).
Wildlife	Baseline Study – "Inventaire de la Sauvagine" – Field Report 2011, Golder (2013); Baseline Study – Beavers – Field Report 2011, Golder (2013); Baseline Report – Upland Breeding Birds, Amphibian and Semi-Aquatic Mammals Surveys Golder (2013); Baseline Report – Aerial Survey of Ungulates, Golder (2013); Baseline Study – "Avifaune Nicheuse" (French), Golder (2013).
Mine Access Trail Corridor	Baseline Study and Field Work Report, AECOM (2014).

The following, is a summary of the proposed environmental baseline studies conducted as of January 2015 and available public information review for the proposed mine site and associated infrastructure:

- A meteorological station has been in operation at the Lac Otelnuk exploration site since July 2010.
- The watercourses within the proposed mine site are located in the Caniapiscau River watershed which flows north up into the Ungava Bay. The Project site is located at the

edge of two (2) sub-watersheds of which one (1) flows through a series of lakes (Alpha Lake, Delta Lake, Lace Lake, du Gouffre Lake) into the Caniapiscau River. The other flows into the Swampy Bay River, upstream of Otelnuk Lake and the Hautes Chutes then into the Caniapiscau River approximately 100 km upstream.

- The Project is located in the territory regulated under the James Bay and Northern Quebec Agreement ("JBNQA"). The JBNQA establishes an environmental protection regime which dictates specific social and environmental impact assessment processes from which the Environmental and Social Impact Assessment ("ESIA") must be elaborated. Chapter II of the Environment Quality Act ("EQA") integrates provincial requirements concerning the impact assessment provided in Chapter 23 of the JBNQA.
- The proposed mining infrastructure is located in a sporadic discontinuous permafrost area (surface cover between 10 and 50 %) according to the Canada Atlas of Natural Resources Canada (1993). However, no permafrost was observed during preliminary summer field work conducted in 2012 and 2013.
- The proposed mine site is located in the spruce-lichen bioclimatic domain within the boreal taiga subzone, which extends between the 52^{nd} and 55^{th} parallels (Saucier et al., 2003) in Quebec and is characterized by low density forest stands.
- No observations of plant species at risk were recorded within a 20 km radius from the Project site. Two (2) occurrences of special status plant species was observed during the summer of 2012 and 2013 rare plant surveys; the rock sedge (C*arex petricosa var. Misandroides*) and Nahanni oak fern (*Gymnocarpium jessoense subsp. parvulum*), two (2) species likely to be designated at risk or vulnerable under the provincial Act Respecting Species at Risk or Vulnerable Species.
- According to public information, observation of a golden eagle (*Aquila chrysaetos*) was recorded near the Project site. In addition, observations of the peregrine falcon (*Falco peregrinus*), the golden eagle, the harlequin duck (*Histrionicus histrionicus*), the bald eagle (*Haliaeetus leucocephalus*), and the short-eared owl (*Asio flammeus*) were recorded near the Project site. If suitable habitats are present in the Project site, these species could be present. Field investigations conducted in 2011, 2012 and 2013 for birds and six (6) bird species at risk were identified within the Project site: the rusty blackbird (*Euphagus carolinus*), the harlequin duck, the bald eagle, the golden eagle, the peregrine falcon, and the barrow's goldeneye (*Bucephala islandica*).
- No observations or records of mammals, amphibians or fish species at risk were recorded at the Project site either in the database or during the field surveys conducted to date.

The projected mining infrastructure is located in a sporadic discontinuous permafrost area (surface cover between 10% and 50%). The mine site is located in the spruce-lichen bioclimatic domain within the boreal taiga subzone, which is characterized by low density stands. Wildlife and fish observed in the Project area is typical of northern environments.

There are no protected areas that are directly included in the Project site. The Collines-Ondulées Provincial Park, located approximately 50 km southeast of the Project site is the only legally protected area present within a 50 km radius of the Project site. However, there are two (2) reserved lands for potential provincial parks that are located in this radius:

- The Lac-Cambrian territory, approximately 35 km northwest of the Project site;
- The Canyon-Eaton territory, approximately 25 km south of the Project site.

Land tenure and organization in the Project site is governed under JBNQA. The Project is situated in the "*Territoire non-organisé Rivière-Koksoak*" (Koksoak river unorganized territory) which is administrated by the Kativik Regional Government ("KRG"). The head office of the KRG is located in the northern village of Kuujjuaq.

The Project site is uninhabited. The closest inhabited areas are:

- The Indian Reserve of Kawawachikamach, located about 155 km south-east of the Project site and inhabited by the Naskapi Nation of Kawawachikamach;
- The town of Schefferville, about 165 km south-east of the Project site;
- The Indian Reserve of Matimekosh and Lac-John, located about 167 km south-east of the Project site and inhabited by the Matimekush-Lac John Innu Nation;
- The Northern Village of Kuujjuaq, about 230 km north of the Project site.

The only access to the Project site is by air or snowmobile in winter. Therefore, to William Sale Parnership ("WSP") knowledge, there is no intensive land use by the communities, which are all located more than 150 km away. The closest railway link is Schefferville to Sept-Îles via, Wabush and Labrador City (Tshiuetin Rail Transportation and Quebec North Shore and Labrador Railway). The closest airport is located in Schefferville.

Engaging with communities and stakeholders is a key approach as the Project progresses. As a first step, the Project promoter initiated a series of information and consultation sessions for representatives of the three (3) Aboriginal communities, that is, the Inuit in Kuujjuaq, the Innus in Matimekosh-Lac John, and the Naskapis in Kawawachikamach. In the same year, information and consultation sessions were led by Golder in Kuujjuaq for institutional stakeholders and the Kuujjuaq local community.

Consultation is a 2-way process of dialogue between the proponent and communities and stakeholders. The consultation is really about initiating and sustaining constructive relationships over time. As the ESIA progresses, other consultation sessions will be done according to the consultation plan.

The process of performing the environmental and social impact assessment is only at its early stage regarding the Lac Otelnuk Project. WSP understanding of the study area is progressing and the initiation of the ESIA has started with the identification of the initial features and issues, as they will influence the potential positive and negative impacts. As the environmental evaluation process progresses, these features and issues, as well as the potential impacts, will be validated.

1.19 Capital and Operating Costs

1.19.1 Capital Cost

The capital cost estimate consists of the direct, indirect and owner's costs for the mine site area (including the power transmission lines), the product delivery system and the port area. Provisions for sustaining capital are also included, mainly for mining equipment replacement and tailings storage expansion. Amounts for closure and rehabilitation of the site and required working capital have been included as well.

The Project will be developed in two (2) distinct construction phases: Phase 1 sized for 30 Mt/y (3 process trains); Phase 2 sized for additional 20 Mt/y (2 process trains). Each phase will be comprised of a multiple of 10 Mt/y process train.

Table 1.6 presents the summary of the capital cost for the major area of the Project.

Table 1.6 – Capital Cost Summary (December 1st, 2014)

AREA	Total USD $M	Phase 1 USD $M	Phase 2 USD $M
Direct Costs			
Mine/ROM	704	486	218
Process	3,325	1,998	1,327
Infrastructure &Tailings	1,037	758	279
Power Transmission and Distribution	898	830	68
Product Delivery System	4,396	2,622	1,774
Port Area	703	520	183
Sub-Total Direct Costs	**11,063**	**7,214**	**3,849**
Indirect Costs			
Construction Indirects	324	203	121
Freight, Spares, First Fills & Heavy Lift	399	247	152
Camp, Catering and Travel	379	287	92
EPCM	750	500	250
Contingency	700	458	242
Sub-Total Indirect Costs	**2,552**	**1,695**	**857**
Other Costs			
Owner's Costs	276	180	96
Power Line Extension to Port	63	63	0
Power to PDS Pumping Stations 2 & 3	232	232	0
Sub-Total	**571**	**475**	**96**
Total CAPEX	**14,186**	**9,384**	**4,802**

1.19.2 Operating Costs

The operating costs have been estimated for the mine site facilities, the product delivery system and the port area. They were estimated as a function of the iron ore concentrate production and were developed for each production year up to the 30-year mine life.

Table 1.7 presents the summary of the operational costs for Year-3 of Phase 1 when the production capacity will reach 30 Mt/y and also Year-9 of Phase 2 when the production capacity will reach 50 Mt/y.

Table 1.7 – Operating Costs Summary (December 1st, 2014)

AREA	Phase 1 Year-3 (30 Mt/y)		Phase 2 Year-9 (50 Mt/y)	
	Total USD	USD per tonne concentrate	Total USD	USD per tonne concentrate
Mine Site	959,459,453	31.98	1,444,946,940	28.90
Product Delivery System	43,984,853	1.46	71,608,409	1.43
Port Area	24,138,432	0.80	38,886,823	0.78
Total	**1,027,582,737**	**34.21**	**1,555,442,171**	**31.12**

1.20 Economic Analysis

An economic analysis based on the production and cost parameters of the Project has been carried out and the results are shown in Table 1.8.

Table 1.8 – Summary of the Life of Project Production, Revenues and Costs

Description	Units	Results
Production – Mineralization	Mt	4,993
Production – Concentrate @ 68.5 % Fe	Mt	139,123.4
Revenue	USD $M	138,878.8
Capital Costs	USD $M	14,186.4
Operating Costs	USD $M	42,360.8
Pre Tax Cash Flow	USD $M	79,109.4
After Tax Cash Flow	USD $M	47,673.5

For the base case, with a 100% equity funded project, variable yearly selling prices including premium based on SNL’s market study and all production being sold to Chinese market have been assumed. The analysis of these estimates returned the financial indicators summarized in Table 1.9.

Table 1.9 – Summary of Financial Indicators – Base Case

	Before Taxes	After Taxes
Project IRR*	15.8 %	13.0 %
NPV @ 6 %*	$ 17,457 million	$ 9,647 million
NPV @ 8 %*	$ 10,388 million	$ 5,240 million
NPV @ 10 %*	$ 5,906 million	$ 2,440 million
Payback Period**	7.0 years	7.3 years

* Based on Free Cash Flow to Equity.
** Calculated from start of commercial production and based on Free Cash flow to Equity.
All monetary values in USD, unless otherwise stated.

Figure 1.4 shows the sensitivity of the NPV and IRR, respectively, for variations in initial and sustaining Capital Costs, Operating Costs and Selling Price.

Figure 1.4 – Sensitivity of Project IRR (After Tax) – Base Case

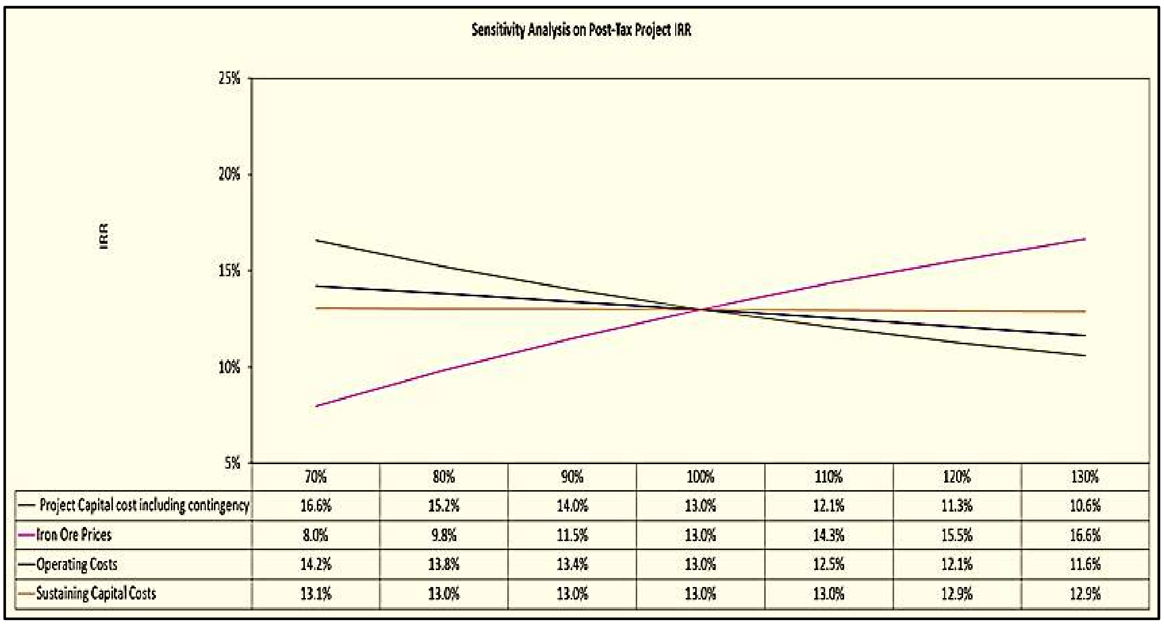
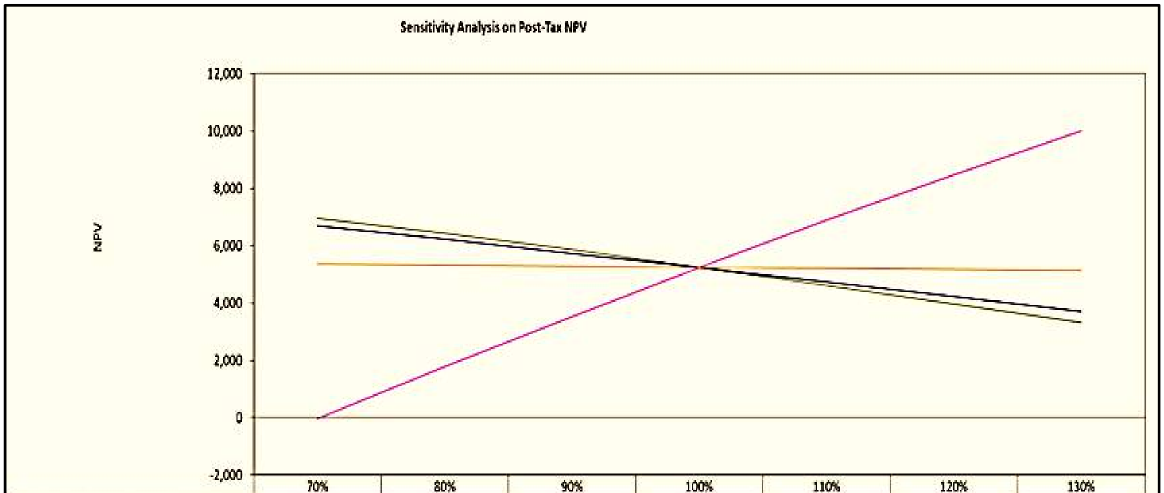


Figure 1.5 – Sensitivity of Project NPV (after tax) to Key Parameters (Base Case)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	6,955	6,430	5,858	5,241	4,608	3,955	3,315
Iron ore Prices	-32	1,785	3,526	5,241	6,898	8,465	9,997
Operating Costs	6,694	6,215	5,729	5,241	4,736	4,215	3,689
Sustaining Capital Costs	5,348	5,312	5,276	5,241	5,205	5,169	5,133

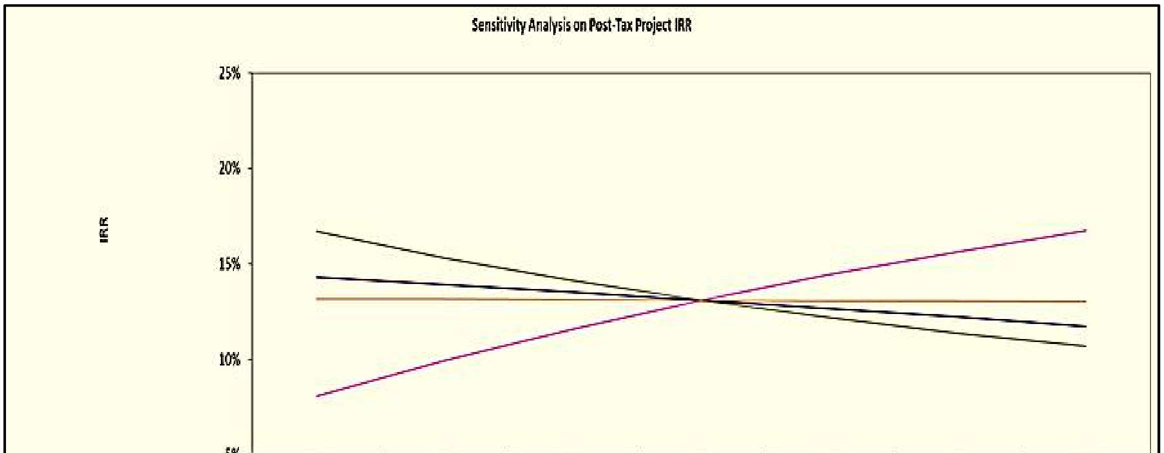
Three (3) additional scenarios have also been analysed. The first one was done assuming only 80 % of the production was sold to China and the rest on the European market. Results showed a slight improvement in the Project economics due to the higher selling price on the European market and lower shipping costs. These are represented in Table 1.10, Figure 1.6 and Figure 1.7 below.

Table 1.10 – Summary of Financial Indicators - Scenario 1
(Production 80 % China – 20 % Europe, 100 % Equity)

	Before Taxes	After Taxes
Project IRR*	15.9 %	13.1 %
NPV @ 6%*	$ 17,783 million	$ 9,839 million
NPV @ 8%*	$ 10,618 million	$ 5,378 million
NPV @ 10%*	$ 6,072 million	$ 2,542 million
Payback Period**	6.9 yrs	7.2 yrs

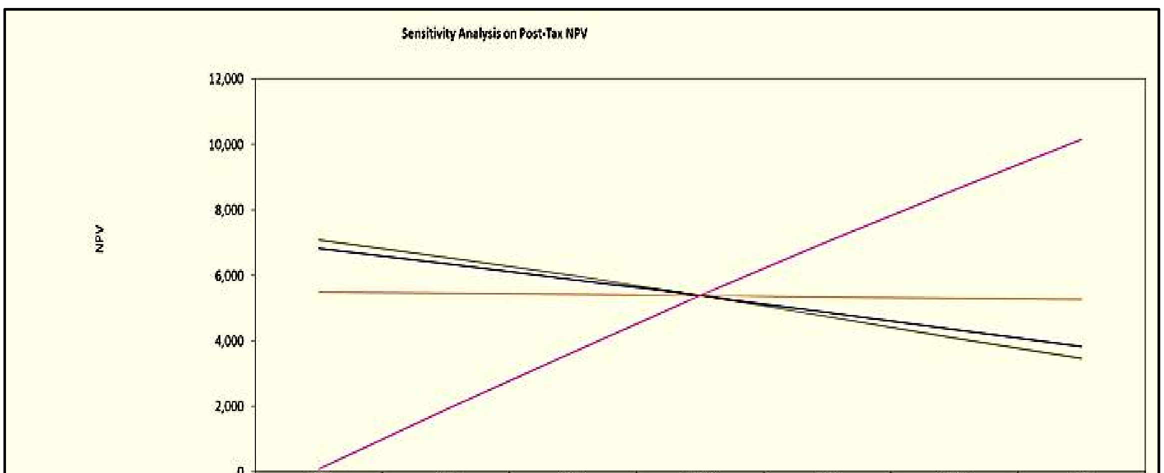
* Based on Free Cash Flow to Equity.
** Calculated from start of commercial production and based on Free Cash flow to Equity.
All monetary values in USD, unless otherwise stated.

Figure 1.6 – Sensitivity of Project IRR (After Tax) – Scenario 1 (Production 80 % China / 20 % Europe, 100 % Equity)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	16.7%	15.3%	14.1%	13.1%	12.2%	11.4%	10.7%
Iron Ore Prices	8.1%	9.9%	11.6%	13.1%	14.4%	15.6%	16.7%
Operating Costs	14.3%	13.9%	13.5%	13.1%	12.7%	12.2%	11.7%
Sustaining Capital Costs	13.2%	13.2%	13.1%	13.1%	13.1%	13.0%	13.0%

Figure 1.7 – Sensitivity of Project NPV (After Tax) – Scenario 1 (Production 80 % China / 20 % Europe, 100 % Equity)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	7,079	6,556	5,992	5,378	4,747	4,095	3,457
Iron ore Prices	79	1,898	3,653	5,378	7,034	8,614	10,159
Operating Costs	6,826	6,349	5,864	5,378	4,869	4,355	3,831
Sustaining Capital Costs	5,486	5,450	5,414	5,378	5,342	5,307	5,271

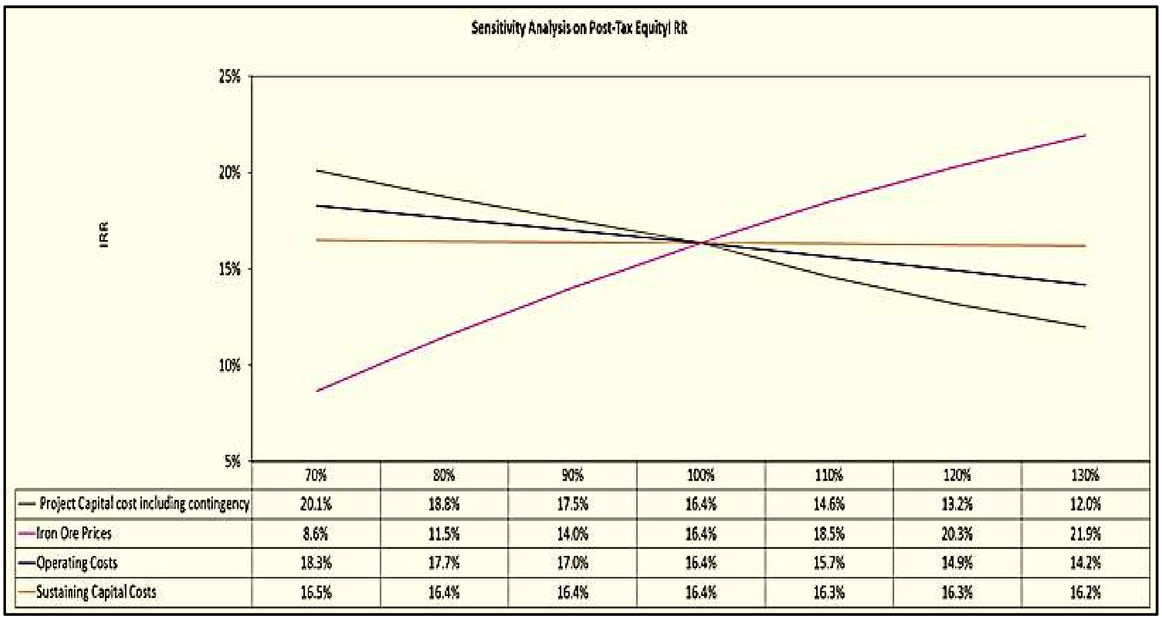
In Scenario 2, the Project funding is changed to 30 % equity and 70 % debt with the production being sold to China as in the base case. As can be seen in Table 1.11, Figure 1.8 and Figure 1.9, the Project economics are improved by 3.3 % after taxes without affecting the NPV.

Table 1.11 – Summary of Financial Indicators -Scenario 2 (Production 100 % China, 30 % Equity / 70 % Debt)

	Before Taxes	After Taxes
Equity IRR*	19.7 %	16.4 %
NPV*	$ 10,174 million	$ 5,519 million
Payback Period**	7.2 yrs	7.5 yrs

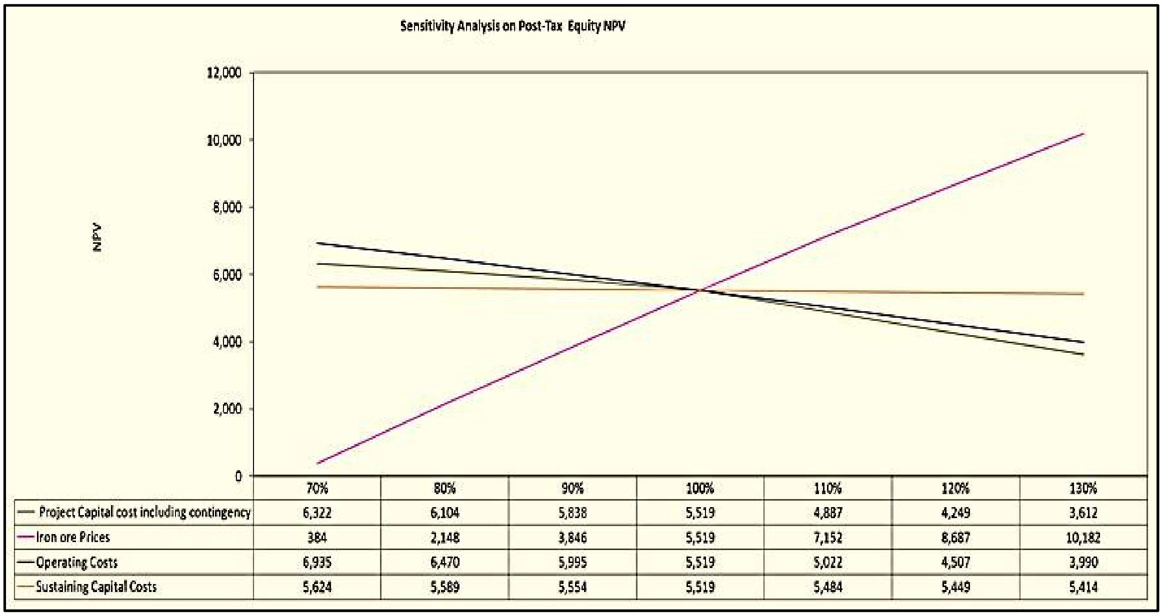
* Based on dividends to shareholders
** Calculated from start of commercial production and based on dividends to shareholders
All prices in USD, unless otherwise stated.
Project NPV discounted at 8 %.

Figure 1.8 – Sensitivity of Project IRR (After Tax) – Scenario 2 (Production 100 % to China, 30 % Equity / 70 % Debt)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	20.1%	18.8%	17.5%	16.4%	14.6%	13.2%	12.0%
Iron Ore Prices	8.6%	11.5%	14.0%	16.4%	18.5%	20.3%	21.9%
Operating Costs	18.3%	17.7%	17.0%	16.4%	15.7%	14.9%	14.2%
Sustaining Capital Costs	16.5%	16.4%	16.4%	16.4%	16.3%	16.3%	16.2%

Figure 1.9 – Sensitivity of Project NPV (After Tax) – Scenario 2
(Production 100 % to China, 30 % Equity / 70 % Debt)



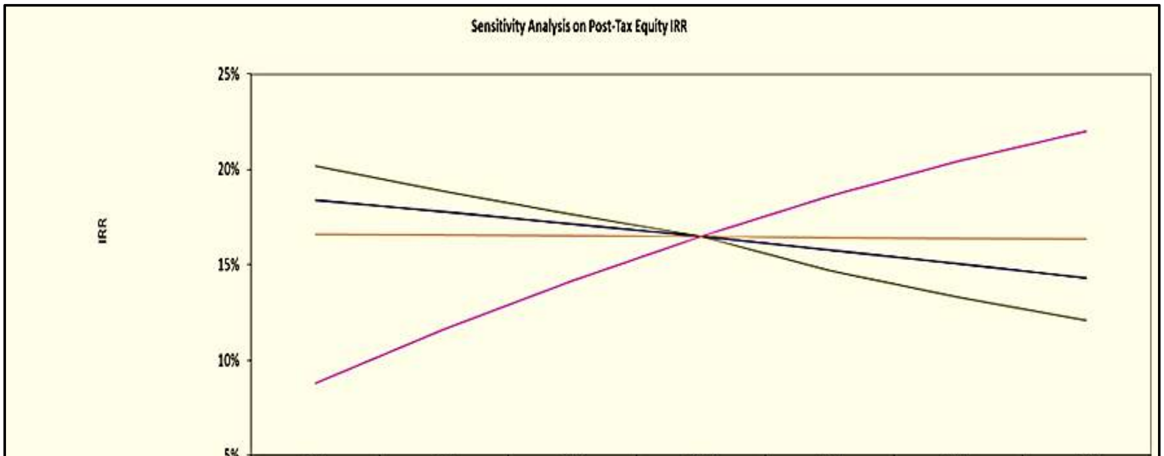
The third scenario is a variation of Scenario 1 showing the effect of the funding alternative proposed in Scenario 2. Again, adding a debt in the equation improves the Project economics by 3.4 % after taxes without really affecting the NPV. The effect due to 20 % of the product being also sold on the European market remains the same. The results are shown in Table 1.12, Figure 1.10 and Figure 1.11 below.

Table 1.12 – Summary of Financial Indicators - Scenario 3
(Production 80 % China – 20 % Europe, 30 % Equity / 70 % Debt)

	Before Taxes	After Taxes
Equity IRR*	19.8 %	16.5 %
NPV*	$ 10,393 million	$ 5,646 million
Payback Period**	7.2 yrs	7.4 yrs

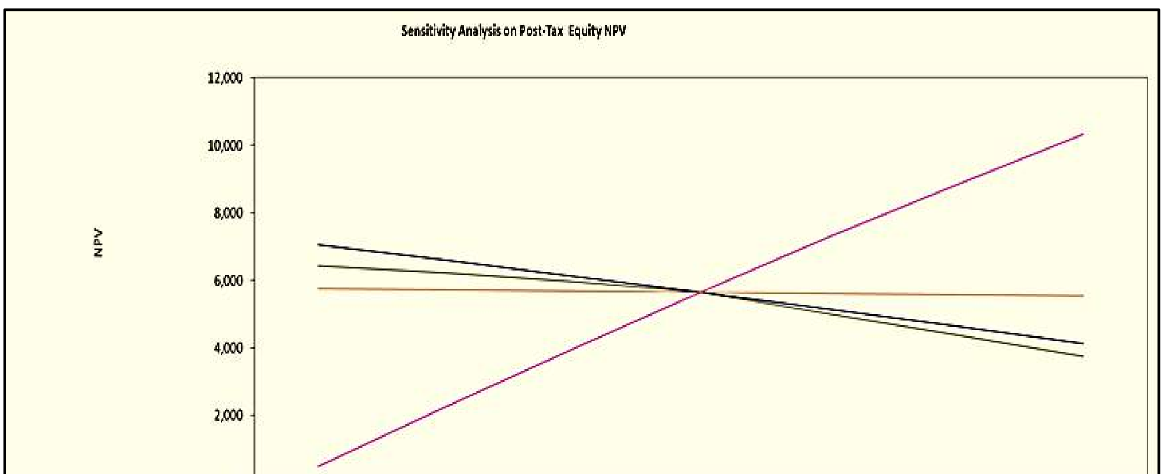
* Based on dividends to shareholders
** Calculated from start of commercial production and based on dividends to shareholders
All monetary values in USD, unless otherwise stated.
Project NPV discounted at 8 %.

Figure 1.10 – Sensitivity of Project IRR (After Tax) – Scenario 3 (Production 80 % China – 20 % Europe, 30 % Equity / 70 % Debt)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	20.2%	18.9%	17.6%	16.5%	14.7%	13.3%	12.1%
Iron Ore Prices	8.8%	11.6%	14.2%	16.5%	18.6%	20.4%	22.0%
Operating Costs	18.4%	17.8%	17.1%	16.5%	15.8%	15.1%	14.3%
Sustaining Capital Costs	16.6%	16.6%	16.5%	16.5%	16.4%	16.4%	16.4%

Figure 1.11 – Sensitivity of Project NPV (After Tax) – Scenario 3 (Production 80 % China – 20 % Europe, 30 % Equity / 70 % Debt)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	6,431	6,216	5,959	5,646	5,014	4,376	3,740
Iron ore Prices	486	2,252	3,961	5,646	7,276	8,824	10,330
Operating Costs	7,056	6,593	6,118	5,646	5,141	4,636	4,120
Sustaining Capital Costs	5,750	5,715	5,681	5,646	5,611	5,575	5,540

The results show that the Project economics are equally good regardless of the scenario used. However, as can be seen in the above graphs, the Project is quite sensitive to variations in the selling price of the iron ore concentrate which is mainly dependant on the demand for this product.

The Feasibility Study has been compiled according to widely accepted industry standards. However, there is no certainty that the conclusions reached in this Feasibility Study will be realized.

1.21 Adjacent Properties

There are four (4) claims within the Lac Otelnuk claim block held by other parties. One of these is within the Mineral Resource area. There are claims that are contiguous and adjacent to the Lac Otelnuk claim block held by third parties.

1.22 Other Relevant Data and Information

None.

1.23 Interpretation and Conclusions

Based on the review of the available information for the Lac Otelnuk Iron Property, we offer the following conclusions:

- The drilling programs have illustrated that the iron formation units have excellent continuity of geology/geometry and TFe grades, with the magnetic Fe grades being more variable due to changes in the magnetite/hematite ratio within the sub-units. The average thickness of the units does not significantly change in the main part of the deposit, but are more variable to the north and south. There appears to be some structural complexity to the northeast of the deposit where possible thrusting has occurred but this was not further explored during the 2013 drilling program as it was not the focus of the campaign;
- The Lac Otelnuk deposits are composed of iron formations of the Lake Superior-type which consists of banded sedimentary rocks composed principally of bands of magnetite and hematite within quartz (chert)-rich rock, with variable amounts of silicate, carbonate and sulphide lithofacies. Lithofacies that are not highly metamorphosed or altered by weathering are referred to as taconite and the Lac Otelnuk deposits are examples of taconite-type iron formation;
- Mineralization in the Lac Otelnuk iron formation consists mainly of magnetite (Fe_3O_4) and hematite (Fe_2O_3); some iron also occurs in silicates, siderite and ferro-ankerite but is economically insignificant. Iron oxide bands containing concentrations of magnetite and/or hematite alternate with grey chert of jasper and are the economically interesting parts of the iron formation that is a gently east dipping interbanded sequence of rocks;
- WGM is satisfied that sampling and assaying for Adriana and LOM's programs since 2007 have been performed well and have been effective leading to the generation of a data set sufficient in quality to support the Mineral Resource estimate;
- Specific gravities for the 2013 Mineral Resource estimation of tonnage were completed using a variable density model based on the relationship generated by WGM between % TFe and measured densities, as WGM determined that a variable density model would more accurately define the local variations based on grade rather than using an average density on a per sub-unit basis;

- As with the previous Mineral Resource estimate, WGM built a relationship between the magnetic Fe determined by Satmagan and that determined by DT where both techniques were used to account for the changeover to Satmagan measurements to replace Davis Tube results during the most recent assaying programs. For consistency with previous Mineral Resource estimates, a % DTWR cut-off was retained based on this relationship. A % Magnetic Fe value was determined for each block and this is reported in the current Mineral Resource estimate along with the % DTWR;
- The 2013 Mineral Resource estimate included the new drilling results from the 2012 exploration program and uses of total of 370 drillholes. WGM re-modeled the upper geological sub-units of the Lac Otelnuk iron formation that were previously defined (2a, 2b, 2c, 3a and 3b) and retaining the transitional 2b-c sub-unit identified in the 2012 estimate. A new internal shale waste unit was also defined in the northern part of the Property. Internally, the continuity of the sub-units was excellent, so WGM had no issues with extending the interpretation beyond 600 m distance. This extension was taken into consideration when classifying the Mineral Resources and these areas were given a lower confidence category. A summary of the NI 43-101 compliant Mineral Resources is provided in Table 1.13;

Table 1.13 – 2013 Categorized Mineral Resource Estimate for Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

Resource Classification	Tonnes (in billions)	TFe Head (%)	DTWR (%)	Magnetic Fe (%)
Measured	16.21	29.3	25.8	17.8
Indicated	4.43	31.5	24.1	16.7
Total M&I	**20.64**	**29.8**	**25.4**	**17.6**
Inferred	6.84	29.8	26.3	17.8

- The metallurgical testing to date demonstrates that the Lac Otelnuk mineralization can be recovered by fine grinding and magnetic concentration to saleable concentrates with low silica and high iron grades. Test work performed on the 30 Y composite samples has demonstrated concentrate grades with <4 % $Si0_2$ and >68.5 % iron with a 27.6 % weight recovery;
- The open pit designed for the Lac Otelnuk Project will provide for a 30-year mine life. The Proven and Probable Mineral Reserves within this open pit include 4,993 Mt of ore at an average Davis Tube Weight Recovery of 26.5 %. The open pit is 11.6 km long and 2.8 km wide, reaches a maximum depth of 130 m and has waste to ore stripping ratio of 0.28 to 1;
- The 30-year mine plan that was developed follows the phased approach of the Feasibility Study, producing 30 Mt/y of concentrate in Phase 1 and 50 Mt/y in Phase 2. During peak production, the total number of 363-tonne haul trucks is expected to reach 50, along with ten (10) cable shovels, two (2) hydraulic shovels, four (4) front end wheel loaders, 16 production drills and a large fleet of support and service equipment;

- The processing plant will use proven and reliable equipment to produce a 68.5 % Fe concentrate that will be transported via two (2) pipelines (755 km) to the Pointe-Noire area in Sept-Îles, QC, where is will be dewatered, store and then ship to its final destination;
- It can be stated that the level of confidence in the test work results is high, since (a) a reputable laboratory was used, (b) significant effort was placed on sample representativeness, and (c) the results were largely coherent and repeatable and were given explanations for instances where these were not true;
- Test work conducted on representative samples of tailings and waste rock have concluded that they should be considered to be non-acid generating with low metal leaching potential and therefore meet the "low risk" classification of the Quebec Directive 019 regulations;
- The system for the tailings and water management has been developed taking into considerations of local topographical, geotechnical, geomorphological and climatic conditions, stringent engineering design and reliable construction concepts, environmental management and closure requirements. The configuration of the Tailings Management Facility ("TMF") provides a tailings storage volume of about 2,350 Mm^3;
- The product delivery system is a concentrate slurry transport system, an economical and reliable means of transporting iron ore concentrate to the project port of export. Further optimization, particularly with respect to the route alignment, will be assessed early in the basic engineering phase of the Project, along with further field investigations;
- A product dewatering-storage-reclaiming facility is designed located near the Port of Sept-Îles. The final product which contains less than 8 % of moisture content will be loaded on the large Cape Size to Chinamax Size bulk carriers through a conveyor/ship loading system at the deep water wharf jointly build by LOM and the Sept-Îles Port Authorities;
- The power for the Project will be supplied by a 735 kV power transmission line connected to the existing Hydro-Québec 735/315 kV substation at Tilly. The 735 kV overhead transmission line is approximately 466 km long and includes one (1) single circuit, one (1) overhead shield wire, and one (1) optical ground wire;
- The project market study by the SNL Metals & Mining indicates future demand for pellet feed. Both trends and forecasts indicate a rebalancing of the pellet feed premium by the time LOM enters into production;
- The economics of the Project are based on the concentrate being shipped primarily to China;
- The Project is technically feasible: the ore can be mined, treated, and delivered to the Port of Sept-Îles for export by employing proven processes and technologies;

- Based on a 30-year mine life and a production of 50 Mt/y of iron ore concentrate as well as the parameters and assumptions set out in this Report, the IRR before taxes varies between 15.9% and 16.1% depending on the scenario.

1.24 Recommendations

Moving forward, we would recommend:

- Carry-out a geotechnical study for the stability of the waste rock piles and overburden stockpiles to confirm the design parameters used in the Feasibility Study.
- Complete geochemical testing on the shale waste unit that appears at the north end of the 30-year open pit. If the test work identifies that the shale waste unit is not a net producer of acid rock drainage, the pit should be redesigned to include this area in the next phase of the Project.
- That pelletizing tests be done, since the product of the Lac Otelnuk Project is a pellet feed.
- Perform additional metallurgical evaluations to:
  - Finalize SAG vs. AG milling options;
  - Evaluate the potential to use of tower mills instead of ball mills;
  - Perform a variability-testing bench scale program with geographically dispersed samples from the first ten (10) years of the mining plan to further evaluate the metallurgical performance of the proposed flow sheets;
  - Map the ore hardness in the mine plan;
  - Develop a metallurgical model and calculate different mass balances for three (3) qualities of ore (rich-average-poor) in order to validate the capacity of the plant and the sizing of the equipment.
- Complete geotechnical investigations for the main access road, the process plant area, the TMF and water management structures, the alignment along the 735 kV power interconnection line and the PDS and the Dewatering-Storage-Reclaiming facilities.
- Perform topographical survey (LiDAR) for the alignment along the power transmission line and the PDS and for the Dewatering-Storage-Reclaiming facility at the port area.
- Perform bathymetry survey on the water bodies along the PDS.
- Finalize the EIA reports for submission to the Quebec and Canadian governments
- Finalize the Social Impact Assessment ("SIA") reports for submission to the Quebec and Canadian governments
- Submit the Project to public consultation on the EIA/SIA reports, permitting, and licences to obtain local community buy-in.

2.0 INTRODUCTION

2.1 Terms of Reference - Scope of Study

In September 2005, Adriana Resources Inc. ("Adriana") acquired the right to earn a 100 % interest in claims, notwithstanding certain royalties held by Bedford Resource Partners Inc. ("Bedford") encompassing its Lac Otelnuk Iron Property in the Labrador Trough, Nunavik, Quebec. Additional contiguous claims were staked by Adriana in 2005 through 2013. These claims comprise Adriana's Lac Otelnuk Iron Property (the "Property"). In January 2012 Adriana with a wholly owned subsidiary of WISCO International Resources Development & Investment Limited ("WISCO") formed a joint venture company: Lac Otelnuk Mining Ltd. ("LOM"). Pursuant to this agreement, the Property was transferred into LOM which is held 60 % by WISCO and 40 % by Adriana.

The Property includes an undeveloped surface exposed, gently dipping taconite iron deposit, known as the Lac Otelnuk iron deposit, first recognized and mapped in 1948. In the 1970s, the first diamond drilling was completed, and metallurgical and economic studies were carried out.

In late-2005, Watts, Griffis and McOuat Limited ("WGM") was retained by Adriana to complete and document a technical review of the Property and make recommendations for an exploration program. In 2007, Adriana initiated its first exploration program on the Property focused on the South Zone. Diamond drilling resumed in summer 2008 and continued through to the fall. The purpose of the 2007 and 2008 drilling was to complete the drilling of a rectangular area of the South Zone, approximately 9.0 km long by 2.5 km wide with holes on 500 m by 600 m centres. The program was successful in meeting this goal.

In December 2008, WGM was retained by Adriana to complete an independent Mineral Resource estimate for the Property. As documented in the NI 43-101 report dated May 7th, 2009, this estimate defined Indicated Mineral Resources totaling 4.29 billion tonnes averaging 29.08 % TFe and Inferred Mineral Resources totaling an additional 1.97 billion tonnes averaging 29.24 % TFe.

In November 2010, Met-Chem Canada Inc. ("Met-Chem") was retained by Adriana to produce a NI 43-101 Preliminary Economic Assessment ("PEA") of the Lac Otelnuk Iron Property using WGM's 2009 Mineral Resource estimate. This scoping level study evaluated options, based on the data available at that time, to establish the viability of the Project at a production rate of 50 million tonnes of pellets per year in order to justify proceeding with other phases of project development.

The study was based on the assumption that an open pit mine and concentrator operation will be constructed at Lac Otelnuk together with the required tailings disposal works and site infrastructure. Pellet production was also included in the concept with an assessment of site location at either the mine or port site included within the Study. The Project also includes construction of a railway to allow transport of either concentrate or pellets, to a new port facility to be constructed. In either case, the end product would have been pellets to be loaded on oceangoing, iron ore vessels using ship loading facilities at the Sept-Îles Port.

This study was positive. The economic analysis of the asset indicates a solid economic performance under the conditions analyzed. Met-Chem concluded that the average grade (+19 % MagFe) and weight recovery (27 %) used in the study needed to be supported by confirmation test work and that further test work on the pelletizing of the Otelnuk concentrate was also required.

In 2010 and 2011, further drilling on the Property was conducted. In August 2011, WGM was retained to update the Mineral Resources using information from 2010 - 2011 infill drilling program on the South Zone. WGM estimated 4.89 billion tonnes of Measured and Indicated Mineral Resources and an additional 1.56 billion tonnes of Inferred Mineral Resources based on a Davis Tube Weight Recovery ("DTWR") cut-off grade of 18 %. No technical report was required in support of this Mineral Resource estimate as it was not deemed to be a material change.

Golder Associates Ltd. ("Golder") was then retained to prepare the environmental and social considerations for the Lac Otelnuk Project. Since 2008, Golder has prepared an early-stage environmental scoping study to support the Project, conducted bio-physical and social baseline studies of the mining area and led several activities to prepare the basis for the Environmental and Social Impact Assessment ("ESIA") of the Project.

In 2012, WGM was retained by LOM to provide an updated Mineral Resource Estimate and technical report based on all drilling through the 2011 program. WGM estimated 11.35 billion tonnes of Measured and Indicated Resources based on a DTWR cut-off grade of 18 % and an additional Inferred Resource of 12.39 billion tonnes. Additional drilling was completed in 2012 consisting of 196 holes aggregating 22,249 m.

In 2013, WGM was again retained by LOM to provide an updated Mineral Resource Estimate based on all drilling and exploration results to date and document its findings in a technical report compliant with NI 43-101 guidelines and standards and Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definitions. The technical report concluded that 20.64 billion tonnes averaging 29.8 % Total Fe head grade at 25.4 % DTWR using a cut-off at 18 % DTWR of resources are contained in the Lac Otelnuk ore body of which 16.21 billion tonnes are classified as measured and 4.43 billion tonnes as indicated. Additionally, 6.84 billion tonnes bearing 29.8 % TFe head grade at 26.3 % DTWR have also been identified and classified as inferred.

In parallel, LOM has mandated SNC Lavalin to produce a feasibility study on the Lac Otelnuk Property based on the assumption that a 50 million tonnes of product per year open pit mine and concentrator operation will be constructed at Lac Otelnuk together with the required tailings disposal works and site infrastructure. Trade-off studies to decide on the final product (pellets or concentrate), product delivery system and power supply type and logistics were part of this Study. A new port facility, capable of servicing + 400,000 DWT vessels, to be constructed and located at the Sept-Îles Port was also assumed.

Met-Chem has been mandated to assemble the present Technical Report that presents the Mineral Reserves and the estimated associated capital and operating costs to develop the Lac Otelnuk Project of the Lac Otelnuk Iron Property owned by Lac Otelnuk Mining Ltd. (LOM). This estimate is considered to be a feasibility study level.

2.2 Source of Information

The majority of the information herein has been derived from the Feasibility Study performed by SNC Lavalin Inc. from October 2013 to March 2015 on behalf of LOM and on the estimate of mineral resources compliant with Canadian Securities Administrators National Instrument 43-101 (NI 43-101) as provided by WGM in their report dated October 31st, 2013.

2.2.1 Contributing Authors

Met-Chem has relied on the responsible Qualified Person as presented in Table 2.1 for the writing of the individual sections of this Report.

2.2.2 Qualified Persons

The following Table 2.1 provides the list of qualified persons responsible for the content of the individual sections of this Technical Report.

Table 2.1 – Qualified Persons and their Respective Sections of Responsibilities

Section	Title of Section	Qualified Persons
1.0	Summary	A. Boilard, MCC
2.0	Introduction	A. Boilard, MCC
3.0	Reliance on Other Experts	A. Boilard, MCC
4.0	Property Description and Location	A. Boilard, MCC
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography	A. Boilard, MCC
6.0	History	A. Boilard, MCC
7.0	Geological Setting and Mineralization	R. W. Risto, WGM
8.0	Deposit Types	R. W. Risto, WGM
9.0	Exploration	R. W. Risto, WGM
10.0	Drilling	R. W. Risto, WGM
11.0	Sample Preparation, Analysis and Security	R. W. Risto, WGM
12.0	Data Verification	M. W. Kociumbas and R. W. Risto, WGM
13.0	Mineral Processing and Metallurgical Test Work	R. Martinez, SLI
14.0	Mineral Resources Estimates	M. W. Kociumbas, WGM
15.0	Mineral Reserve Estimates	J. Cassoff, Met-Chem
16.0	Mining Methods	J. Cassoff, Met-Chem
17.0	Recovery Methods	R. Martinez and J. Lord, SLI
18.0	Project Infrastructure	R. Martinez and J. Lord, SLI
19.0	Market Studies and Contracts	Reliance on SNL
20.0	Environmental Studies, Permitting and Social or Community Impact	E. Giroux, WSP
21.0	Capital and Operating Costs	S. Buccitelli and M. Côté, SLI
22.0	Economic Analysis	S. Buccitelli, SLI
23.0	Adjacent Properties	A. Boilard, MCC

Section	Title of Section	Qualified Persons
24.0	Other Relevant Data and Information	Not Applicable
25.0	Interpretation and Conclusions	A. Boilard and related QPs
26.0	Recommendations	A. Boilard and related QPs
27.0	References	A. Boilard and related QPs

Additionally, Met-Chem is responsible for the development of the Sections 1 to 6, 15, 16 and 23 through 27. Met-Chem is also responsible for the assembly of this NI 43-101 Technical Report for the Feasibility Study made on the Lac Otelnuk Project.

2.3 Effective Date and Declaration

This Report is considered effective as of March 25th, 2015 and is in support of the Adriana Resources Inc. / Lac Otelnuk Mining Ltd. press release, dated April 22nd, 2015, entitled "Adriana announces completion of the Feasibility Study on the Lac Otelnuk Project".

The current Report provides an independent Technical Report for the Feasibility Study of the Iron mineralization of the Lac Otelnuk Deposit, in conformance with the standards required by NI 43-101 and Form 43-101F1. The estimate of Mineral Reserves contained in this Report conforms to the CIM Mineral Resource and Mineral Reserve definitions.

Met-Chem is not insider, associate or an affiliate of Lac Otelnuk Mining and Adriana Resources Inc. ("ADI") and neither Met-Chem nor any affiliate has acted as advisor to LOM and ADI, its subsidiaries or its affiliates, in connection with this Project.

It should be understood that the Mineral Reserves presented in this Report are estimates of the size and grade of the deposits based on a number of drillings and samplings and on assumptions and parameters currently available. The level of confidence in the estimates depends upon a number of uncertainties. These uncertainties include, but are not limited to, future changes in product prices and/or production costs, differences in size and grade and recovery rates from those expected, and changes in Project parameters. In addition, there is no assurance that the Project implementation will be realized.

The comments in this Report reflect Met-Chem's best judgment in light of the information available at the time of preparation. Met-Chem reserves the right, but will not be obligated, to revise this Report and its conclusions if additional information becomes known to Met-Chem subsequent to the effective date of this Report.

2.4 Site Visit

Mr. Richard W. Risto, Senior Associate Geologist with Watts, Griffis and McOuat Limited, visited the Property on August 28th and August 29th, 2007 and from August 13th to August 16th, 2008.

Mr. Jeffrey Cassoff, Lead Mining Engineer from Met-Chem Canada Inc. visited the site on June 19th, 2013.

Sam Buccitelli, Project Manager with SNC-Lavalin Inc., visited the site on June 19th, 2013.

Julien Lord, Engineering Manager with SNC-Lavalin, visited the site on June 19th and September 10th, 2013.

2.5 Units and Currency

All prices and costs in this Report are expressed in United States of America dollars (USD or $) unless otherwise specifically stated.

3.0 RELIANCE ON OTHER EXPERTS

Met-Chem performed a cursory review of the information provided by other consultants for completion of this NI 43-101 Report; however, each consultant remains fully responsible for their own work, and Certificates of Authors that comply with NI 43-101 regulation are included in this Report.

In the preparation of Chapters 4 to 6 of this study, Met-Chem largely drew from, and summarized the WGM reports on the resources estimates, as noted in the text and listed under Section 27 (References) of this Report. In several instances, Met-Chem added to, or updated, information found in WGM's reports. In addition, Met-Chem has relied on information provided by Adriana, such as public Management Discussion & Analysis ("MD&A") or other documents filed with SEDAR, for the description of the Property agreements and royalties.

A reasonable amount of verifications was completed by Met-Chem on the main items addressed in the last resources estimates by WGM (2013) and the results were presented to LOM in a Technical Notes (internal document).

The list and status of the claims forming the Property were checked on GESTIM, the public Register of mining rights in Quebec.

However, Met-Chem has not independently verified the legal title to the Property and the status of Adriana's or LOM's Property agreements.

The portions of this Study relating to geology and mineral resource were prepared using essentially the NI 43-101 Technical Report prepared for Lac Otelnuk Mining Ltd. by WGM dated October 31st, 2013.

Met-Chem has not carried out any independent geological surveys of the Property and did not review drill core and results. Met-Chem has relied for the geological descriptions and program results solely on the basis of reports, notes and communications completed by or for LOM.

Met-Chem has also relied on the Market Study performed by SNL Metals and Mining titled "Market Study Lac Otelnuk Project" and dated October 16th, 2014 and revised on March 24th, 2015 for the pricing of the iron ore concentrate used for the financial analysis of the Project.

Met-Chem has relied on Deloitte's Mrs. Geneviève Provost, LL M. Fisc., CPA, CA and Mr. Marc-Antoine Brault Brissette, LL. L, LL. M for the review of the financial analysis model produced by SNC Lavalin Inc. in relation to the federal, provincial and mining taxes calculations.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Description and Location

The Property is located in Nunavik, Province of Quebec, in the central portion of the Labrador Trough iron range as shown in Figure 4.1. The centre of the Property is situated approximately 155 km in a straight line northwest of Schefferville and 225 km south of Kuujjuaq. Schefferville is located approximately 1,200 km northeast of Montréal.

Figure 4.1 – Location Map

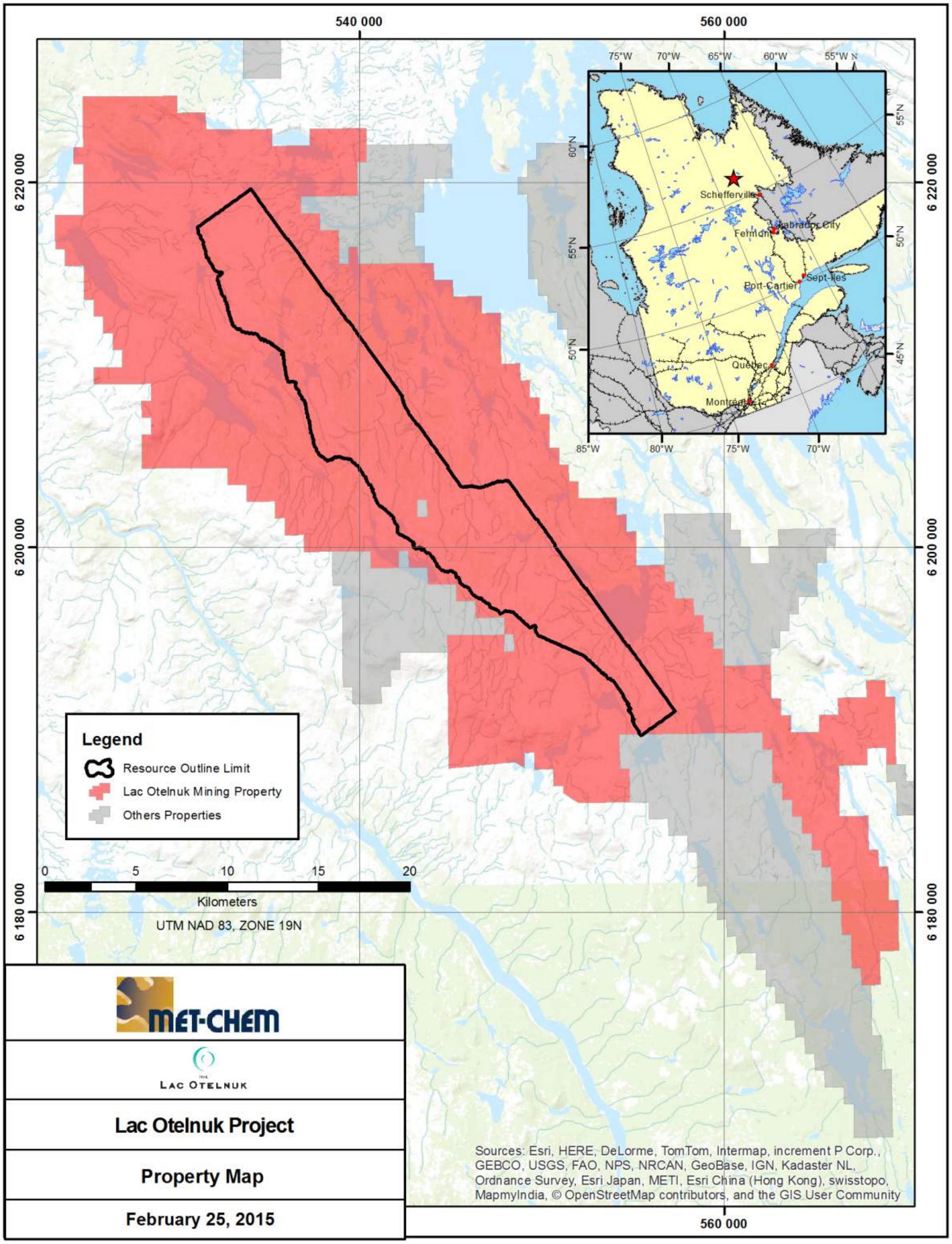
Source SLI

The Property has an elongate shape stretching along the NW direction over about 64 km between latitudes 56°10.0'N and 55°43.5'N and longitudes 68°37.5'W and 67°53.5'W. The Property straddles NTS map sheets 24C01, 24C02, 23N15, 23N16, 23O12 and 23O13.

4.2 Property Ownership and Agreements

On January 29th, 2015 Met-Chem verified the status of the claims on the public Register of mining rights in Quebec (GESTIM) via the Quebec Ministry of Natural Resources website. The records indicate that the Property consists of 1,398 contiguous map-staked mineral claims covering approximately 673 km^2 and registered as 100 % under Lac Otelnuk Mining Ltd. ("LOM") as per Figure 4.2.

Figure 4.2 – Property Map



Adriana owns 40 % of LOM, a company incorporated in the Province of Quebec in November 2011. LOM is a joint venture company with WISCO International Resources Development & Investment Limited ("WISCO") who earned the remaining 60 % interest. LOM owns two (2) properties within the Labrador Trough in the Nunavik region: the Lac Otelnuk property and the December Lake property. The December Lake property is comprised of 160 contiguous mineral claims covering approximately 74 km^2 that are registered to LOM as the 100 % owner. The December Lake property is located to the NW of the Otelnuk Property, within NTS map sheet 24C10, and the two (2) are separated by a distance of about 47 km. For more details please consult map on Figure 23.1. Although these 160 claims are owned by LOM, they are outside of the iron resources that are the subject of this Report.

All the claims on the Property were active and in good standing at the time of writing this Report. The claims were registered between July 2005 and October 2014 and have expiration dates ranging from April 3rd, 2015 (currently identified by GESTIM as being processed) and September 30th, 2016. Table 4.1 provides a summary listing of the claims on the Property with a general description. A complete listing of the active claims is available on GESTIM.

The claims have a validity of two (2) years and can be renewed indefinitely for two-year periods, under certain conditions, provided the required exploration work is completed and fees are paid. Excess credit related to the required assessment work for the entire Property amounts to over CAD$ 46.2 M while about CAD$ 1.6 M of work are required for the next renewal of all the claims, and CAD$ 159,264 are due as renewal fees (Table 4.1). Excess work on one claim may be spread to other claims held by the same owner within a radius of 4.5 km.

The Property has not been surveyed but all the claims are registered as map-designated claims that have the same boundaries as the land lots, or parts of.

The claims give the owner exclusive rights to explore for mineral substances, with a few exceptions like hydrocarbons, sand and gravel. The claims do not convey the surface rights but access to the claims is granted to carry out exploration work. However, the claim holder cannot enter land granted for non-mining purposes or land leased for mining surface mineral substances without permission from the current holder of these rights.

Table 4.1 – Summary of the Claims of the Otelnuk Property

Registration Date	Location (Map Sheet)	Number of Claims	Cumulative Area (ha)	Excess Work Credit (CAD$)	Required Work (CAD$)	Required Fees (CAD$)
2005	24C01	280	13,472	18,663,643	504,000	31,920
	24C02	53	2,547	143,933	95,400	6,042
	23N16	266	12,846	25,951,865	478,800	30,324
2006	24C01	19	916	188,549	34,200	2,166
	23N16	20	965	150,223	36,000	2,280
2008	24C01	42	2,023	0	56,700	4,788
	23N16	81	3,910	372,558	109,350	9,234
2009	24C01	32	1,540	477,495	28,800	3,648
	24C02	30	1,442	430	27,000	3,420
	23N16	30	1,451	0	27,000	3,420
2010	24C01	41	1,974	244,247	36,900	4,674
	24C02	4	192	430	3,600	456
2011	24C01	27	1,301	0	12,150	3,078
	24C02	121	5,821	66,949	54,450	13,794
	23O12	10	485	0	4,500	1,140
	23O13	104	5,037	0	46,800	11,856
	23N15	2	96	0	900	228
	23N16	105	5,069	0	47,250	11,970
2013	24C02	8	385	0	1,080	912
	23N15	10	482	0	1,350	1,140
	23N16	111	5,239	0	14,850	12,546
2014	23N16	1	48	0	135	114
	24C01	1	48	0	135	114
TOTAL		**1,398**	**67,291**	**46,260,323**	**1,621,350**	**159,264**

4.3 Royalties and Encumbrances

Ten (10) claims on the Property carry an encumbrance related to a Protected Area for a planned Park over Eaton Canyon on the Caniapiscau River. In addition, one (1) claim registered under Gilles A. Tremblay and another one (1) held by Groupe-Conseil Delro Inc. lie within the southern sector of the Property. These are shown in Figure 23.1.

LOM has a commitment to pay 1.25 % of its gross revenue on 328 of its land claims to the former owner of these claims, and must pay a minimum royalty advance of CAD$ 450,000 in November of each year until commencement of commercial production.

4.4 Permits

Work permits are required to carry out diamond drilling activities and operate the base camp at Baie Gignard. Various permits will be required for development and mining activities.

4.5 Environmental Considerations

The Otelnuk Iron Project will be subjected to Provincial (Quebec) and Federal Environmental Assessments. Under the Environmental Quality Act (Quebec Ministry of Sustainable Development), the Project will have to address different mechanisms of authorization for the Project. The mining and concentrating activities of the Project fall within the territory governed by the James Bay and Northern Quebec Agreement ("JBNQA").

The Environmental and Social Impact Assessment studies are still in the preliminary stages and significant activity will begin following the Feasibility Study.

Additional information on environmental issues is provided under Section 20 of this Report (Environmental Studies, Permitting and Social or Community Impact).

4.6 First Nation Issues

Adriana signed a Letter of Intent ("LoI") with Makivik Corporation ("Makivik"), the development corporation mandated to manage the heritage funds of the Inuit of Nunavik. The LoI provides for Adriana to foster communications with Nayumivik Landholding Corporation of Kuujjuaq, the Northern Village of Kuujjuaq and the Kativik Regional Government ("KRG"). The Naskapi Nation of Kawawachikamach holds a seat on the Kativik Board. Nayumivik Land Holding Corporation is an affiliate of Makivik that holds title to the Inuit Lands.

Details on the LoI are provided in WGM's May 7th, 2009 technical report and under Section 20 of this Report.

4.7 Risks

Met-Chem is not aware of any other significant factors or risks that may affect access, title or the right or ability to perform work on the Property.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Property has no road access. Several lakes on the north and south parts of the Property are accessible from Schefferville and Kuujjuaq via fixed-wing float or ski-equipped aircraft. Access to Schefferville and Kuujjuaq is provided by daily scheduled air service from Quebec City, Montréal and Sept- Îles. Weekly passenger and freight train service between Schefferville and Sept-Îles is available. The village of Caniapiscau, situated about 160 km southwest of the Property, provides an alternative access route. The village is connected by the Trans-Taiga road from Val-d'Or via Matagami and Radisson.

5.2 Physiography

Topography on the Property is flat to gently rolling, with elevations varying from 260 to 380 m above sea level. A NW-SE trending, 5-10 m high cliff face representing the surface exposure of the iron formation occurs on the northern half of the Property. The Property is poorly drained and has extensive swampy areas.

5.3 Climate and Vegetation

The Project area is under the influence of humid, sub-arctic continental taiga climate conditions experiencing very severe winters and cool summers. Daily mean temperatures for Schefferville average 24.1 °C and 22.6 °C in January and February, respectively, and +12.4 °C and +11.2 °C in July and August respectively. Details on the weather in Schefferville are presented in Table 5.1. The Project area is located in the sporadic discontinuous zone of permafrost.

Table 5.1 – Schefferville - Historical Weather Data

Month	Daily Temperature (°C)		Total Rainfall (mm)	Total Snowfall (cm)	Average Snow Days
	Maximum	Minimum			
January	5.1	-48.3	0.26	53.72	17
February	5.1	-50.6	0.29	33.26	14
March	9.4	-45	1.4	54.65	17
April	13.1	-36.1	9.04	50.49	14
May	28.3	-23.3	26.12	22.38	11
June	34.3	-7.8	69.53	5.76	4
July	31.7	0	96.06	0.15	0
August	28.7	-3.3	81.94	0.38	0
September	26.7	-9.4	102.99	11.05	6
October	20.6	-19.4	24.46	50.78	18
November	9.8	-35.6	4.51	62.75	21
December	5	-47.2	0.73	53.04	19
TOTAL			**417.33**	**398.41**	**141**

Source: Environment Canada, Weather Station at Schefferville airport (Data: 1948-1993)

Although the Project lies in the northern part of the Province of Quebec, Canadian miners are experienced operating mines under harsh climatic conditions like those prevailing in the Otelnuk area.

The Project is located in the broad transitional zone between boreal forest and treeless tundra. Lichen woodlands form a mosaic with treeless ridges, forested valleys sparsely occupied by black spruce trees, and meadows.

5.4 Local Resources

Kuujjuaq, the closest community to the Property, has a population of approximately 2,200. Kuujjuaq is the administration center of Nunavik and head offices for Makavik Corporation, Kativik Regional Government, Katavik development Council, and Nunavik Board of Health and Social Services. Kuujjuaq has a modern hospital, schools, accommodations, stores and banking facilities. Daily service to the airport in Kuujjuaq and charter flight services are available.

In 2011, Schefferville had a population of 213 inhabitants (Canada census). However, the town has since experienced an influx of workers, principally triggered by the re-start of iron production in the region.

Schefferville provides services such as accommodation, basic supplies and equipment, contractors and charter flight operators. The town is connected to Sept-Îles by weekly passenger and freight train and via a modern airport.

In 2011, some 540 members of the Nation Innu Matimekosh-Lac John lived in the nearby Matimekosh community. Kawawachikamach, a community located some 20 km north of the town of Schefferville, is the home of the Naskapi First Nation of Canada. Statistics Canada indicated a population of 586 Naskapi people in 2011 living in a modern community that has its own school, medical clinic and recreational complex.

Although part of the labor force required for a mining operation at Lac Otelnuk could be found locally, a significant portion would come from other regions of Eastern Canada and training programs will certainly be required.

5.5 Infrastructures

A 25-m high water falls, with hydro-electricity generating potential, is located 15 km north of the centre of the Property, on a river west of Lac Otelnuk. The nearest Hydro-Québec power lines are in Schefferville, where local needs are served by the Menihek Lake power plant located in Labrador. There seems to be more than adequate supply of water available for exploration and mining purposes.

Ample space seems to be available on the Property for the establishment of the infrastructure for a mining and processing operation. The surface outline of the currently defined mineral resource essentially occupies the central portion of the northern half of the Property.

No harvestable timber is present on the Property.

There has been no mining activity on the Property or in the surrounding area and as such there are no mine workings, tailings impoundment areas, waste piles or other infrastructure on or near the Property.

6.0 HISTORY

6.1 Ownership History

The original 129 claims of the Property were acquired by Adriana in 2005 from Bedford Resource Partners Inc. ("Bedford"), a private company. Adriana staked additional claims, contiguous with those of Bedford later in 2005, and through to 2013.

On January 12th, 2012, Adriana closed a Joint Venture Agreement (JVA) with a wholly-owned subsidiary of WISCO International Resources Development & Investment Limited ("WISCO") to engage in the development and operation of the Lac Otelnuk Project. LOM, the joint venture company with WISCO, was incorporated in the Province of Quebec in November 2011. Pursuant to the JVA, WISCO funded an aggregate of CAD$ 91,634,000 of which CAD$ 51,634,000 was paid directly to Adriana and the remaining CAD$ 40,000,000 was paid to LOM. WISCO acquired a 60 % interest in LOM while Adriana holds the remaining 40 %.

Additional information can be found in WGM's 2013 technical report as well as in Adriana's documents filed with SEDAR or available on their website.

Met-Chem has not verified the details on the agreements and transactions affecting the Lac Otelnuk Property.

6.2 Historical Exploration and Development

The presence of significant iron formation on the Lac Otelnuk Property area was reported initially in 1948 by Norancon Exploration (Quebec) Limited, a Noranda/Conwest joint venture. A summary of the main exploration and development activities on the Property is presented in Table 6.1. However, additional details can be found in WGM's 2013 technical report.

Adriana's 2007 drilling program was the first ground exploration reported after King's 1976 field program. LOM and Adriana's activities are described more fully under the Exploration and Drilling sections of this Report.

Table 6.1 – Summary of the Main Exploration and Development Activities on the Property

Company	Year	Activities
Norancon Exploration	1948	• Regional exploration.
King Resources Company	1970-1971	• Claim staking; • Field mapping; • Ground magnetic survey; • Drilling 702.9 m in 10 holes; • Metallurgical test work; • Preliminary "resource" estimate and economic studies (N Zone).
	1973	• Drilling 21 holes for a total of 645.7 m; • Preliminary "resource" estimate (N Zone).
	1974	• Metallurgical test work (samples from the N Zone); • Preliminary economic study.
	1975-1976	• Field mapping; • Drilling 307.8 m in 5 holes; • Preliminary "resource" estimate (S Zone); • Bulk sampling (18 long tons) and metallurgical testing (samples from the N Zone).
Phoenix Resources Company	1981	• Pilot plant, pelletizing tests at Lakefield and in Germany.

6.3 Historical Resource Estimates (1970 – 1977)

Several "mineral resource" estimates were completed between 1970 and 1977. However, Met-Chem will not comment on these estimates as they are outdated, irrelevant for the purposes of this Report and superseded by a current estimate completed by WGM in 2013.

6.4 Historical Drilling

King completed the first drilling on the Property in 1973. Systematic drilling on the Property by Adriana started in 2007 and the last historical program was completed in 2011.

Subsequently, Adriana drilled 196 holes, 157 of which were delineation holes, in 2012 (Section 10, Drilling).

Table 6.2 provides a summary of the historical drilling on the Property. Additional details can be found in WGM's 2013 technical report.

Table 6.2 – Summary of the Historical Drilling Programs on the Property

Operator	Year	Number of Holes	Meterage (m)
King	1970	10	702.9
	1973	21	645.7
	1976	5	307.8
TOTAL	**1970-1976**	**36**	**1,656.4**

6.5 Production

There has been no mining activity on the Property.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

WGM has relied for our geological descriptions and program results solely on the basis of historic reports, notes and communications with LOM and Adriana's personnel. Additional results and descriptions have been summarized in previous WGM NI 43-101 technical reports.

7.1 Regional Geology

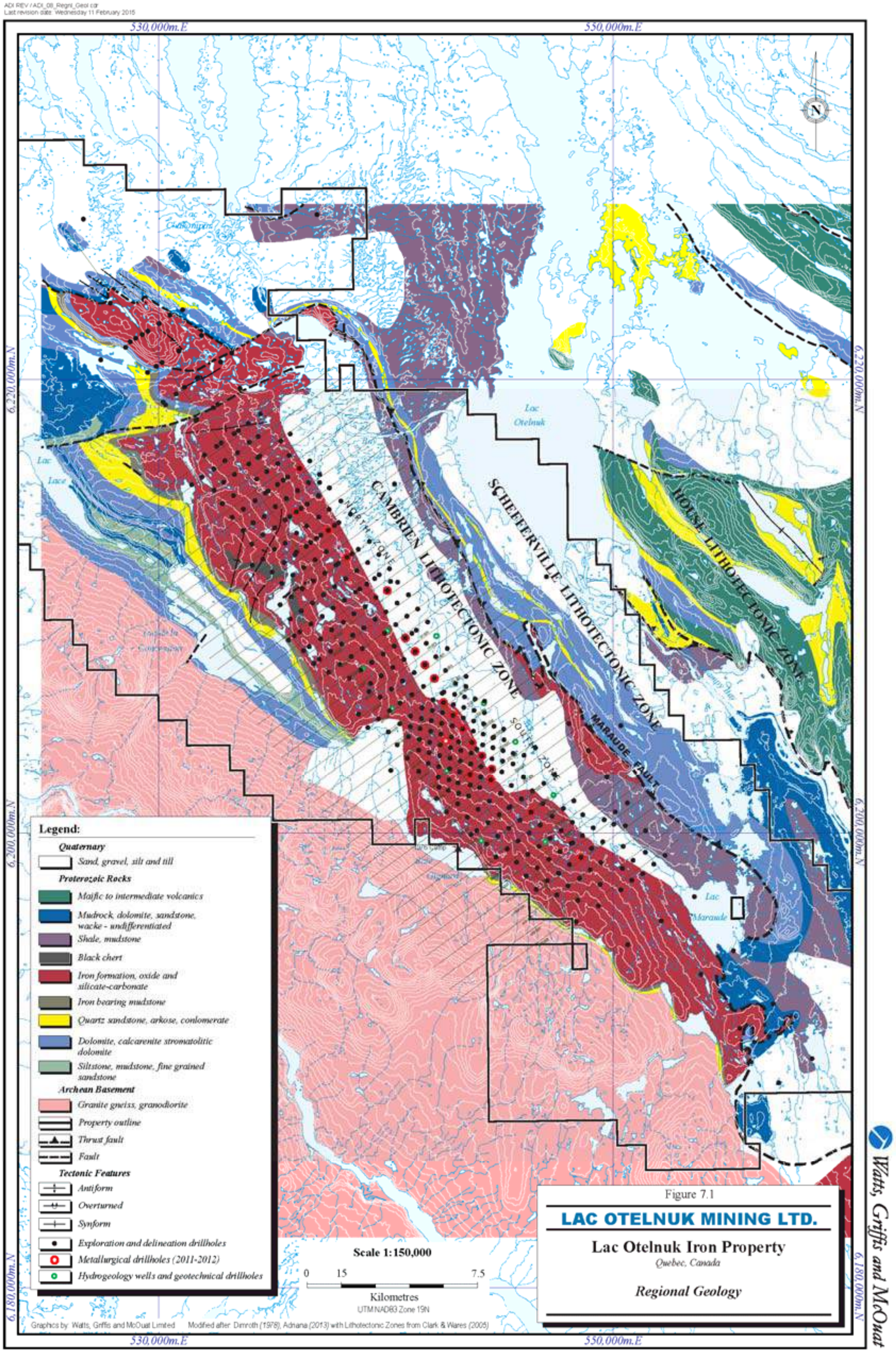
The Property is situated in the Churchill Province, of the Labrador Trough ("Trough") adjacent to Archean basement gneiss. The Trough, otherwise known as the Labrador-Quebec Fold Belt, extends for more than 1,100 km along the eastern margin of the Superior Craton from Ungava Bay to Lake Pletipi, Quebec. The belt is about 100 km wide in its central part and narrows considerably to the north and south.

The Trough comprises a sequence of Proterozoic sedimentary rocks, including iron formation, volcanic rocks and mafic intrusions. The southern part of the Trough is crossed by the Grenville Front representing a metamorphic fold-thrust belt in which Archean basement and Early Proterozoic platformal cover were thrust north-westwards across the southern portion of the southern margin of the North American Craton during the 1,000 Ma Grenvillian Orogeny (Brown, Rivers, and Callon, 1992). Trough rocks in the Grenville Province are highly metamorphosed and complexly folded. Iron deposits in the Gagnon Terrane, Grenville part of the Trough, include Lac Jeannine, Fire Lake, Mont-Wright, Mont-Reed, and Bloom Lake in the Manicouagan-Fermont area and the Luce, Humphrey and Scully deposits in the Wabush Labrador City. The high-grade metamorphism of the Grenville Province is responsible for re-crystallization of both iron oxides and silica in primary iron formation, producing coarse-grained sugary quartz, magnetite, and specular hematite schists (meta-taconites) that are of improved quality for concentration and processing.

LOM's Lac Otelnuk Property is located north of the Grenville Front in the Churchill Province where the Trough rocks have been only subject to greenschist or sub-greenschist grade metamorphism and the principal iron formation unit is known as the Sokoman Formation.

Figure 7.1 shows the regional geology of the Property area after Dimroth, 1978. This map, modified by WGM to consolidate various rock types, is based on mapping and geological compilation by Dimroth in the 1960s. Since then exploration by Adriana and perhaps others has led to some revisions to the known geology that is not incorporated into the map.

Figure 7.1 – Regional Geology



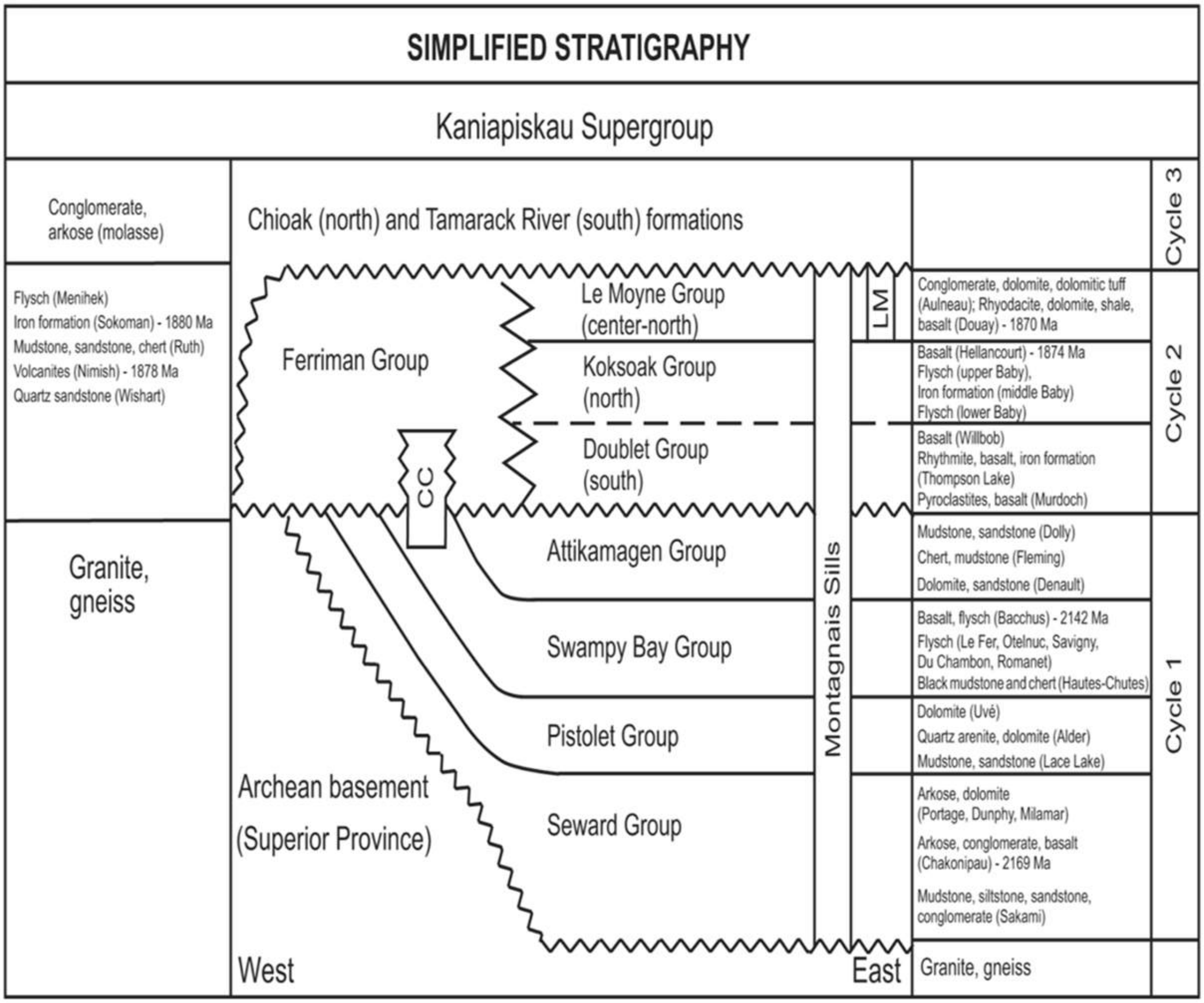
The recent synthesis by Clark and Wares (2006) develops modern lithotectonic and metallogenic models of the Trough north of the Grenville Front. The lithological units of interest on the Property due to their iron content are members of the Sokoman. The Sokoman Formation is the same iron-bearing unit that hosts the Gagnon or Grenville Terrane iron deposits, but in the central part of the Trough, where the Lac Otelnuk Property is located, it is less metamorphosed. The Sokoman Formation, member of the Ferriman Sub-Group, is overlain by the Menihek Sub-Group (mudstone and shale) and underlain by the Wishart Formation (quartzite), the Denault Formation (dolomite) and the Attikamagen Formation (shale). Wishart Formation (quartzite), and the Denault Formation (dolomite), a member of the Attikamagen Group. The Ferriman Group represents Cycle 2 of 3 volcaniclastic cycles defined in the Trough. The Attkamagen Group containing the Denault dolomite and the Fleming and Doly Formations, are mainly argillaceous rocks and are the upper members of the Cycle 1 sequence. The regional stratigraphic column after Clark and Wares (2006) is shown as Figure 7.2.

Clark and Wares (2006) defined lithotectonic zones ("LTZ") that divide the Trough or Orogen into subdivisions separated by tectonic discontinuities. These zones are defined by consistent lithologic assemblage or structure style traceable over large areas.

The Otelnuk Deposit is in the autochthonous Cambrien LTZ. The NW-SE trending Maraude Fault, underlying the east part of the Property separates the Cambrien LTZ from the Schefferville LTZ. The Schefferville LTZ in this area is comprised of the Pistolet Group of dolomite, sandstone and mudstone, a member of Cycle 1 which is thrust over the Cambrien LTZ along the Maraude Fault. The older Cycle 1 rocks also underlie the northwest margin of the Property.

Iron deposits in this part of the Trough are taconites, or weakly metamorphosed iron formation. Taconite iron deposits in the Trough include New Millennium's KéMag and LabMag deposits (Howells River Deposit), Cap-Ex Iron Ore Ltd.'s Greenbush Deposit, Century Iron Mines Corporation Rainy Lake Deposit and the December Lake deposit. The "Direct Shipping Ore" deposits located near Schefferville, and mined by IOC from 1954 to 1980, and adjacent deposits under renewed exploration and development by New Millennium, and Anglesey Mining Plc, subsidiary Labrador Iron Mines Limited, are taconite deposits that have been upgraded by supergene leaching.

Figure 7.2 – Schematic Stratigraphy of the Labrador Trough



7.2 Property Geology

7.2.1 General

The Property is situated on the western edge of the Trough. Archean gneisses form the basement and dip gently east. The basement gneisses are unconformably overlain by the gently northeast dipping sedimentary succession defined as the Kaniapiscau Supergroup, which includes the Ferriman Group and iron-bearing formations belonging to the Sokoman Formation. The sedimentary succession is peneplained and consequently wedge shaped. Towards the western edge of the Trough, the older, lower units of the sequence are successively exposed as the upper younger units have been removed by erosion. To the northeast, the Ferriman rocks are overlain by the Menihek shale and mudstone.

Within the Lac Otelnuk Property, for most part, the structural geology is very simple. However, in the far northern and north-western portion recent drilling has shown increased complexity with inferred thrusts or possible overturned folds. In general for the majority of the Property the iron formation is generally northwest-southeast striking, very flat-lying, monoclinic to gently inclined and rolling, with an average easterly dip of 5°. The individual members of the sedimentary succession are exposed as a series of benches or mesas in the west-central portion of the north half of the Property. The iron formation forms the top of the column in the eastern part of the Property and is mainly covered by glacial drift.

Metamorphism within the Property appears to be of low to moderate grade. Changes in grain size mineralogy and rock texture related to regional metamorphism are not visually detectable.

Within oxide iron formation units, the most distinguishable compositional feature through the local stratigraphic column is the rather abrupt changes from dominantly magnetite to dominantly hematite, and corresponding change of the silica from chert over to jasper. These oxidation potential variations and changes in iron grade define the sub-unit or member lithology units. The iron carbonate minerals, principally siderite and ferro-dolomite, are widespread but are more abundant in the upper and middle iron formation units. These features all appear to be related to primary deposition. There appears to be parts of three cycles present, Units 2, 3 and 4. Sub-units 2a, 3a, 4a are magnetite-rich, 2b, 4b are hematite-rich, the "c" sub-units and Unit 1 are lean (with respect to iron content). The average thickness of the iron formation is approximately 150 m from the top of Unit 1 to the base of sub-unit 4b and traced over a strike length of 42 km by widespread exploration drilling.

7.2.2 Lithology

The unit and sub-unit names and descriptions are modified after IOS'2007 program report for Adriana. Original nomenclature was derived from MPH reports based on drill programs conducted through the 1970s. Adriana's drilling programs have validated this terminology and descriptions and lithological codes based on this nomenclature are used in Adriana's drill logs and drill hole/assay database.

The average thicknesses are estimated from LOM-Adriana's drill holes, but inter-fingering of sub-units obscures estimation of sub-unit thicknesses and average thickness. Figure 7.3 is the Lac Otelnuk stratigraphic column after LOM, 2013. Lithologic Units and members are further described in Table 7.1.

Figure 7.3 – Lac Otelnuk Stratigraphic Column

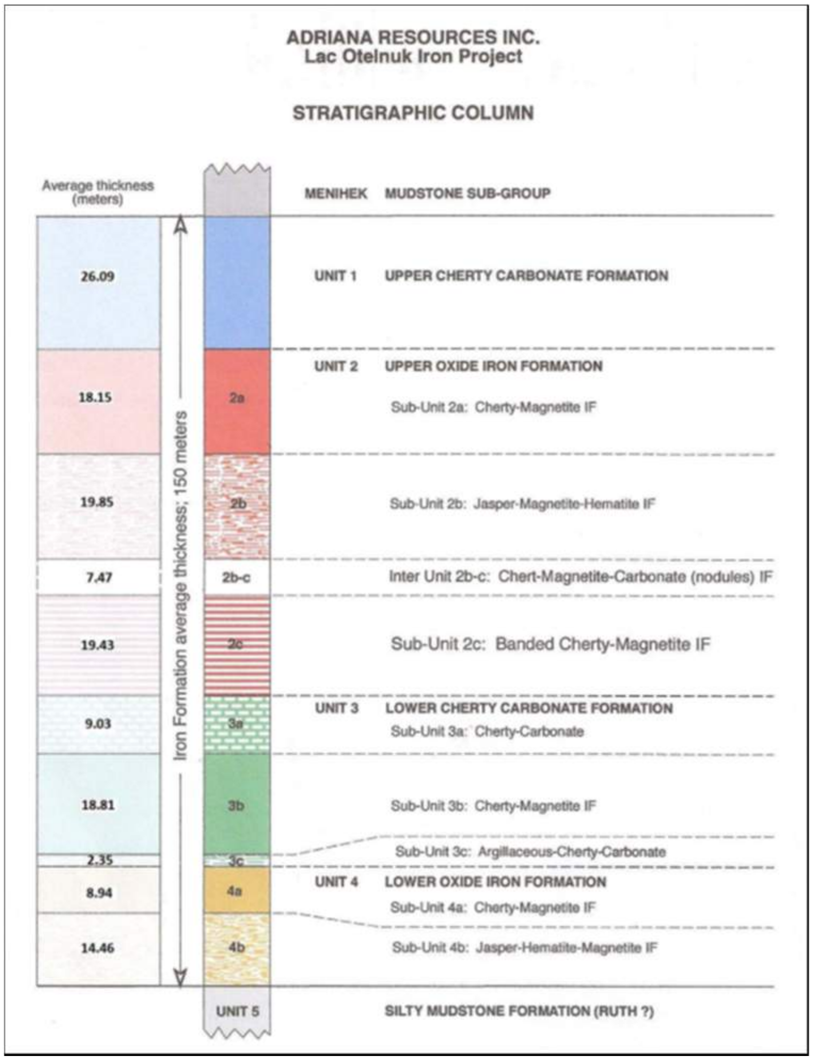


Table 7.1 – Stratigraphic Description

	Description
Menihek Sub-Group	The Menihek Sub-group is represented on the Property by grey and black shales. There is a gradation between the fine sedimentation of mudstone bands and the coarser sedimentation of silts. This formation is not magnetic.
Unit 1 (22-26 m, Avg.: 23 m) - Upper Cherty Carbonate Formation or Black Calcareous Jaspilite	This unit consists of alternating bands or fragments of green and white chert with sections of cherty carbonate. Magnetite exists in the form of disseminated grains weakly concentrated in a matrix of green-white chert. The concentration of magnetite increases towards the base.
Unit 2 – Upper Oxide Iron Formation or Upper Red Iron Sandstone and Jaspilite	This is the principal potential economic unit and comprises the following three distinctive sub-units: • Sub-unit 2a (1-28 m, Avg.: 14 m) - Cherty Magnetite or Black Calcareous Iron Sandstone Sub-unit 2a consists of alternating bands of homogenous micritic grey chert with thin bands of magnetite sub-parallel to bedding, locally slightly crenulated with a high reunification density. Thick bands of magnetite can show syn-sedimentary folds. Fine grains of magnetite are disseminated in the grey chert matrix. Carbonates are ubiquitous at less than 5 % but are found more commonly at the top of the sub-unit. • Sub-unit 2b (3-38 m, Avg.: 20 m) - Jasper Magnetite-Hematite or Upper Red Iron Sandstone and Jaspilite Sub-unit 2b is distinguished by its red color due to the presence of hematite dust within the jasper matrix. The matrix is cut by mm- to cm-thick bands of bedding parallel magnetite. The shades of red vary throughout the sub-unit due to changes in the concentration of hematite dust. Clusters of magnetite and hematite grains are also present scattered in the jasper matrix. Another feature of this sub-unit is the presence of oolitic and pisolitic horizons. The nucleus is almost always a piece of chert and / or jasper and the cortex is made up of thin layers of hematite and magnetite. These grains have a diameter of 0.5 to 1 mm for oolites and 1 to 15 mm for pisolites. The concentric texture of the oolites may be obliterated by recrystallization. Some concretions of carbonate and silica nodules may also be present, as well as clastic horizons of jasper and / or chert. There are also grey horizons that lack hematite dust but always have oolites, which are intermixed with the hematite layers. Such intercalation always occurs to the east of the baseline. Interfingering obscures minimum thickness and average thickness. • Inter Unit 2b/2c Inter-unit 2b/2c is a thin grey-black layer characterized by the absence of the red hematite dust of Sub-unit 2c and the absence of layering of Sub-unit 2c. The cherty matrix is micritic to sparitic. There are a few bands of white chert. Carbonate nodules are characteristic of this sub-unit and magnetite is present in disseminated form. The beginning of this sub-unit is marked by a small section of strongly folded chert, devoid of magnetite. • Sub-unit 2c (9-29 m, Avg.: 20 m) - Banded Cherty Magnetite Iron Formation or Black Iron Sandstone and Jaspilite Sub-unit 2c is characterized by a prominent package of iron-rich banded rock, segregated between bands (mm-cm scale) rich in magnetite and bands rich in silica and carbonate. The color varies between grey-white (chert and magnetite), green (chert), brown (altered chert), and yellow-white (chert carbonate). The intensity of color is dependent on iron content (e.g., light green to pale brown altered chert may contain iron silicate). When the concentration of iron silicate is significant, alteration may turn the rock bright red-brown. The carbonates may also give a brown color due to alteration. There is very little hematite and magnetite, which is mainly present in the form of thin bands parallel to bedding.

	Description
Unit 3 – Lower Cherty Carbonate or Black Jaspilite and Black Iron Sandstone	Similar to the sub-unit stratigraphy of Unit 2, three sub-units of Unit 3 can be clearly distinguished: 3a (chert carbonate), 3b (chert magnetite and carbonate) and 3c (chert carbonate clay). Sub-unit 3a is typically magnetite poor, but the last drill hole showed a variation in content with increased magnetite. Details are as follows: • Sub-unit 3a (1-15 m, Avg.: 9 m) - Cherty-Carbonate or Black Jaspilite Sub-unit 3a is a thin grey horizon with limited layering and rare bands of magnetite. In some holes, scattered magnetite grains are visible in the grey chert matrix. Generally this level is very similar to Unit 1, with a low concentration of magnetite. The main characteristic is spherical mm- to cm-length carbonate clusters. These clusters are white or reddish (siderite and / or ankerite). • Sub-unit 3b (11-25 m, Avg.: 20 m) - Cherty-Magnetite or Black Iron Sandstone Sub-unit 3b shows an increase of magnetite content in a grey chert matrix. It is very homogenous with few fractures. Magnetite exists as disseminated grains and thin bands sub parallel to bedding. The abundance of these bands is constant, but the thickness varies widely from a few mm to a few cm. Variations in thickness occur along individual bands. Jasper clasts are present in some boreholes. This sub-unit has a strong resemblance to 2a, which also has a layer of chert magnetite. • Sub-unit 3c (0.7–6 m, Avg.: 2.3 m) - Argillaceous-Cherty-Carbonate or Black Jaspilite Sub-unit 3c is massive in appearance with a fine-grained white-grey matrix. It is easily discernible from the facies above, having a high concentration of carbonate and clay laminations in carbonated chert horizons. Magnetite and hematite are very scarce, existing as a few scattered grains or fragments in the carbonate or clay. The contact with the iron formation is sharp and distinct.
Unit 4 – Lower Oxide Iron Formation or Lower Red Iron Sandstone and Jaspilite	Again three sub-units are distinguishable. From top to bottom these are: • 4a: Iron and chert magnetite formation. • 4b: Iron jasper-magnetite-hematite formation. • 4c: Iron jasper hematite-magnetite formation The unit is well exposed in outcrop and very persistent over most of the length of the Property. The three sub-units are quite distinguishable in outcrop. This is the lowermost iron-bearing unit. Sub-unit 4c was not intersected in Adriana's drilling program. • Sub-unit 4a (3.7-18, Avg.: 9 m) - Cherty-Magnetite Iron Formation or Black Iron Sandstone Sub-unit 4a is characterized by grains of magnetite and some hematite scattered in a chert matrix. The concentration of magnetite is significant. In the grey chert matrix, a few clasts of carbonate altered to a reddish color are observed. These clasts may contain a magnetite nucleus. There are layers of red hematite dust. This sub-unit is easily identifiable by its abundance of magnetite, which cause a high magnetic susceptibility. Its bluish color contrasts sharply with the deep red jasper of Sub-unit 4b. The boundaries of this sub-unit are therefore easily recognizable. • Sub-unit 4b (10-17 m, Avg.: 14.5 m) - Jasper-Hematite-Magnetite Iron Formation or Lower Red Iron Sandstone and Jaspilite Sub-unit 4b is characterized by alternating layers of red jasper, layers of highly concentrated fine-grained hematite, and layers of white chert. These characteristics make it a very recognizable horizon. All these layers are sub-parallel to bedding with angle variations ±10º. Magnetite is present in the form of disseminated grains in the bands of red jasper. These grains are locally concentrated to form fine bands parallel to the jasper banding. Hematite is common in the matrix in the form of cm-scale grey bands. The magnetic properties of these sections are likely due to the presence of martite (a mixture of magnetite and hematite), producing the dark metallic lustre.
Unit 5 - Silty Mudstone, Ruth Formation	This grey-green rock is very homogenous with few fractures and no iron mineralization. This unit is in contact with Sub-unit 4b, Sokoman Formation, no intercalations or conglomerates are apparent. The matrix is micritic, but there are a few silty bands with larger grains - passing from chemical to terrigenous sedimentation. This feature would assign these mudstones to the Ruth Formation, which underlies the Sokoman Formation. None of Adriana's South Zone drill holes cut the unit in its entirety. Maximum intersection length was approximately 11 m.

Figure 7.4 shows the geology of the Property in plan view. Figure 7.5 is a representative geological and drillhole cross section through the Main Zone.

Of particular note is the up-dip rise in surface elevation from northeast to southwest. This elevation differential would have a positive impact on stripping and waste rock removal in an eventual open pit mining operation.

7.3 Mineralization

Mineralization in the Lac Otelnuk iron formation consists mainly of magnetite (Fe_3O_4) and hematite (Fe_2O_3). Minor iron also occurs in silicates, siderite and ferro-ankerite. Iron oxide bands containing concentrations of magnetite and/or hematite alternate with grey chert of jasper. As described under Geology the iron formation and economically interesting parts of it are part of a gently east dipping interbanded sequence of rocks.

The Lac Otelnuk mineralization has been shown by drilling to extend over an area 36 km north-south and approximately 4 to 6 km east-west. The iron formation within this area, as described under the Geology Section of this report, is wedge shaped with the acute angle of the wedge to the west. This shape is the result of the gentle inclination of the beds to the east and the peneplaned surface. The iron formation sequence thus thins to the west. To the east it is overlain by a thickening sequence of younger Menihek Formation sediments. Units 2 and 3, see Table 7.1 and Figure 7.3 to Figure 7.5 are of economic interest and include the Mineral Resources (see Section 14 of this report). The maximum vertical thickness of this sequence is about 120 m; the average vertical thickness of the resources generally ranges from 90 m to 100 m.

In June 2011, SGS Minerals completed a geometallurgical study of the deposit. This work aimed to establish the domains of significantly different rock properties that can influence comminution and mineral separation. A robust statistical analysis of the available data associated with the 2007-2008 drilling campaign in the South Zone was carried out to account for the measurable variation that can be used for geometallurgical classification. Results are summarised in a report titled: "A Geometallurgical Investigation into Lac Otelnuk Iron Ore Deposit" prepared for Adriana Resources, Project CALR-11727-004. Fourteen composites that comprise iron ore with Davis Tube weight recoveries ("DTWR") greater than 18 % were proposed to be tested in the bench scale program; they were prepared from the ¼" rejects stored from the 2007-2008 drilling campaign. A reconnaissance testing for grindability was carried out on 39 samples mostly targeting lithological unit 2b, the thickest among the five investigated in this study.

Figure 7.4 – Property geology

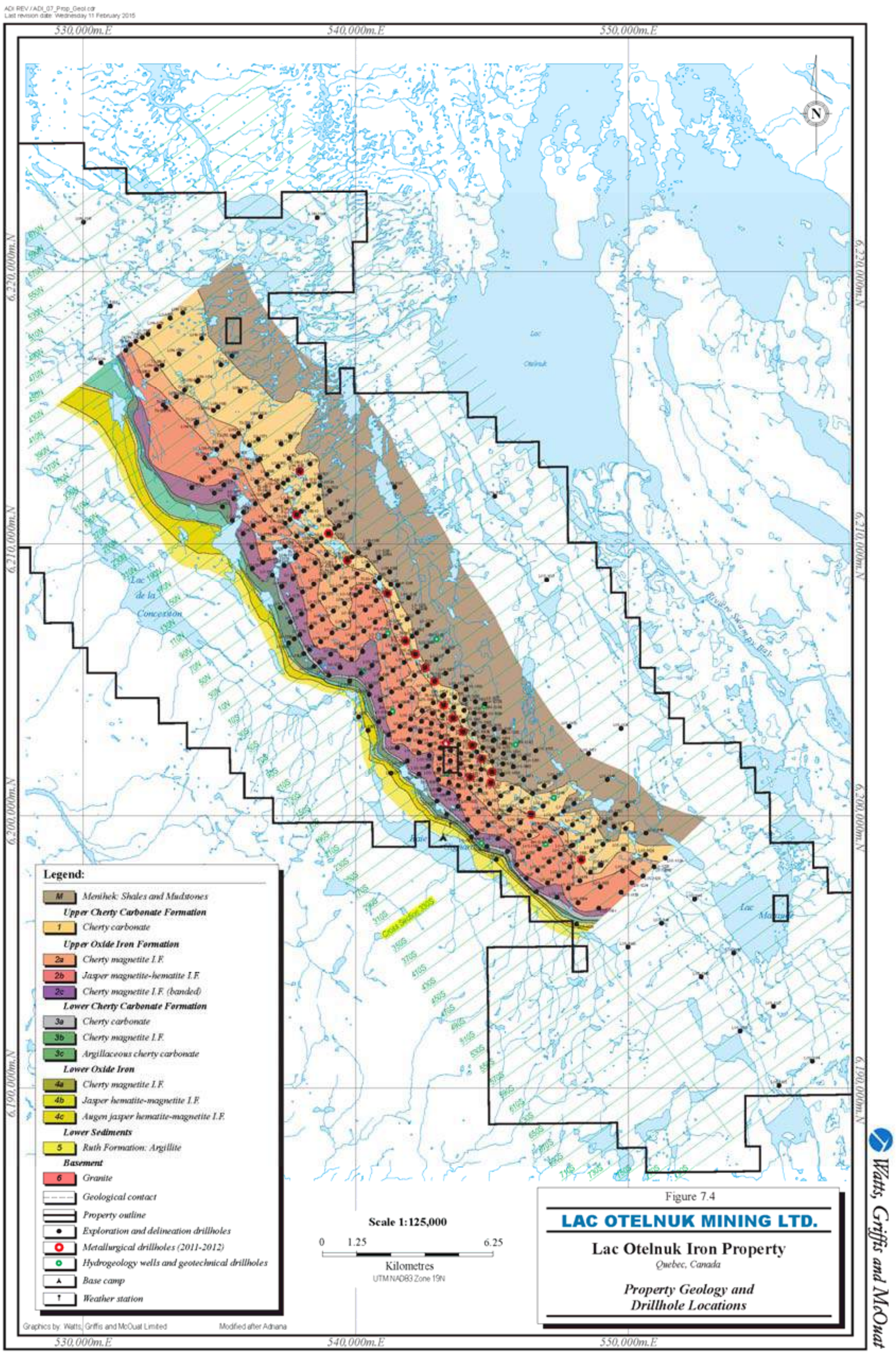
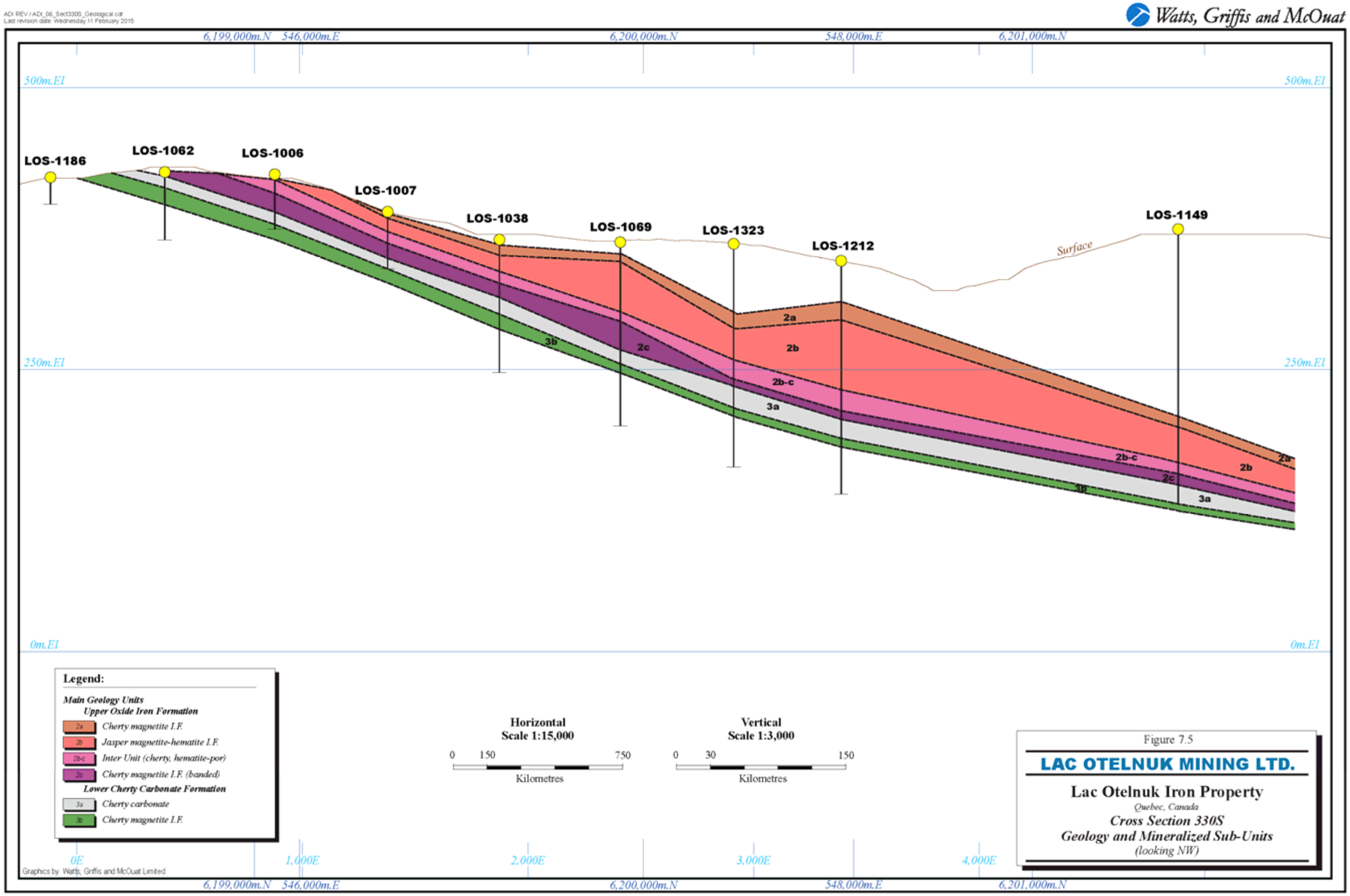


Figure 7.5 – Representative geological and drillhole cross section



From the statistical analysis it appears that subunits 2a and 2b are bimodal in terms of several elements including Fe, DTWR and SiO_2. This detail was one aspect WGM took note of during its interpretation towards completing the Mineral Resource estimate detailed in Section 14.0 of this report.

In September 2011, SGS Minerals carried out a mineralogical and iron deportment study for Adriana that was presented in a report titled: "An Investigation into the Mineralogical Characteristics of Ninety Eight Ore Variability Samples". SGS selected the samples to represent various potential "ore"-types on the Property.

X-Ray Diffraction ("XRD") analysis was performed on every fifth sample of the QEMSCAN data and for accurate speciation of the Fe-silicate and Fe-oxide mineral assemblage. The mineral distributions for the 98 samples were analyzed by QEMSCAN.

Table 7.2 lists the mineral assemblages for the 20 samples subject to XRD study. SGS Minerals also carried out an additional investigation for LOM in 2013 titled: "An Investigation into the Characteristics of 15 Composites from the Lac Otelnuk Deposit", dated January 15th, 2013.

From the QEMSCAN and XRD work SGS Minerals found the dominant mineral assemblage to consist of magnetite, hematite, ankerite, siderite, quartz, talc and minnesotaite, with minor calcite, chlorite and pyrite and other trace phases. Across the 98 samples examined in the 2011 investigation, magnetite abundance varies from 5.1 to 49.7 wt%; hematite from nil to 30.3 wt%; ankerite from 0.1 to 23.6 wt%; siderite from 0.1 to 30.3 wt%; quartz from 25.3 to 77.0 wt%; talc from nil to 8.7 wt%; minnesotaite from 0.4 to 32.9 wt%; calcite from nil to 3.3 wt%; chlorite from nil to 2.8 wt%; and pyrite from nil to 2.7 wt%. Other trace minerals typically occur in concentrations less than 1 wt%. The mineral distributions of the 98 samples were categorized into seven groups of similar mineral assemblies.

The mineral distribution for the High Magnetite Group is shown graphically as Figure 7.6.

SGS concluded from the 2011 investigation that:

- The results from the XRD and QEMSCAN are in close agreement. The samples are dominated by quartz (25.3 % to 77.0 %), followed by magnetite (5.1 % to 49.7 %), minnesotaite (0.4 % to 32.9 %), siderite (0.1 % to 30.3 %), hematite (nil to 30.3 %), ankerite (0.1 % to 23.6 %), talc (nil to 8.7 %), and calcite (nil to 3.3 %); and
- Fe-oxides carries most of the Fe (18.2 % to 96.6 %), the remainder is accounted for by siderite, and minnesotaite and lesser by ankerite and rare pyrite.

Table 7.2 – Summary of XRD Analysis (after SGS Minerals Services 2011)

Sample	Major	Moderate	Minor	Trace
193855	Quartz	Hematite, Magnetite	Ankerite	*Siderite, *Talc, *Stilpnomelane, *Calcite
193871	Quartz	Magnetite	Hematite, Ankerite, Siderite, Talc	*Stilpnomelane
194012	Quartz	Minnesotaite	Siderite, Ankerite, Magnetite	*Hematite, *Stilpnomelane, *Zussmanite
194093	Quartz	Magnetite	Hematite, Ankerite, Talc	*Stilpnomelane
194167	Quartz	Magnetite	Hematite, Ankerite, Siderite, Talc	*Stilpnomelane, *Calcite
194322	Quartz	Magnetite, Siderite	Minnesotaite, Ankerite	*Stilpnomelane, *Kaolinite
194334	Quartz		Magnetite, Hematite, Ankerite, Siderite, Talc	
194506	Quartz	Magnetite	Siderite, Ankerite, Minnesotaite	*Stilpnomelane, *Kaolinite
194548	Quartz	Magnetite	Ankerite, Siderite, Minnesotaite	
194603	Quartz	Siderite, Minnesotaite	Ankerite, Magnetite	*Stilpnomelane
194679	Quartz	Magnetite	Siderite, Minnesotaite, Ankerite	*Stilpnomelane
194821	Quartz	Magnetite	Ankerite, Siderite, Minnesotaite	*Talc, *Zussmanite
62510046	Quartz	Siderite	Minnesotaite, Magnetite, Ankerite	*Stilpnomelane, *Zussmanite
62510072	Quartz	Magnetite	Ankerite, Siderite, Talc, Minnesotaite	*Stilpnomelane, *Zussmanite
62510154	Quartz	Siderite	Magnetite, Minnesotaite	*Ankerite, *Talc, *Zussmanite
62510190	Quartz	Magnetite	Ankerite, Hematite, Talc	*Minnesotaite, *Stilpnomelane, *Siderite
62510196	Quartz	Magnetite, Siderite, Minnesotaite	Ankerite	*Stilpnomelane, *Zussmanite
62510314	Quartz	Magnetite	Ankerite, Hematite, Talc	*Siderite, *Stilpnomelane
62510346	Quartz	Magnetite	Minnesotaite, Ankerite, Siderite	*Stilpnomelane, *Kaolinite, *Hematite
6251051	Quartz	Magnetite	Ankerite, Siderite, Minnesotaite	*Stilpnomelane, *Kaolinite

Mineral Assemblage Relative proportions based on peak height
* Tentative identification due to low concentrations, diffraction line overlap or poor crystallinity.

Figure 7.6 – Mineral assemblage for the high-magnetite group (after SGS Minerals Services 2011)

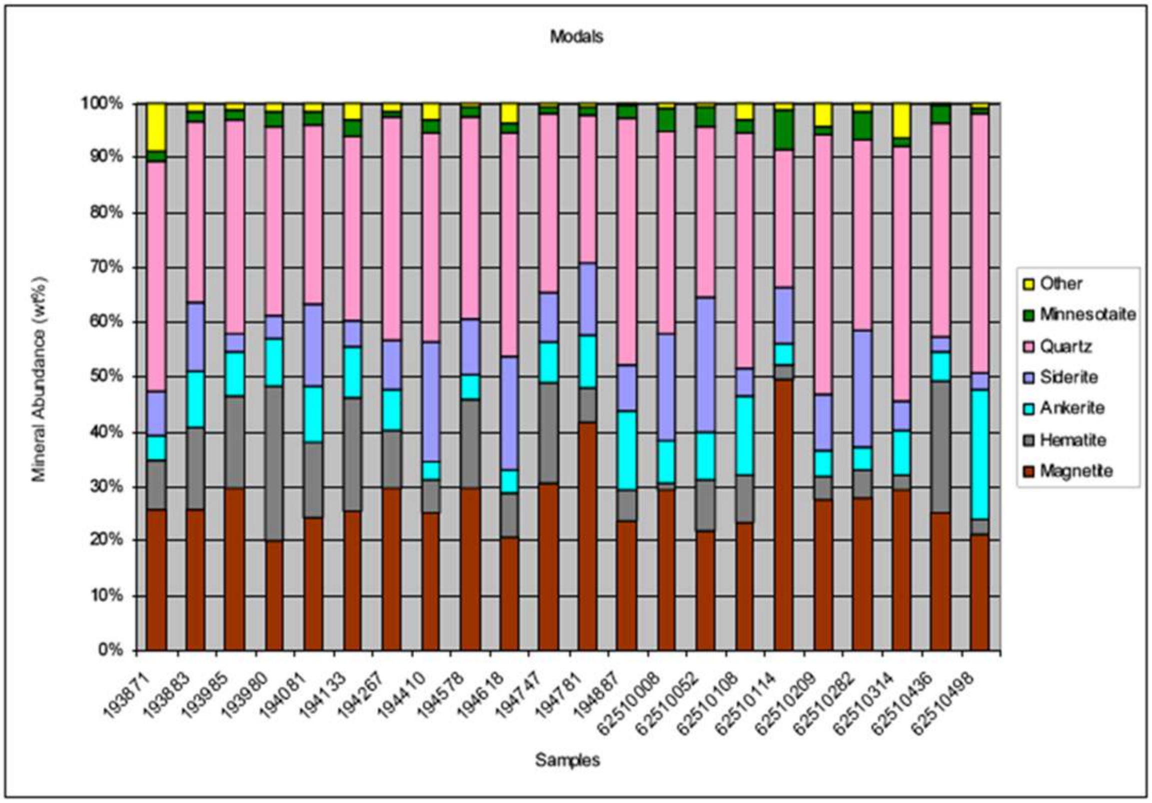


Table 7.3 provides a summary of the chemical composition of the various member units of the Lac Otelnuk stratigraphic sequences as indicated by whole rock analysis ("WRA") results for all of the 9,525, 2007 through 2012 drill core samples. In this table MagFe and DTWR is from DT tests where results are available or estimated and adjusted from Satmagan results when Satmagan in lieu of DT tests is available.

Figure 7.7 shows patterns for silica, alumina, TFe and MagFe along a typical drillhole trace. The Menihek Formation is missing in this example. Stratigraphically, and almost always, structurally, on the LOM's Property Menihek Formation sits above Unit 1 (U1). Higher levels of alumina are associated with Unit 5, and Unit 4. Unit 3c is often associated with a distinctive alumina positive anomaly as is the lowermost part or contact of Unit 2b. The alumina peak associated with 3c corresponds often to minor shale components. Alumina characteristically is higher in Units 2b, 2a, and 1, than in 2c, 3a and 3b. The contact of 2b and 2c is also associated with a distinctive change in silica level. The contact area is often associated with a slight increase in silica. Silica, upwards into 2b, decreases slightly before increasing again upwards into 2a. A transitional, often thin member 2b-c has been defined in many of the drillholes by logging. It consistently is located between 2b and 2c and coincides with the sharp change in alumina and silica levels.

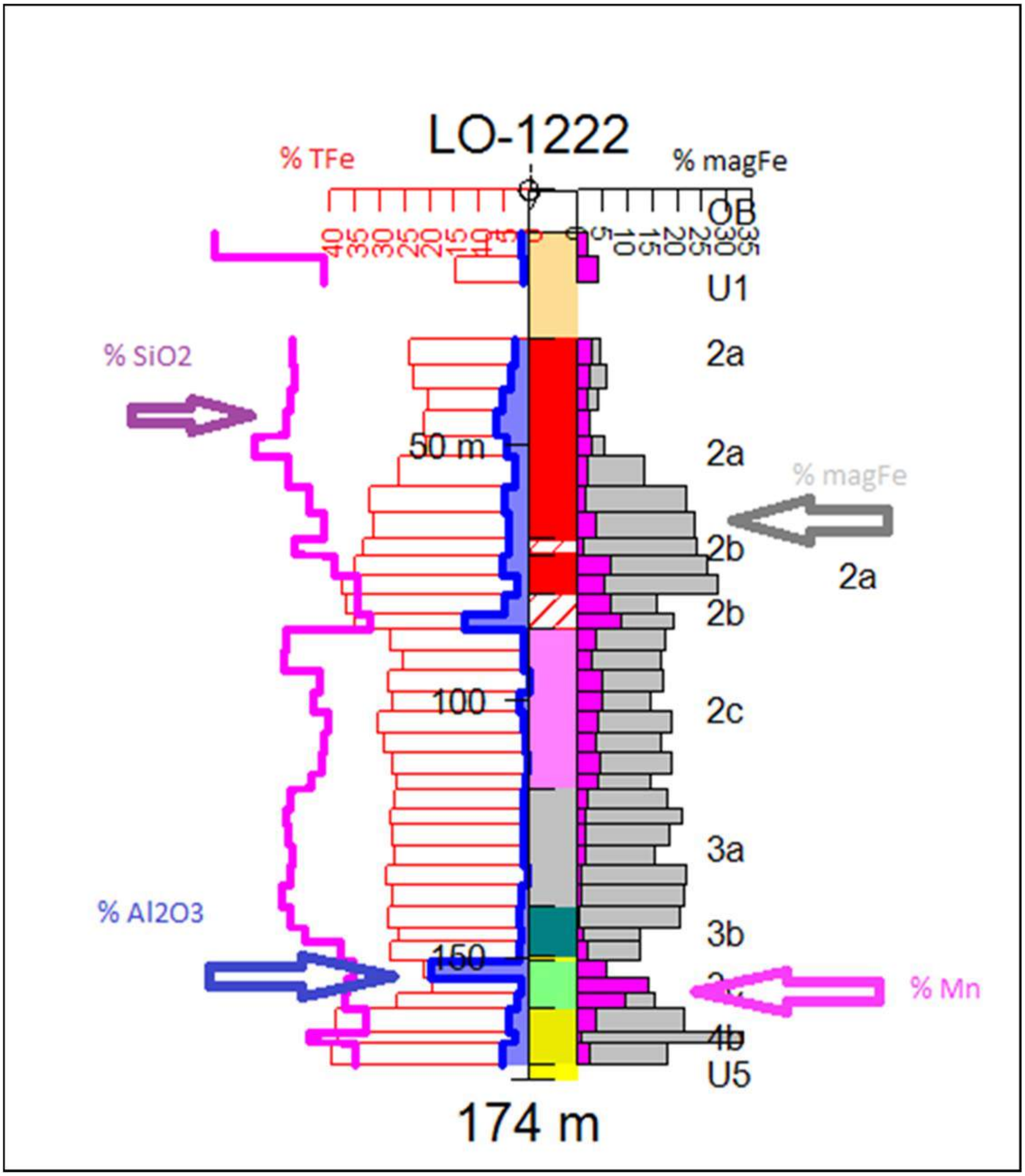
Table 7.3 – Average Composition of Adriana's 2007 to 2012 Drill Core Samples

Lith	Sample Count	TFe (%)	MagFe[1] (%)	DTWR[2] (%)	SiO_2 (%)	Al_2O_3 (%)	TiO_2 (%)	Na_2O (%)	K_2O (%)	MgO (%)	CaO (%)	Mn (%)	P (ppm)	LOI (%)	Sample Count S	S (%)	Sample Count SG	SG pycn
2a	828	28.56	17.05	24.84	44.59	0.19	0.02	0.03	0.06	2.21	3.03	0.68	0.03	8.19	591	0.03	229	3.39
2b	1493	33.96	16.61	24.06	40.14	0.20	0.02	0.04	0.04	1.71	2.62	1.11	0.04	5.28	1067	0.01	395	3.60
2b-c	647	26.64	15.16	22.12	47.10	0.06	0.01	0.02	0.01	2.18	4.09	0.86	0.03	7.42	495	0.01	208	3.37
2c	1539	27.77	15.24	22.18	42.70	0.05	0.01	0.02	0.01	3.07	3.22	0.81	0.04	10.26	1057	0.02	372	3.39
3a	640	25.88	10.67	15.64	47.75	0.04	0.01	0.02	0.01	2.40	5.13	0.52	0.01	7.08	425	0.01	182	3.34
3b	1226	27.76	19.11	27.82	48.19	0.07	0.01	0.02	0.02	2.75	3.71	0.25	0.02	5.27	786	0.03	338	3.38
3b-c	6	24.96	13.54	19.83	44.32	0.06	0.01	0.02	0.03	3.45	4.75	0.42	0.02	11.10	4	0.07	2	3.32
3c	803	22.98	1.75	2.72	36.77	0.39	0.02	0.03	0.06	4.30	4.35	1.33	0.07	18.84	691	0.50	248	3.30
4a	632	31.49	12.97	18.99	43.12	0.11	0.01	0.03	0.03	2.07	2.60	0.81	0.03	5.70	403	0.14	197	3.53
4b	1147	37.35	5.59	8.61	35.18	0.24	0.02	0.04	0.08	1.56	2.22	1.98	0.03	4.22	802	0.08	349	3.77
BDM	4	7.17	0.00	0.05	60.48	12.91	0.52	1.45	3.45	2.67	1.08	0.17	0.12	6.29	4	0.42	0	
DOL	2	1.28	0.00	0.00	26.35	0.95	0.04	0.02	0.06	15.10	21.25	0.07	0.05	33.35	2	0.05	0	
Fault Zn	1	19.20	10.01	15.61	47.90	0.22	0.04	0.06	0.05	2.91	8.29	0.86	0.01	12.30	0		1	3.07
Menihek	5	4.46	0.06	0.24	61.98	15.06	0.59	1.12	4.42	3.13	0.56	0.10	0.14	6.09	5	0.44	2	2.79
SHL	40	25.86	0.01	0.10	28.71	0.32	0.02	0.03	0.04	4.58	4.45	1.52	0.11	22.72	40	0.29	15	3.39
TRZN	2	29.34	4.47	7.07	37.75	0.04	0.01	0.01	0.02	3.65	2.61	0.73	0.07	13.55	2	0.02	0	
U1	199	20.50	2.06	3.20	46.75	0.56	0.03	0.04	0.20	4.18	3.73	0.55	0.04	14.57	160	0.06	33	3.19
U5	312	9.65	0.05	0.20	58.49	12.59	0.54	0.48	5.23	2.94	0.38	0.59	0.05	4.49	222	0.02	45	2.90
Total	**9526**														**6756**		**2616**	

Notes: 1. MagFe estimated from DT tests or Satmagan. Where determined by Satmagan values have been normalized to DT results.
2. DTWR from DT tests. Where DT not available DTWR was estimated from Satmagan determinations. See text of report.
3. Averages and counts based on sample assays 2007 through 2013

Mn also shows a characteristic pattern. Mn is higher in 3c, 4, 2c and 2b and lower in 2a and 3b. These characteristic patterns assist with geological interpretation and modelling.

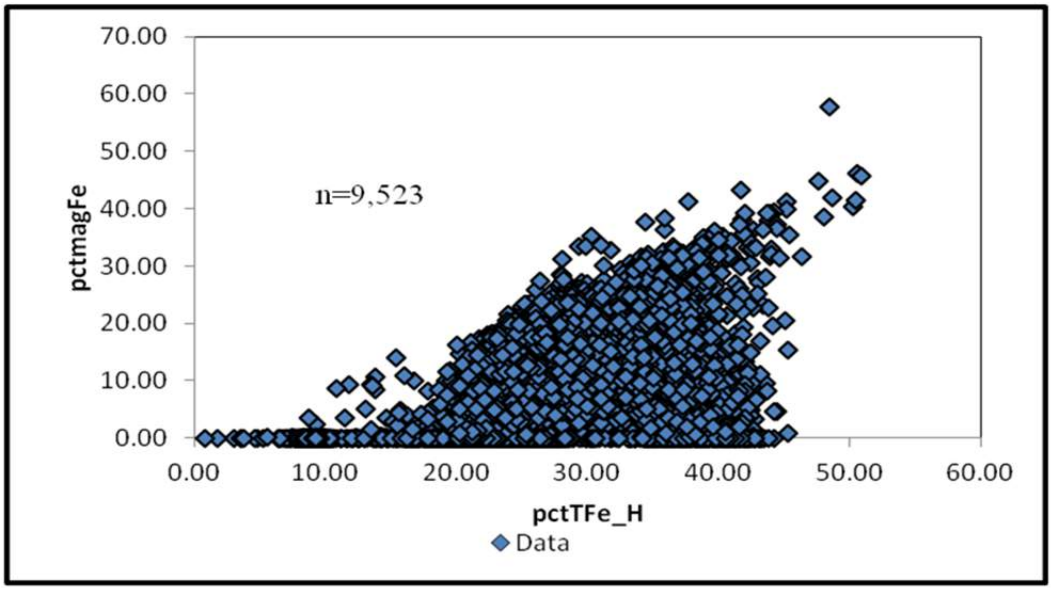
Figure 7.7 – Typical geochemical patterns along drillholes



MagFe and %DTWR for sub-units and samples vary on the basis of overall Fe concentration in the samples, absolute percent hematite and magnetite as well as hematite/magnetite ratios. There is no simple relationship between %TFe and %MagFe or

%TFe and %DTWR (Figure 7.8) due to variations of magnetite: hematite ratios throughout. Sub-units 2c to 3b appear to be distinctively low in aluminum. This low aluminum is evident both in Head assays and DTCs. Sub-unit 4b is particularly high in iron, but magnetic iron is low because most of the iron in the sub-unit is in the form of hematite.

Figure 7.8 – %MagFe vs. %TFe



For Adriana's drilling program 2007 to 2010, see Section 11, Davis Tube tests on drill core samples were routinely carried out. For the 2011 drilling programs a changeover was made to complete Satmagan determinations in the place of the DT tests. For QA/QC purposes some DT tests were still maintained and certain samples had both DT and Satmagan determinations. Figure 7.9 shows the relationship between MagFe determined by Satmagan and determined by DT for 2011 and 2012 samples and 2013 Check assay corrections where both techniques were used.

Clearly there is strong correlation between the two types of measurements with minor scatter. For a few samples Satmagan measurements are significantly different from DT. Some of these results represent errors that still persist even after the 2013 Check assay program (Section 11.10 of this report). Where MagFe from DT is significantly higher than MagFe from Satmagan it is possible the Davis Tube Concentrates contain some hematite. Statistically MagFe from Satmagan is biased very slightly higher than MagFe from the DT tests. This may be due to slight calibration error for the Satmagan instrumentation or perhaps minor loss of fine magnetite in the DT tests. This bias appears to have diminished slightly from previously with the inclusion of 2012 and 2013 assay results.

Figure 7.9 – MagFe from DT vs. MagFe from Satmagan for all samples where both determinations were completed

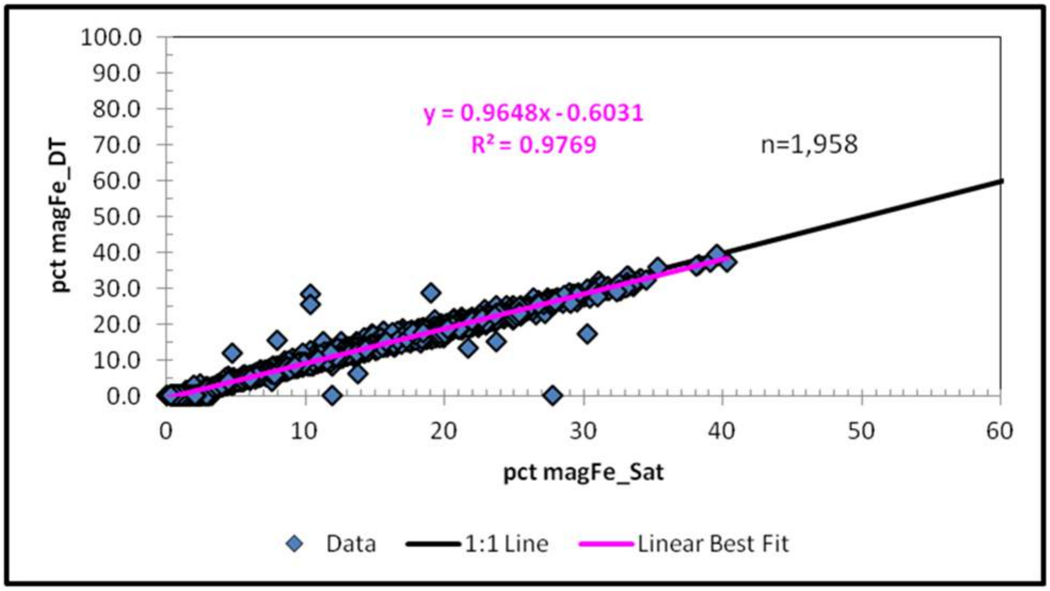
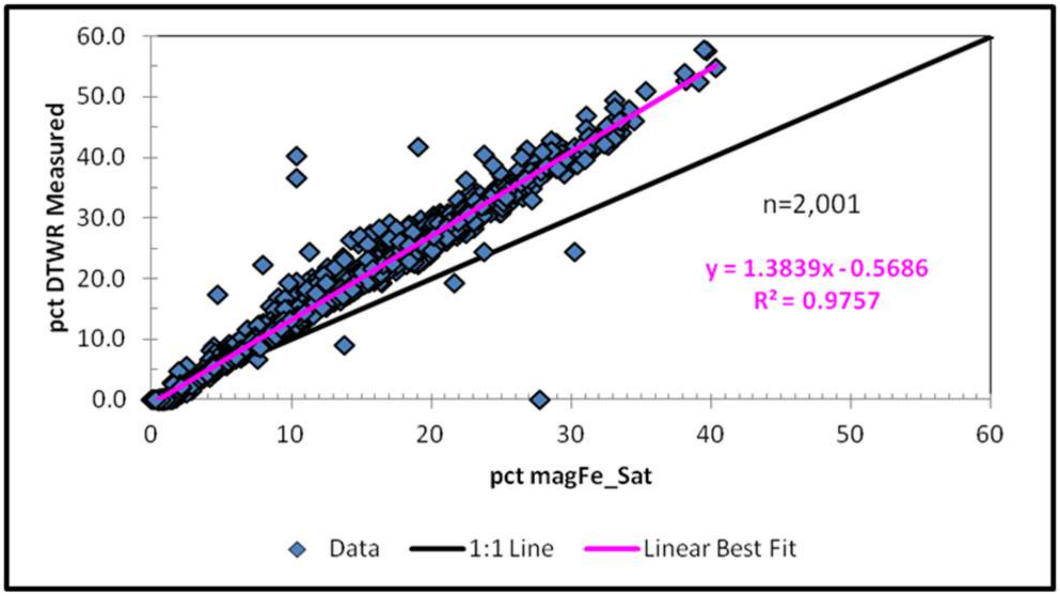


Figure 7.10 shows the relationship between measured DTWR from DT tests and %MagFe determined by Satmagan.

Measured %DTWR from DT tests and %MagFe from Satmagan are also generally strongly positively correlated. Again, a few assay errors are still present that persist even after the 2013 Check assay program (a few samples that were selected for assay checks inadvertently did not get re-assayed). For the purposes of the Mineral Resource estimate and to maintain context from the previous estimate, WGM has estimated DTWR and MagFe Final from Satmagan MagFe when no DT tests results are available. These estimates were made using the relationships defined by the linear best fit equation for measured DT MagFe and Satmagan MagFe and DTWR and Satmagan MagFe. The adjustment for MagFe resulted in a slight lowering of the raw Satmagan MagFe values reflecting the fact that raw MagFe values from Satmagan are statistically slightly higher than MagFe from DT tests as shown on Figure 7.9.

Figure 7.10 – Measured DTWR from DT tests vs. %MagFe from Satmagan



WGM used the regression functions derived prior to the 2012 assay results rather than the ones shown on Figure 7.9 and Figure 7.10. The inclusion of 2012 and 2013 results has changed the relationships between MagFe derived from Satmagan and DT slightly. WGM views these change in the relationship and regression functions as immaterial and has elected for continuity to use the mathematical relationship delineated in the previous report:

- $\begin{Bmatrix} \text{\%MagFeDT for Mineral Resource estimate} \\ \text{estimated from Satmagan} \end{Bmatrix} = 0.9645 \times \text{\%MagFe\_Sat} - 0.6291$
- $\begin{Bmatrix} \text{\%DTWR for Mineral resource estimate} \\ \text{estimated from Satmagan} \end{Bmatrix} = 1.3862 \times \text{\%MagFe\_Sat} - 0.6206$

8.0 DEPOSIT TYPES

The Lac Otelnuk deposits are composed of iron formations of the Lake Superior-type. This type of iron formation consists of banded sedimentary rocks composed principally of bands of magnetite and hematite within quartz (chert)-rich rock, with variable amounts of silicate, carbonate and sulphide lithofacies. Such iron formations have been the principal sources of iron throughout the world (Gross, 1995). Table 8.1, after Eckstrand, editor (1984), presents the salient characteristics of the Lake Superior-type iron deposit model.

Lithofacies that are not highly metamorphosed or altered by weathering are referred to as taconite. Herein taconite can be both magnetite and/or hematite dominant. The Lac Otelnuk deposits are examples of taconite-type iron formation. Strongly metamorphosed taconites are known as meta-taconite, particularly when magnetite-rich or itabirite, when hematite-rich. The iron deposits in the Grenville part of the Labrador Trough in the vicinity of Fermont and Wabush are meta-taconite.

A number of models have been considered for the origin of iron formation and associated lithofacies. According to Gross (1995), the two principal genetic but controversial models are:

1. Volcanogenic and hydrothermal effusive or exhalative; and
2. Hydrogenous-sedimentary with derivation of the iron, silica and other constituents by deep weathering of a landmass.

Gross reports that iron-oxidizing micro-organisms might have played a role. Oolites are generally common in iron formation.

Gross (1968) suggested that hot springs along a volcanic arc could have been an adequate source of iron and silica to form the iron formations in the 1,200 km long Labrador Trough in about 50,000 years. Precipitation and deposition of iron minerals in marine sub-basins is controlled largely by pH and eH (redox potential) with different mineral species stable under different conditions. James (1954) used this concept in the study of iron formations in the Lake Superior region and defined four primary facies of iron formation: oxide, silicate, carbonate and sulphide. The different facies of iron formation are, however, rarely found in simple, distinct successive contiguous zones.

Table 8.1 – Deposit Model for Lake Superior - Type Iron Formation (After Eckstrand - 1984)

Commodities	**Fe (Mn)**
Examples: Canadian - Foreign	Knob Lake, Wabush Lake and Mount Wright areas, Que. and Lab. - Mesabi Range, Minnesota; Marquette Range, Michigan; Minas Gerais area, Brazil.
Importance	Canada: the major source of iron. World: the major source of iron.
Typical Grade, Tonnage	Up to billions of tonnes, at grades ranging from 15 to 45 % Fe, averaging 30 % Fe.
Geological Setting	Continental shelves and slopes possibly contemporaneous with offshore volcanic ridges. Principal development in middle Precambrian shelf sequences marginal to Archean cratons.
Host Rocks or Mineralized Rocks	Iron formations consist mainly of iron- and silica-rich beds; common varieties are taconite, itabirite, banded hematite quartzite, and jaspilite; composed of oxide, silicate and carbonate facies and may also include sulphide facies. Commonly intercalated with other shelf sediments: black
Associated Rocks	Bedded chert and chert breccia, dolomite, stromatolitic dolomite and chert, black shale, argillite, siltstone, quartzite, conglomerate, red beds, tuff, lava, volcaniclastic rocks; metamorphic equivalents.
Form of Deposit, Distribution of Ore Minerals	Mineable deposits are sedimentary beds with cumulative thickness typically from 30 to 150 m and strike length of several kilometres. In many deposits, repetition of beds caused by isoclinal folding or thrust faulting has produced widths that are economically mineable. Ore mineral distribution is largely determined by primary sedimentary deposition. Granular and oolitic textures common.
Minerals: Principal Ore Minerals - Associated Minerals	Magnetite, hematite, goethite, pyrolusite, manganite, hollandite. - Finely laminated chert, quartz, Fe-silicates, Fe-carbonates and Fe-sulphides; primary ormetamorphic derivatives
Age, Host Rocks	Precambrian, predominantly early Proterozoic (2.4 to 1.9 Ga)
Age, Ore	Syngenetic, same age as host rocks. In Canada, major deformation during Hudsonian, and in places, Grenvillian orogenies produced mineable thicknesses of iron formation.
Genetic Model	A preferred model invokes chemical, colloidal and possibly biochemical precipitates of iron and silica in euxinic to oxidizing environments, derived from hydrothermal effusive sources related to fracture systems and offshore volcanic activity. Deposition may be distal from effusive centres and hot spring activity. Other models derive silica and iron from deeply weathered land masses, or by leaching from euxinic sediments. Sedimentary reworking of beds is common. The greater development of Lake Superior-type iron formation in early Proterozoic time has been considered by some to be related to increased atmospheric oxygen content, resulting from biological evolution.
Ore Controls, Guides to Exploration	1 Distribution of iron formation is reasonably well known from aeromagnetic surveys. 2. Oxide facies is the most important, economically, of the iron formation facies. 3. Thick primary sections of iron formation are desirable. 4. Repetition of favourable beds by folding or faulting may be an essential factor in generating widths that are mineable (30 to 150 m). 5. Metamorphism increases grain size, improves metallurgical recovery. 6. Metamorphic mineral assemblages reflect the mineralogy of primary sedimentary facies. 7. Basin analysis and sedimentation modelling indicate controls for facies development, and help define location and distribution of different iron formation facies.

Author G.A. Gross

Commonly, rapidly fluctuating eH and pH environments have resulted in interlaying of different iron mineral species requiring contrasting stability fields and diagenesis has added at least some complication to mineral distribution. Remarkably different facies are often sharply delineated with layers of hematite juxtaposed, but in sharp contact with beds of magnetite. Iron silicates are not often as clearly delineated, with the silicates as a group precipitating over a wide range of redox potential. As a result, silicate facies overlaps substantially with those of oxides and carbonate.

For iron formation to be mined economically, there will be a minimum iron content required at a given market price, the iron oxides must also be amenable to concentration (beneficiation) and the concentrates produced must be low in manganese and deleterious elements such as silica, aluminium, phosphorus, sulphur and alkalis. For effective bulk mining, the silicate and carbonate lithofacies and other non-economic rock types interbedded within the iron formation must be sufficiently segregated from the economic iron-bearing areas.

9.0 EXPLORATION

WGM has relied, for our descriptions of exploration program results, solely on the basis of historic reports, notes and communications with LOM and Adriana personnel and various geophysical contractors. Additional results and descriptions have been summarized in previous WGM NI 43-101 Technical Reports and two recent reports on Adriana and LOM's 2011 Phase I and II drill and 2012 programs (Adriana, 2012, Gestion Otelnuk, 2011 and Gestion Otelnuk & LOM, 2013) and previous reports on its 2007, 2008 and 2010 programs by IOS and Gestion Otelnuk, filed for assessment with the MRNF.

Historic exploration is summarized under the History Section of this report. Adriana commenced exploration on the Property in 2007. Since inception Adriana and LOM's programs were planned and supervised by Frank Condon, P.Eng., and Gilles A. Tremblay, P.Eng., experienced Professional Geological Engineers and consultants to Adriana and LOM.

9.1 General

Adriana's exploration programs from inception in 2007 to 2011 have consisted mainly of diamond drilling programs described in Section 10.0. The only other components other than this diamond drilling have been an airborne imaging survey to build a digital terrain model ("DTM") carried out by Eagle Mapping in 2008 and a reconnaissance geological mapping program conducted in 2011 covering a part of the NW part of the Property. Additionally a program to search out and identify historical (1970s) drill collar locations for survey was conducted in 2011 in parallel to regular drill collar surveying. This surveying, similar to the surveying for Adriana's drillhole collars was carried out by Groupe Cadoret Arpenteurs-Geometres ("Cadoret") based in Sept-Îles, Quebec. Adriana has not carried out any geophysical surveying.

One other component was the re-assay of 15 samples of archived drill core from the 1970s. These samples were sent to SGS-Lakefield in 2007. In summer 2013 a Check assaying program spanning all field programs 2007 through 2012 was conducted.

9.1.1 Eagle Mapping Imaging Program 2008

Eagle Mapping in 2008 flew an airborne survey in 2008 at an elevation of 1920 m ASL and acquired 530 colour photos at a scale of 1:10,000. Aerial triangulation used the airborne GPS data in conjunction with the ground survey coordinates and a large number of common tie points on each photo to provide the photo orientation parameters necessary to capture mapping data in the correct projection. The result of aerial triangulation is a series of geo-referenced stereo models for topographic and feature collection in 3D. The mapping project was referenced to NAD83, UTM Zone 19N datum. Elevations were provided in metres above mean sea level.

Eagle Mapping scanned the film at 16 microns. The scanned imagery was aero-triangulated to produce geo-referenced stereo models for mapping compilation. The stereo models were used to compile a digital terrain model ("DTM"). This digital terrain model comprised of gridded XYZ points and break-lines (polyline features of dramatic changes in the terrain). Planimetric features, such as: rivers, streams, lakes, marsh, tree lines, scrub lines, roads, trails and buildings were also captured. Final maps were produced in AutoCAD at 1:1,000-scale. Originally 1 metre contours were completed to cover the South Zone only and for outlying areas, 10 m contours were generated. Final plots were printed at 1:5,000-scale for planning purposes.

In 2010, the 1 m contour coverage was extended to the North Zone and in 2012 the 1 m contour coverage was extended to cover the the west edge of the Property and the proposed tailings dam.

The Eagle Mapping DTM was used for the Mineral Resource estimate to define the surface topographic profile, see Section 14.

9.1.2 2010 Reconnaissance Geological Mapping Program

In summer-2010 a reconnaissance geological mapping program covering a part of the north part of the Adriana Otelnuk Property was carried out. A report authored by Julien Helou, and filed for assessment with the MRNF is titled: "Rapport d'exploration Géologique du nord de la formation de fer du lac Otelnuk". The program comprised 23 mapping traverses to map lithology and geological structure. An aggregate of 58 grab samples were collected of representative mineralization and lithologies and submitted to SGS-Lakefield for analysis for major elements by XRF. The program was successful in mapping and sampling low-dipping iron formation similar to that known on southern parts of the Property. Recommendations from this work included the selection of collar locations for diamond drill holes to advance the delineation of the iron formation in the north part of the Otelnuk Property.

9.1.3 Search for and Survey of Historic Drill hole Collars

Cadoret was initially contracted in 2007 to survey in the collar locations for Adriana's initial program using differential GPS. Since that time they have returned to the Property every drill program to pick-up the collar locations. During their first visit to the Property in September 2008 they surveyed two of the historic collar from the 1976 drill program. In September 2010 they surveyed the collars for 32 of the historic collars from the 1970 to 1976 drill programs. In total all but two of the historic collars (76-210S-3 and 76-50S-1) have been located and surveyed.

The details of whether Cadoret or the project geologists located the collars are not known to WGM. It is also not known what was actually located and surveyed in each particular case– whether casing or drill set-up logs or drillers refuse.

WGM has previously recommended that these details be documented.

10.0 DRILLING

WGM has relied for our descriptions of drilling programs and results solely on the basis of historic reports, notes and communications with LOM and Adriana personnel including Gestion Otelnuk & LOM (2013), Adriana (2012) and Gestion Otelnuk (2011) that summarize, respectively the 2012, 2011 Phase I and II diamond drill programs. Additional results and descriptions have been summarized in previous WGM NI 43 101 Technical Reports.

10.1 Previous Drilling

10.1.1 Adriana's 2007 Drilling Program

Following recommendations made by WGM in its 2007 report, Adriana initiated planning to drill the South Zone. General planning and supervision of the two year drilling program was contracted to Minroc Management Limited ("Minroc"), an Ontario registered Private Company managed by Gilles A. Tremblay, P.Eng. The aim of the program was to drill test an area of the South Zone 250 m wide by 9 km along strike with vertical drillholes centred on a 600 m by 500 m grid aligned to the historic MPH cut grid. The first few Adriana drillholes were designed to test the entire iron formation stratigraphy, but due to the poor performance of the drills most of the 2007 drilling targeted only the upper magnetite-rich iron formation sub-units 2a, 2b and 2c. The readily identified Sub-unit 3a was used as a marker to end the drillholes.

Field operations, including drilling, core logging, core splitting, expediting and camp operations were supervised IOS.

Energold Drilling Corp. ("Energold") provided two lightweight drills, ancillary equipment, two drill crews and a drill foreman. Core size was BTW (4.20 cm) diameter. Initially, drill and crew moves were attempted using All-Terrain Vehicles ("ATV") equipped with trailers. This proved unworkable and subsequently an A-Star 350 B2 helicopter was chartered from Sept-Îles-based Canadian Helicopters Limited to move the crews, equipment and core. The drill program started July 18th and the last hole was completed September 25th. Drilling totaled 27 holes aggregating 2,195 m.

Table 10.1 summarizes Adriana's drilling programs.

Table 10.1 – Summary of LOM and Adriana Drilling Programs

Program	Area	Number of Holes	Aggregate Meterage (m)
2007	South Zone	27	2,195
2008	South Zone	41	5,205
2010	South Zone	41	5,874
2011 – Phase I	South and North Zones	26	3,592
2011 - Phase II	South and North Zones	83	11,702
2012	South and North Zones	196	22,249
Total		**414**	**50,229**

Notes: Three piezometer wells drilled for pump tests are excluded.
Meterages and drillhole numbers are approximate depending on whether abandoned and re-drilled drillholes are counted or not counted.

Julien Hélou, a geologist working for IOS completed most of the core logging. At the end of the program IOS completed a program report titled: "Drilling Campaign to Define Resources: Lake Otelnuk Iron Deposit Presented to Frank Condon, P.Eng. Adriana Resources by Dennis Lahondés, B.Sc, Mikaël Block, B.Sc, and Réjean Girard, Geo. IOS Services Géoscientifiques Inc. Project no. 625, Town of Saguenay, May 27th, 2008".

10.1.2 Adriana's 2008 Drilling Program

The purpose of the 2008 program was to complete the grid drilling of the designated area of the South Zone over a strike length of 9 km. Core Logix Drilling Solutions Inc. ("Core Logix") based in Sussex Corner New Brunswick was contracted by Adriana to carry out the drilling program using Adriana's Atlas Copco P4 drill rig and drill equipment. All the drilling was done using BQ (3.65 cm diameter) core equipment with two 12-hour shifts per day, 7 days per week. Drilling commenced May 30th and was completed September 23rd. All drill and crew moves were facilitated using the A-Star helicopter. Drilling for the 2008 program totaled 41 drill holes aggregating 5,203 m. This total included one short hole (LO-N-1064) that was drilled on the southern end of the North Zone.

Drill collar coordinates were pre-planned and were to be located as close to their assigned grid locations as practical using GPS. Drill setup protocols included ensuring that each drill hole was setup in the correct location prior to start of drilling and that the target depth was reached prior to drillhole closure. Geologists were to visit the drills at least once per day to liaise with drill operators and check progress. Upon completion, drillhole collars were staked with a marker and labelled with an aluminum tag.

The 2008 program was a continuation of the 2007 program and the field drilling protocol remained largely unchanged. The one major change was to target the entire iron formation sequence to Unit 5 rather than only the upper members. Field operations including drilling, core logging, core splitting, expediting and camp operations were supervised by Marc Léonard (Project Manager), a seasoned geologist with iron exploration and drilling experience. Mr. Léonard reported directly to Mr. Tremblay under an arrangement with Minroc. Reporting directly to the Project Manager, Julien Hélou and Simon Carrouée, geologists, supplied under arrangement with IOS, carried out all core 2008 logging activities.

At the end of the program Gestion Otelnuk prepared the report dated January 31st, 2010 titled: "Report On the 2008 Diamond Drilling For Adriana Resources Inc." [in three volumes], prepared by Marc-A. Léonard, M.Sc. and Gilles A. Tremblay, P.Eng.

There was no 2009 drill program on the Otelnuk Property.

10.1.3 Adriana's 2010 Drilling Program

In 2010, Adriana purchased a second Atlas Copco P4 rig and the drilling was again done by personnel provided by Core Logix. The program was planned and supervised by Gilles Tremblay of Gestion Otelnuk and again was focused on the South Zone. Most of the drillholes were infill holes on a staggered grid pattern covering the central portion of the South Zone grid.

Forty-one (41) holes aggregating 5,874 m were completed.

Core logging and sampling personnel were provided by IOS under contract to Adriana and was completed by M. Block, M. Bolduc and J. Hélou. IOS at program end compiled a program report titled: "Resources Definition Drilling Campaign: Lake Otelnuk Iron Deposit Vol. 1: Report, [with] appendices 1 and 2" dated: May 11th, 2011 presented to Frank Condon and Gilles Tremblay of Adriana Resources by Mikaël Block, P. Geo. and Réjean Girard, P. Geo., Project: 625, Ville de Saguenay.

MRB and Associates Inc. ("MRB") was responsible for final database design and assembly.

10.1.4 Adriana's 2011 Drilling Program

The 2011 diamond drill program was planned and supervised by Gilles Tremblay (P.Eng.) of Gestion Otelnuk, with assistance from Daniel Lytwynec (Project Field Manager) and Frank Condon (P. Eng. and Director of Quebec Operations for Adriana). The program comprised two phases: a Phase I from May 13th to August 15th and a Phase II started August 16th and terminated December 7th. Phase I consisted of 27 BQ-size drill holes, including three piezometer wells were drilled for a pump test and two (2) PQ-size drill holes for the purpose of collecting bulk samples for metallurgical testing and one PQ hole with a dual purpose to test the potential for an artesian well to supply potable water, aggregating a total of 3,664 m. During Phase II, 81 BQ and 2 additional PQ holes aggregating 11,702 m were completed. Total 2011 drilling aggregated 112 holes for 15,366 m.

The Baie Gignard Camp was opened on April 17th to prepare for the 2011 drill campaign. Expediting services were supplied by Air Saguenay (1980) Inc. operating De Havilland Otter aircraft from Schefferville Quebec on skis, and later on floats from their Squaw Lake Base near Schefferville. An A-Star 350 B2 helicopter supplied by Canadian Helicopter Limited was mobilized and based in the Camp prior to the start of drilling. The first drill was placed into operation on May 13th with the second beginning on May 28th. The drill crews are supplied by Core Logix who operated Adriana's owned Atlas Copco P4 rigs. A third diamond drill under contract from Core Logix was placed in service on November 8th, but only completed one PQ hole before drilling was suspended for the winter. Drilling was done in two 12 hour shifts per day, seven days a week. Drilling equipment and supplies are moved by helicopter. The helicopter was also used to transport drill crews, and drill core from the drill to camp where it is logged and sampled. ATVs are also used for backup transport.

Phase 2 of the 2011 drill campaign was essentially delineation drilling designed to test the extension of the Main Zone on strike to the northwest and southeast over an additional strike length of approximately 26 km. A much wider drill hole spacing of 600 m by 1,000 m and 1,200 m by 1,000 m was used than the 600 m by 500 m grid pattern of the 9 km strike length Main Zone. Delineation drilling has now been carried out over a total strike length of 35 km. Infill drilling, if warranted or required will be done in subsequent drilling campaigns.

Four holes were also drilled along the south western edge of the Main Zone to complete the South Zone grid pattern started in 2007, and seven holes were drilled northeast of the Main Zone to test the down dip extension of the iron formation.

The four larger diameter PQ holes were drilled for the purpose of collecting representative samples for metallurgical grinding tests. Three holes were drilled as "twins" to three BQ holes in the Main Zone. The PQ holes were logged by the geologist and shipped un-split to SGS-Lakefield.

Julien Helou – Geologist and Mathieu Vallée – Junior Geological Engineer carried out core logging, sampling and geological mapping (see the Exploration Section of this report).

10.2 Latest Drilling Programs

10.2.1 LOM's 2012 Drilling Program

The 2012 diamond drilling program carried out by Gestion Otelnuk under contract to LOM. The program comprised 157 BQ, 21 NQ and 18 PQ diamond drill holes totaling 22,249 m and was completed between May 3rd and December 13th..

The objectives of the 2012 drill program were to:

- Further expand and upgrade the Lac Otelnuk Mineral Resource,
- Conduct hydrogeology tests and establish hydrogeology monitoring wells in the area of the proposed initial open pit mine,
- Investigate sub-surface soil and bedrock conditions for the proposed tailings facility and;
- Collect large diameter PQ core samples for pilot plant metallurgical testing.

10.2.2 BQ Delineation Drilling Program

The Baie Gignard Camp was opened on March 13th and between March 16th and April 23rd over 500 tonnes of fuel and materials were air lifted from the Schefferville Airport to Baie Gignard by ski equipped Twin Engine Otter aircraft operating under a contract with Air Inuit. Two helicopters provided by Canadian Helicopters Ltd. and drill crews provided by Core Logix were mobilized in early May. Routine expediting services through the season were supplied by Air Saguenay (1980) Inc. and Norpaq Adventures operating De Havilland Otter aircraft on skis, and later on floats from the Squaw Lake Base near Schefferville. As soon as snow conditions permitted, expansion of the Baie Gignard camp to accommodate up to 80 persons commenced.

Delineation diamond drilling with BQ coring equipment began on May 6th and quickly ramped up to four drill rigs operating two 12 hour shifts per day, seven days a week. All four drill rigs and drilling equipment are owned by LOM and operated under contract to Core Logics Inc. The primary objective of the 2012 delineation drilling program was to complete the grid pattern drilling along the 36 km strike length of the Lac Otelnuk deposit at a drillhole spacing of 600 m by 500 m.

In the early stage of the campaign, drilling was limited to in-fill drill sites prepared in late 2011 in anticipation of an early season start. As field conditions improved drilling was focused first on defining the southwest limit of the Main Zone of the deposit between Lines 50S and 220S, and a row of holes to the northeast of the Main Zone (at the 2000 m stations of the grid) to further test the down dip extension of the deposit as it dips under the caprock cover. Delineation drilling then progressed to completing the 500 m by 600 m grid pattern on the North Zone of the deposit from Line 30S north to Line 250N. The North Zone had previously been tested on a much wider grid pattern. Delineation drilling was completed in October. All delineation holes are vertical (-90°) and penetrate the entire iron formation stratigraphy (Unit 2 through 4) terminating in Unit 5, argillite. Eight of the delineation holes encountered problems and had to be re-drilled. During the 2012 campaign, 157 BQ delineation holes were drilled totaling 17,919 m.

Core logging and sampling procedures used for the delineation drilling program were done in accordance with LOM's 2012 Diamond Drilling Protocol and Guideline Manual prepared by F. Condon with input and recommendations provided by WGM.

10.2.3 NQ Hydrogeology Drilling Program

On July 7th one of the drills (Drill C) was converted to NQ drilling equipment for the purpose of drilling 10 vertical NQ size holes (BH-12-1 to BH-12-10 totaling 1,652.4 m) at the perimeter of the proposed initial mine pit to measure the hydraulic properties near the position of the future pit walls through which water flow will take place. The hydrogeology program was designed and supervised by Golder. Packer tests were conducted at 20 m intervals to measure the hydraulic conductivity K-values at various depths. Automated water level logger probes were installed on eight of the wells and ground temperature monitoring instrumentation was installed on the other two wells to investigate possible permafrost conditions. The hydraulic data collected from the field investigation will be combined with other geological and hydrogeological information to produce a groundwater flow model for the assessment of pit water inflow and the extent of groundwater level drawdown. Drill cores collected from the hydrogeology drilling program were routinely split and assayed and are included in the drillhole database.

10.2.4 NQ Geotechnical Drilling Program

Eleven vertical NQ boreholes (BH-12-11 to BH-12-21 totaling 131.77 m) were drilled through overburden into bedrock at the location of the dykes of the proposed Tailings Management Facility ("TMF"). This program was also designed and supervised by Golder. Water monitoring wells for groundwater level monitoring were performed in each borehole and Packer tests for hydraulic conductivity of the shallow bedrock in nine of the boreholes

10.2.5 PQ Sample Drilling Program

Large diameter (85 mm) PQ drilling commenced August 24th with Drill D and a second drill was converted to PQ equipment on November 6th. The purpose of the PQ drilling program was to collect samples for future metallurgical testing. The PQ holes were drilled as twin holes to previously drilled BQ delineation holes for control purposes. Eighteen PQ holes were completed totaling 1,933 m including 1,658.6 m of iron formation. The program produced an estimated 33.2 tonnes of sample material for metallurgical testing. Drill core was logged and shipped in boxes to SGS-Lakefield.

MRB continued with responsibility for the Project's database management and all geomatic services.

10.2.6 Adriana's Drill Hole Collars and Down-Hole Surveying

All of Adriana's drillholes are vertical so they don't require down-hole attitude surveys.

Final collar location surveys since inception of drilling in 2007 have been done by Cadoret using differential GPS. Surveys are all NAD 83 Zone 19. Cadoret have made several visits to the Property to collect data as LOM and Adriana's programs advanced. Cadoret has issued several reports containing its survey results. Cadoret's last visit to the Property to support the 2012 drilling program was in November 2012. Subsequently they issued a report in March 2013 detailing their work. Cadoret states the relative precision for collar locations is 5 cm.

LOM's protocol includes marking all collars with a painted post with metal tag, but it is not clear if Cadoret was able to precisely identify all collars for surveying. Regardless, drillhole locations are of sufficient accuracy to support a Mineral Resource estimate.

Most of the historic collar locations from the 1970s were also surveyed by Cadoret.

10.3 WGM Comment on LOM and Adriana's Drilling Programs

WGM cannot comment on quality of recent drill supervision from first-hand observation since it has made no visits to the Property since 2008. Certainly during WGM's last site visit in 2008 the then current Project Manager, Mr. Marc Léonard, was visiting the drills on a periodic basis to keep-up on drilling issues. For the 2010 campaign WGM is aware that there were some issues between IOS who were contracted to log core and perform the sampling, but not to supervise the drilling and drill supervision was under the auspices of Gilles Tremblay. WGM is not aware of any drilling issues since that time.

The Header table in the drillhole database would still benefit from further upgrading. Core size for individual drillholes and drilling dates remain incomplete.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

WGM has relied for our descriptions of sample preparation and analyses solely on the basis of historic reports, notes and communications with LOM and Adriana personnel and the analytical laboratories themselves. Additional descriptions have been summarized in previous WGM NI 43-101 Technical Reports.

11.1 Sample Procedure and Sample Security

11.1.1 Historic Sampling

MPH managed all of the pre-2007 explorations on the Property for King. Drill core was split and half core was sent to Lakefield Research Ltd., the predecessor of SGS-Lakefield for assay and Satmagan test work. The second half core from the samples and the un-sampled material was retained in the core trays on the Property.

The historic drill core in 2005 was found by Adriana in very good shape and in 2007, 15 samples of archived core were check assayed at SGS-Lakefield. Results of this comparison were excellent. Figure 11.1 shows the comparison between historic Soluble Fe Head ("SFe") assays for these 15 samples and 2007 SGS-Lakefield XRF TFe assays. Figure 11.2 shows %MagFe calculated from 2007 Davis Tube results for the 15 historic samples plotted versus original Satmagan %MagFe results. Figure 11.3 shows the relationship between the historic Satmagan results and DTWR using DTWR from the 2007 test work results at SGS-Lakefield.

Figure 11.1 – %SFe_H (Historic Samples)

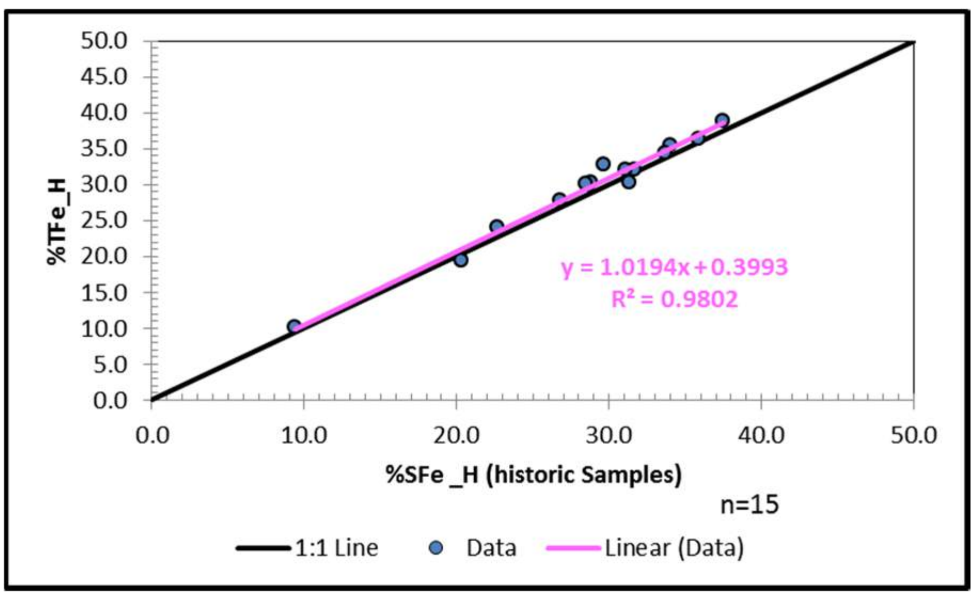


Figure 11.2 – %MagFE_Sat_Historic Sample

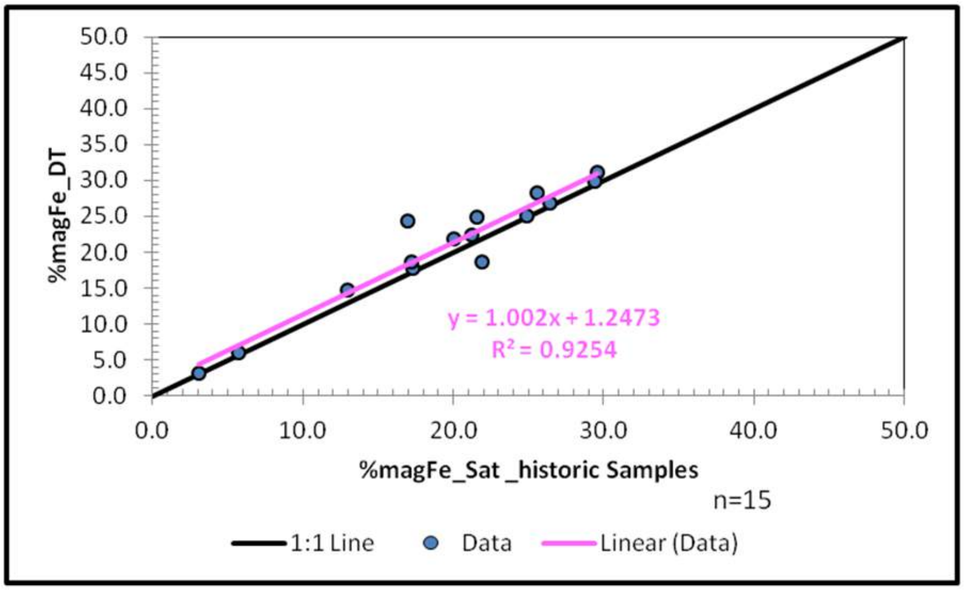
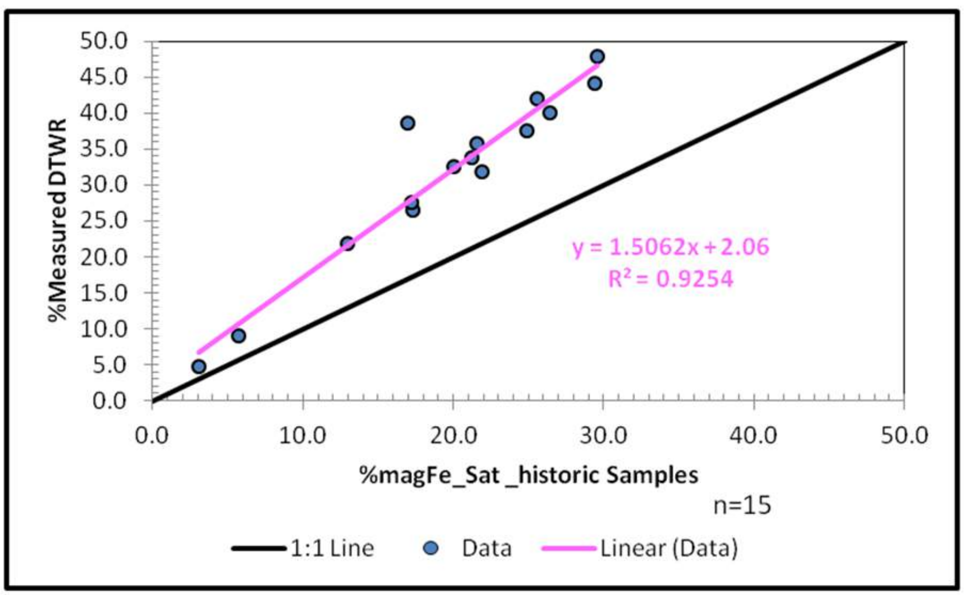


Figure 11.3 – %MagFe_Sat_Historic Samples



WGM understands that when the historic drill core was moved from the original MPH camp site to the Adriana's campsite on the shore of Baie Gignard some of it was dropped and tray labels lost, so now much of it is not trusted.

Quality assessment reports including drill logs for the MPH-King programs including Soluble iron and Satmagan Head assays are available from MRNF assessment files. The core log data and assay results have been incorporated into the project database by MRB.

11.1.2 LOM and Adriana Sampling

a) Core Logging

Adriana to date has operated five field drilling programs on the Property (2007, 2008, 2010, 2011 Phase 1 and Phase 2 and 2012) and WGM understands that procedures have remained much the same. Julien Hélou, Geologist, has logged core since Adriana program inception and provided continuity to the process. A document, Condon (2010), updated for each of Adriana's drill programs provides protocol and guidelines for carryout the field activities.

Drill core is delivered to the campsite by helicopter on a daily basis where it is unpacked and ordered. Core trays were labelled with aluminum tags denoting drillhole identification and box number.

Core logging software has changed through the years but descriptive core logging lithology codes developed by MPH from the King programs have been used in all programs.

Core logging included Rock Quality Index ("RQD"), magnetic susceptibility measurements on 0.25 m to 1 m intervals down the core and core photography.

b) Sampling

Sample intervals are marked on the core by the logging geologists using china markers or lumber crayons and then recorded in 3-part sample books. The entire iron-rich section of the drill core was sampled leaving no gaps. Sample lengths were based on geological criteria and sample lengths have averaged approximately 4 m (2007 - 3.9 m, 2008 - 4.3 m, 2010 - 3.6 m, 2011 - 4.3 m and 2012 - 3.9 m). These sample lengths are similar to what was done by MPH for the King programs. The Adriana protocol included shoulder samples bordering mineralized intervals. Blanks, Standards and Duplicates have been used for some programs, but not all programs. More descriptions of the QA/QC are in Section 11.3 to 11.10 of this report. Sampling statistics for the various Adriana programs are summarized in Table 11.1.

One portion of the 3-part sample tickets are stapled into the core trays at the beginning of each sample interval. Aluminum tags, also recording the sample identification information were stapled into the trays accompanying the paper tags.

Table 11.1 – Summary of LOM and Adriana Samples for Analysis and Test Work 2007 - 2013

Program	Assay Type	Number of Assays
2007	Routine Samples/Routine WR XRF Head Assays plus sent for DT test	515
	Fe determinations on DTCs (XRF WR)	488
	Sulphur determinations on Heads	481
	Sulphur determinations on DTCs	167
	SG by gas comparison pycnometer	0
	Bulk Density determinations by water immersion	0

Program	Assay Type	Number of Assays
2008	Routine Samples/Routine WR XRF Head Assays plus sent for DT test	1,045
	Fe determinations on DTCs (XRF WR)	905
	Sulphur determinations on Heads	0
	Sulphur determinations on DTCs	0
	SG by gas comparison pycnometer	313
	Bulk Density determinations by water immersion	0
2010	Routine Samples/Routine WR XRF Head Assays DT test	1,297
	DT tests	1,294
	Fe determinations on DTCs (XRF WR)	1,129
	Sulphur determinations on Heads	0
	Sulphur determinations on DTCs	0
	SG by gas comparison pycnometer	76
	Bulk Density determinations by water immersion	8
2011	Routine Samples/Routine WR XRF Head	2,718
	DT tests	2,138
	Fe determinations on DTCs (XRF WR)	1,705
	Satmagan on Heads	2,254
	Sulphur determinations on Heads	2,718
	Sulphur determinations on DTCs	0
	SG by gas comparison pycnometer	856
	Bulk Density determinations by water immersion	0
2012	Routine Samples/Routine WR XRF Head	3,967
	DT tests	349
	Fe determinations on DTCs (XRF WR)	309
	Satmagan on Heads	3,959
	Sulphur determinations on Heads	3,631
	Sulphur determinations on DTCs	0
	SG by gas comparison pycnometer	1,291
	Bulk Density determinations by water immersion	79
2013	Check assaying	
	Most Head samples re-assayed for WR-XRF, Satmagan and pycnometer SG	398
	A series of samples (SGS-Lakefield certificate CA02900-JUN13) with only Satmagan checks.	135

Compilation includes shoulder samples to mineralized intervals but does not include any inserted QA/QC materials;
The listing also does not include samples sent to the Secondary lab – MRC;
DT tests don't equal XRF assays on DTC because some DT tests produced no magnetic concentrate or insufficient sample for analysis;
The numbers of samples and assays are not completely accurate because some samples have been re-assayed more than once.

For the 2007 program, core splitting was done using a Longyear-type splitter with an extended handle. This was arduous and quality of the split core samples was less than ideal, especially for, but not limited to, quarter core splits that IOS was doing to produce field Duplicate samples. For the later programs, routine core splitting was done using a hydraulic splitter and this proved much more effective and sample quality was much better. Core splitting to provide quarter core field duplicate samples was done using a diamond saw.

Split core samples were placed into plastic sample bags with the second portion of the 3-part sample tickets and stapled shut. The bags were also labelled with indelible marker. Samples were packed into 5 gallon steel pails, labelled with a sequential pail number and the analysis quotation identification. Samples were sent as batches from the Property by aircraft to Squaw Lake, Schefferville. From there, the samples went by rail and truck to SGS-Lakefield, Lakefield, Ontario.

c) Core Storage

Dedicated core storage buildings were constructed at the camp site in 2008 and all historic and Adriana core is now stored securely on racks in these two buildings.

d) WGM Comments of Phase I Drill Core Logging and Sampling

WGM made two (2) site visits to the Property as described under Section 14.0 to review field program procedures and monitor results. Only one of these visits (September 2007) was made during a period when logging and sampling was in progress. On the basis of our observations, WGM believes that the core handling and core splitting was done to an adequate standard.

11.2 Sample Preparation and Analysis

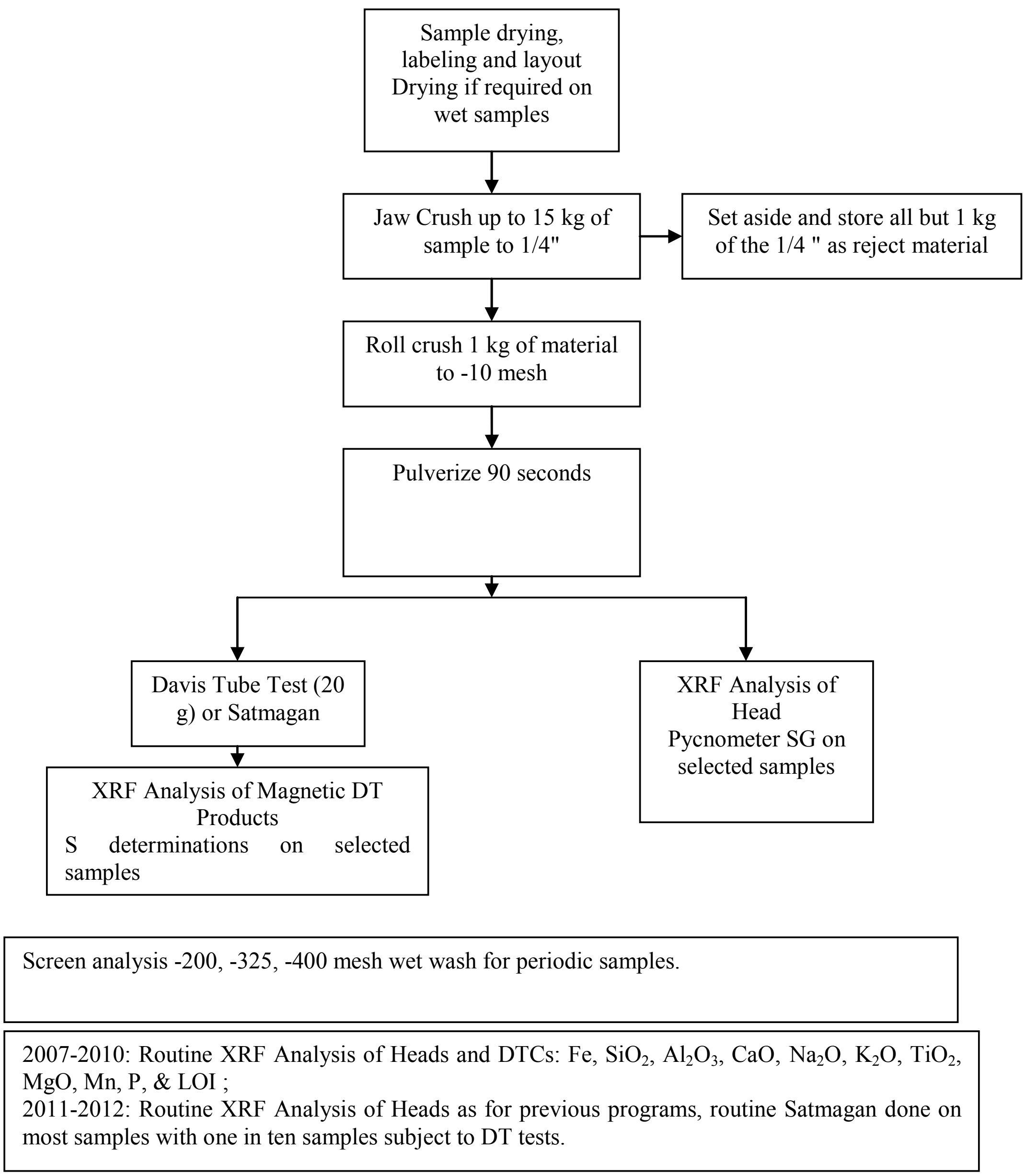
11.2.1 LOM and Adriana 2007 to 2013 In - Laboratory Sample Preparation and Analysis

Drill core samples collected by Adriana from 2007 through 2013 were submitted by Adriana/LOM to SGS Lakefield at its Lakefield, Ontario facility. SGS-Lakefield is an accredited laboratory meeting the requirements of ISO 9001 and ISO 17025. Although WGM has reviewed the assay results and a selection of Certificates generated by SGS-Lakefield and believes they are generally accurate, WGM is relying on the laboratory as an expert in the field of analyses. Adriana's drill core samples were analysed mainly by Borate fusion X-Ray Florescence ("XRF") methods.

Adriana's analysis protocol from inception in 2007 through 2010 also included Davis Tube tests. For the 2011 program Davis Tube tests were partially discontinued and replaced by Satmagan determinations. For LOM's 2012 program Satmagan determinations remained the primary method for estimating magnetic components of mineralization with periodic DT tests to check/confirm Satmagan results.

For the period 2007 to 2010 samples were jaw crushed to ¼ inch and then 1 kg of this material was roll crushed to -10 mesh and then pulverized for 90 seconds in a ring and puck pulverizer (Figure 11.4). The 90 second grind time was determined by initial test work to optimize iron grade, silica levels and iron recovery. Screen analysis of products was done on a periodic basis. The sample was then split with one portion going for Head analysis by Borate Fusion XRF and 20 g as feed to Davis Tube test. The Davis Tube magnetic concentrates ("DTC") were analysed by XRF for major elements. The Davis Tube tails were discarded.

Figure 11.4 – Sample Processing Flow Sheet at SGS-Lakefield 2007 – 2012



Sulphur determinations were done on some samples and phosphorous was also determined on some samples by Inductively Coupled Plasma ("ICP"). Specific gravity on selected pulps was completed using a gas comparison pycnometer. WGM understands that these samples were selected on the basis of trying to be representative of all rock types.

For the 2011 and 2012 programs Satmagan determinations of magnetic Fe were completed on many, but not all samples. Many were still subject to only Davis Tube tests. A selection of samples had both Davis Tube tests and Satmagan determinations completed. Comparative results where both determinations were completed are previously shown on Figure 7.9 and Figure 7.10.

11.3 LOM and Adriana 2007-2013 Quality Assurance and Quality Control

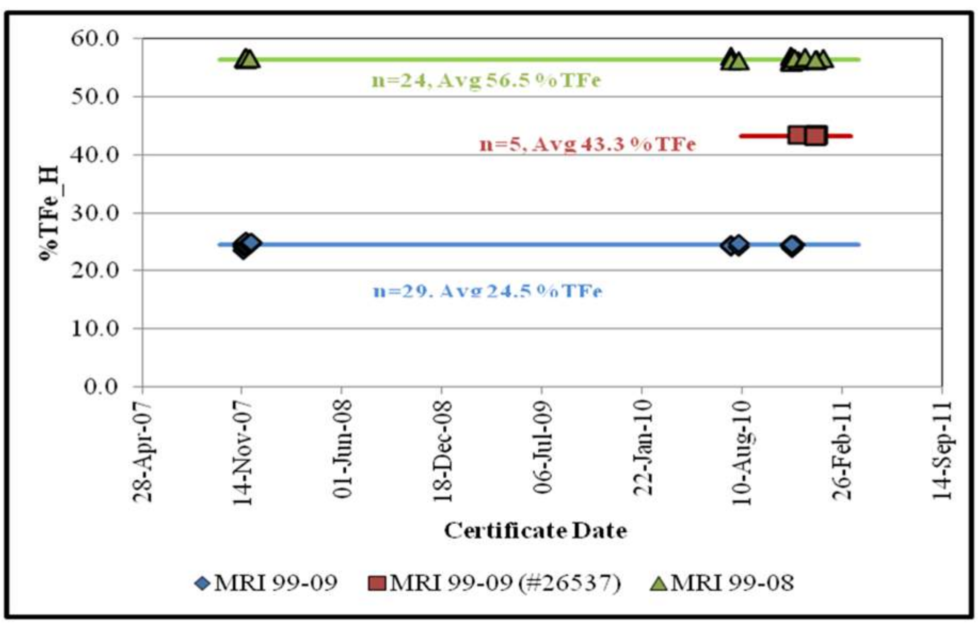
LOM and Adriana's QA/QC protocol includes both in-Field and in-Laboratory components with the In-lab components being SGS-Lakefield's internal QA/QC procedures with minor revision. The protocol has changed slightly through the various drilling campaigns. During the summer and fall of 2013 it conducted a review of 2007 through 2012 assays and performed Check assaying on a number of samples from previous programs. Most of this re-analysis was conducted on new pulps made from original reject in storage at SGS-Lakefield. Some of the selected sample rejects could not be located. Some samples from re-split and sub-sampled drill core were also check assayed. The new assays were used to improve the quality of the sample assay database and some of the original assays judged as erroneous were replaced.

11.4 Analysis Standards

IOS during their management of the drilling programs during the 2007 and 2010 campaigns inserted Reference Standards (FSTDs) into the sample stream in the field. They used three different Standards MRI 99-08, MRI 99-09 and MRI 99-09 (#26537). These Standards were prepared in 1999 by COREM (Consortium de recherches minérales), a joint commercial / government-run laboratory and metallurgical facility in Quebec. MRI-00-09 represents homogenized material from an iron-titanium-vanadium deposit, while MRI-99-08 is the magnetite concentrate from the same ore.

Figure 11.5 presents results for all samples of these three Standards submitted to SGS-Lakefield with routine samples and analyzed as a part of Adriana's 2007 and 2010 drilling campaigns. All samples returned consistent values indicating SGS-Lakefield was producing precise analytical results.

Figure 11.5 – % TFe Results for Three (3) Field-Inserted Reference Standards 2007 and 2010 Drilling Programs

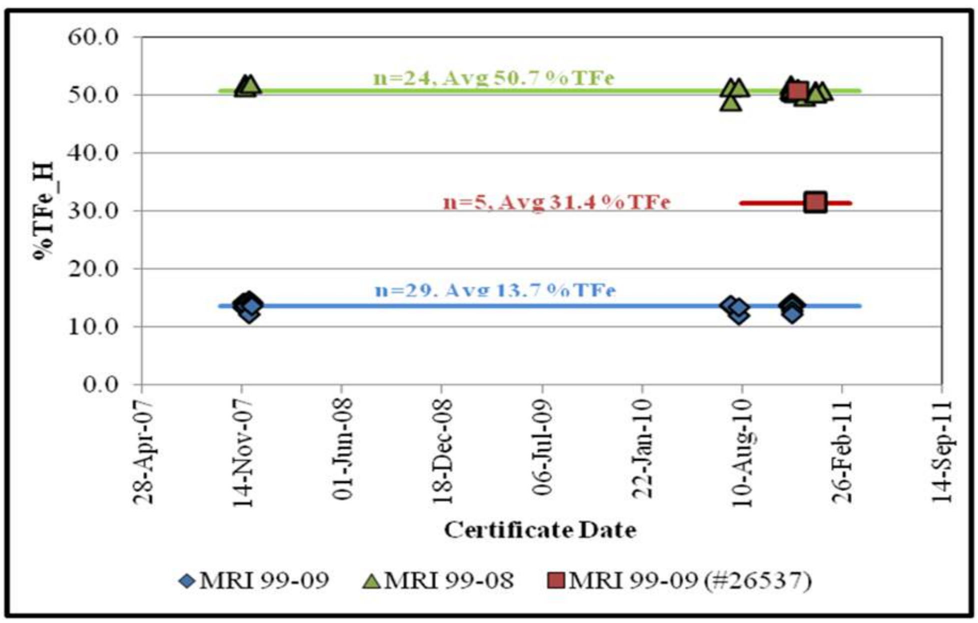


These same Standards also had Davis Tube tests completed. Figure 11.6 presents results for %MagFe calculated from the DT tests. Most values returned are appropriate for the samples and indicate precise results. There is however a couple of sample mix-up or other errors of some type indicated. For two of the MRI-08 samples no MagFe can be calculated from DT tests. For one of these samples (62510317) there is no DTC weight even though there is a DT feed weight. For the other sample (004126) there is no XRF analysis of the DTC. And on Figure 11.6 it is clear that one sample designated as MRI 99-09 (#26537) has a MagFe value consistent with MRI 99-08. This result is likely the DT results for either 62510317 or 004126 and the magnetic concentrates were mixed-up.

There are also a few other irregularities indicated with a few samples having TFe_DTC values lower than expected. These errors appear to probably be lab errors but were never followed-up with check assays.

No field-inserted Standards were used for the 2012 program. LOM substituted increased Secondary Lab check assaying described in Secondary Lab Check Analysis.

Figure 11.6 – % MagFe Results for Three (3) Field-Inserted Reference Standards 2007 and 2010 Drilling Programs



11.5 Analysis Blanks

Blanks, (FBLKs) or non-mineralized Bracket or Shoulder Samples were inserted into the sample stream in the field during the 2007 and 2010 programs. In 2007 these Blanks consisted of blocks of pure quartz that were inserted as the first sample in each hole. In 2010 drill core from the Ruth Formation, Unit 5 was used. These samples were again inserted as the first sample in each drillhole. No Blanks were inserted into the sample stream for the 2008 or 2011 programs. For these programs reliance was on sampling unmineralized shoulder samples to mineralized intervals. These shoulder samples consisted of several lithologies: Menihek shale, Unit 1, or Unit 5 – Ruth Fm, and in one case a sample of dolostone. These lithologies are known to contain only minor mineralization and in particular, little significant magnetite. Shoulder or bracket sampling was also maintained in 2007, 2011 and 2012 programs. For the 2012 program the Blanks consisted of Menihek shale which was inserted into the sample steam at a frequency of one instance per 20 Routine samples.

Figure 11.7 shows TFe Head assay results for the field-inserted quartz block blanks used in 2007. All samples except one returned low values. The one anomalous value probably indicates a sample mix-up either in the field or in the lab. TFe for these samples (excluding the one anomalous value) averaged about 0.6 %TFe. IOS argued these slightly elevated values were due to carry over contamination from previous samples or metallic iron derived from milling equipment during sample preparation. WGM does not disagree. DTWR for all of these samples except for one were less than 0.5 %. For many of these samples XRF WR analysis of the magnetic concentrates could not be completed because the magnetic concentrates were too small, indicating minimum to nil magnetite in the materials. One FBLK of pure quartz (62510316) reported a highly anomalous DT result. This sample was selected as part of the 2013 Check assay program for reanalysis. The new Satmagan result showed a very low value proving the original 2007 value was in error.

Figure 11.7 – % TFe Results for 2007 Field-Inserted Blanks

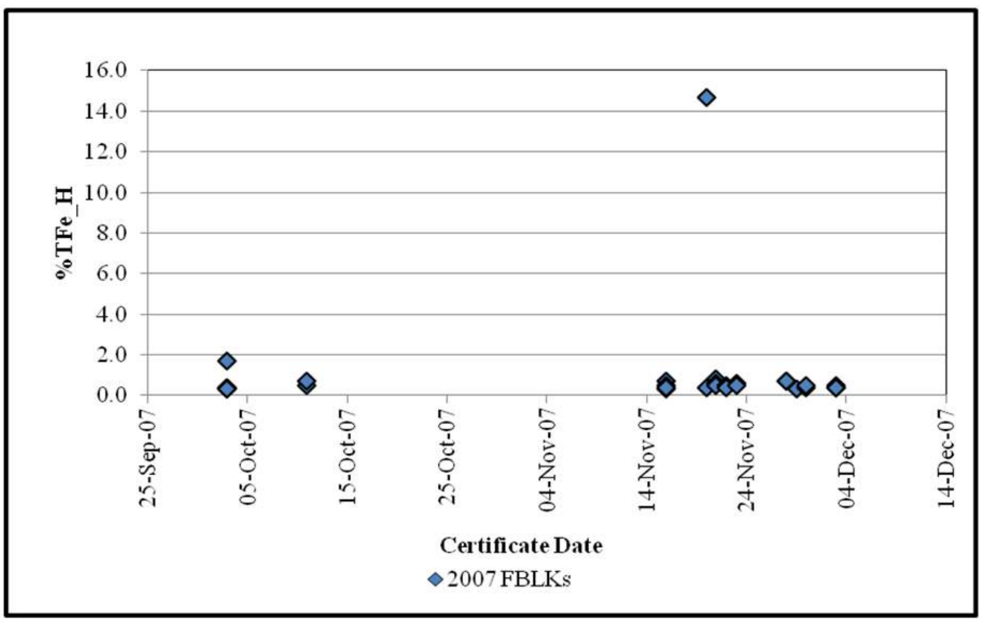


Figure 11.8 shows the results for the 2010 field-inserted Blanks. The natural Blanks do contain some iron, averaging approximately 9.3 % TFe. Results are reasonably tightly clustered suggesting no sample mix-ups in the field or in the lab. DTWR for these 2010 samples averaged 0.06 % and there were no outliers. These results indicate these samples contained minimal magnetite as should be the case because the materials used were not oxide facies iron formation.

Figure 11.8 – % TFe Results for 2010 Field-Inserted Blanks

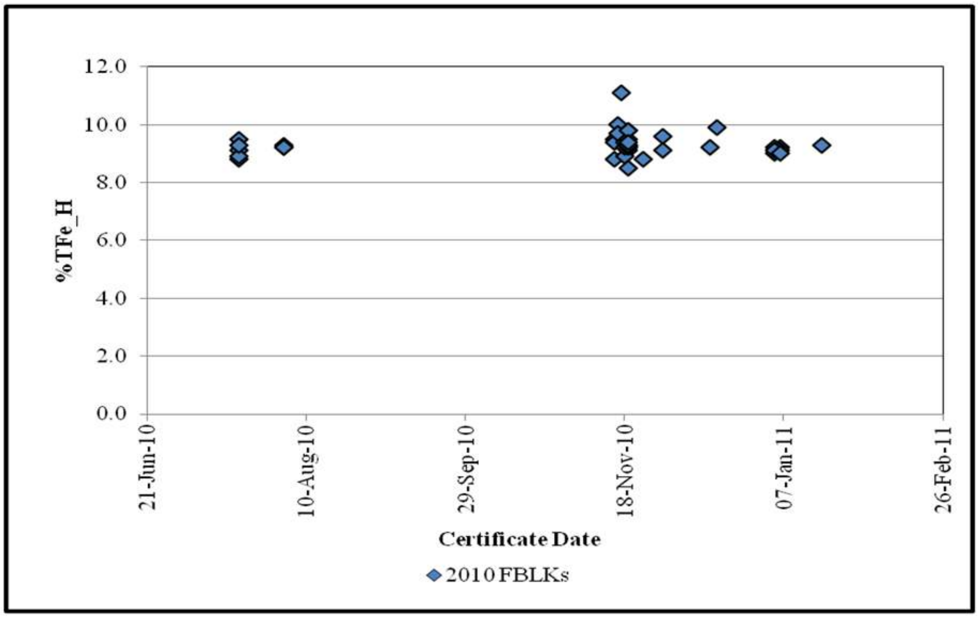


Figure 11.9 shows results for %MagFe in the shoulder or bracket samples to mineralized intervals collected as part of all programs from 2007 through 2011. The values of zero MagFe are for samples that had Davis Tube tests completed that resulted in insufficient concentrate to assay. The samples that show very minor MagFe are samples where Satmagan determinations, rather than DT tests were completed. Satmagan is more sensitive than Davis Tube at the low range of magnetite content. These results for inserted Blanks and non-mineralized bracket samples all indicate minimal sample mix-ups and all samples except one have returned values expected from the materials tested.

Figure 11.9 – % MagFe Results for Bracket or Shoulder Samples 2007 - 2011

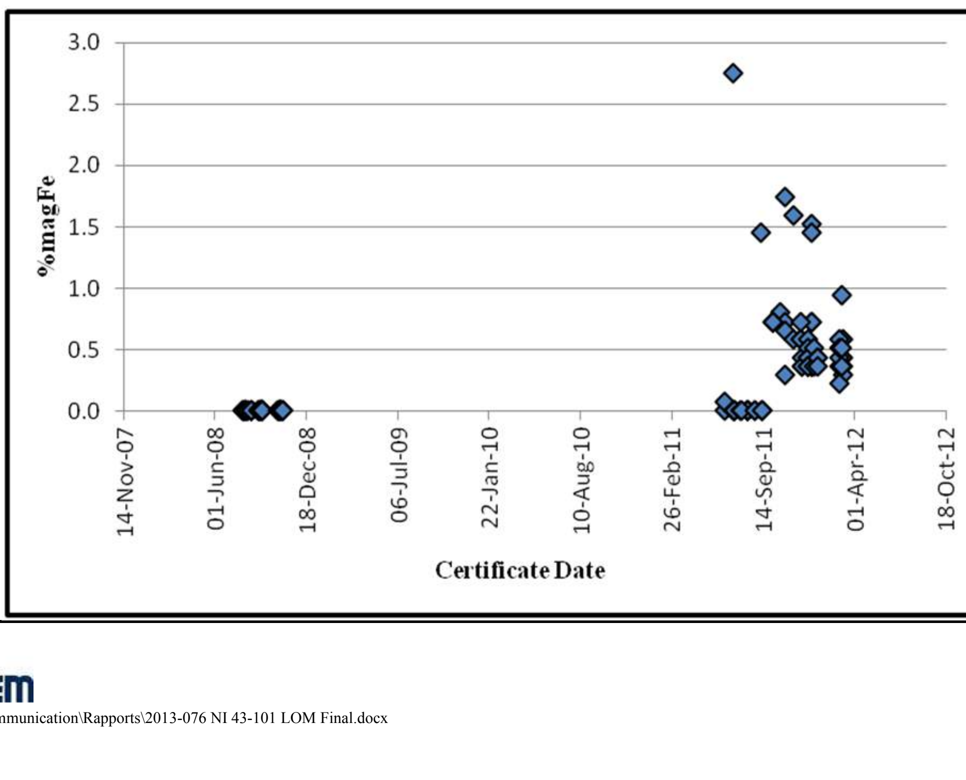


Figure 11.10 and Figure 11.11 show results for Field-inserted Blanks for the 2012 drilling program in terms of TFe and MagFe from Satmagan.

Figure 11.10 – % TFe Results for 2012 Field-Inserted Blanks

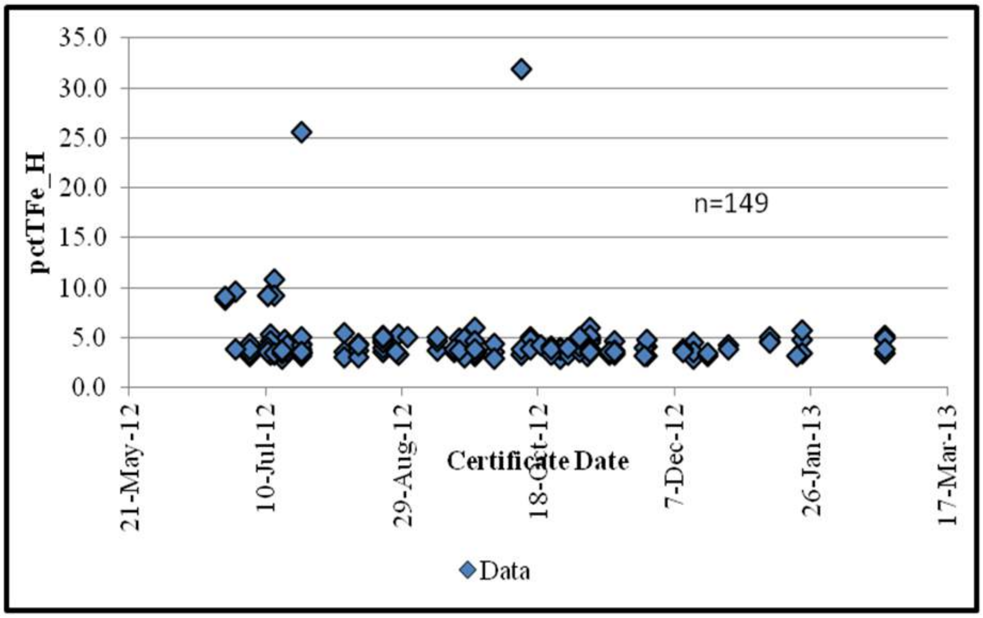
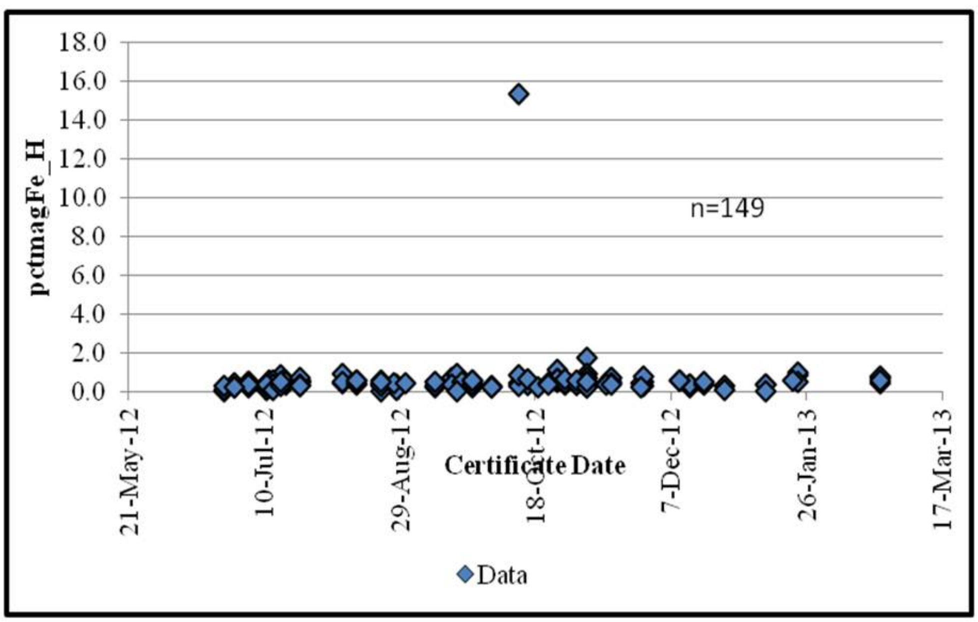


Figure 11.11 – % MagFe Satmagan Results for 2010 Field-Inserted Blanks



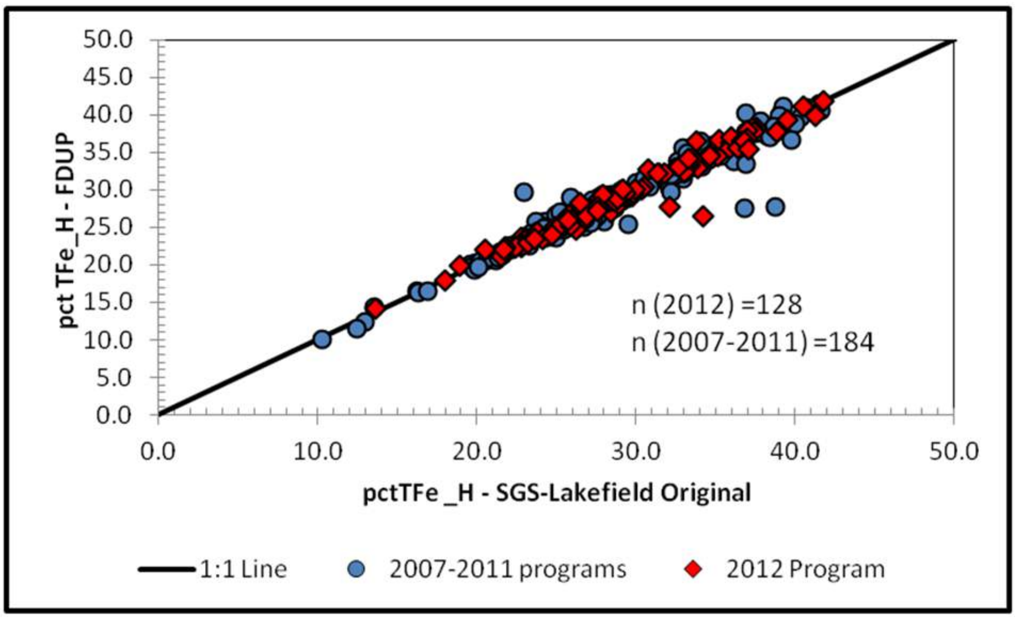
It can be seen that Blanks performed generally as expected but for rare cases anomalous results were reported. WGM understands no anomalous cases were followed up until the 2013 Check assay program.

11.6 Core Duplicates

Field Duplicates (FDUPs) have been collected through all of Adriana's drilling campaigns 2007 through 2012. The Duplicates have consisted of ¼ split core. In 2007 these ¼ core samples were cut using a manual splitter that produced poor quality sample. Thereafter the Duplicate samples were sawn thereby generating improved quality samples. The Duplicates are inserted at a frequency of one per 30 routine samples and are numbered so the sample numbers assigned do not follow immediately after the samples designated as the original.

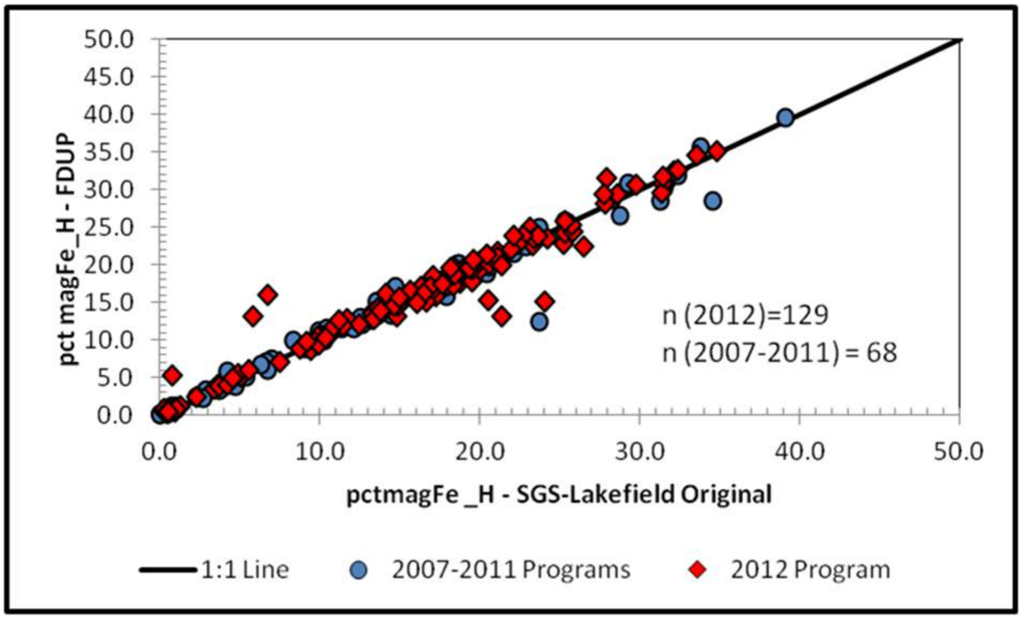
There were 184 core Duplicate samples from 2007 through 2011 and 128 pairs for the 2012 program with XRF analysis completed on Heads. Figure 11.12 shows results for these sample pairs in terms of TFe. There is good correlation between the sample assays and no bias evident indicating generally precise data. There are a few cases samples where original assays reported are significantly higher than the Duplicate which may indicate sample mix-up or contamination error.

Figure 11.12 – % TFe Results for 2007 through 2012 Field Duplicate Sample Pairs



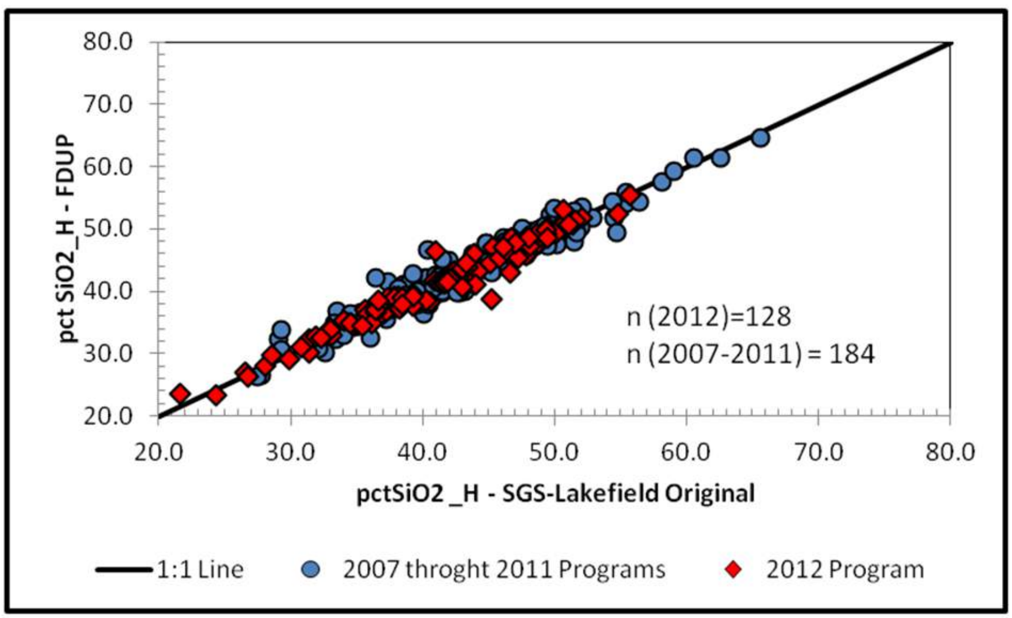
For MagFe from Satmagan there were 68 FDUP sample pairs for the programs 2007 through 2011 and for the 2012 program 129 FDUP sample pairs. Figure 11.13 shows these results for MagFe determined by Satmagan on field-inserted Duplicates. Again results are good with generally excellent correlation between sample pairs. For a few samples sample mix-ups are a possibility. Three FDUPs that were re-analysed during the 2013 Check assay program because of excessive difference in DT results between Duplicate and original had their original DT results replaced.

Figure 11.13 – % MagFe from Satmagan for 2007 through 2012 Field Duplicate Sample Pairs



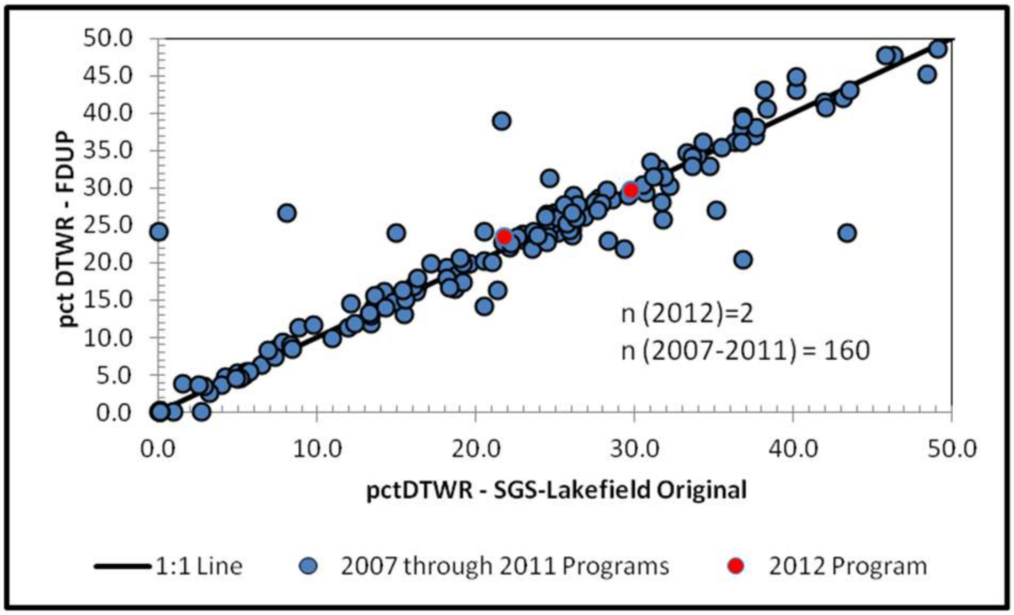
Results for silica are shown on Figure 11.14. Results are again good and silica.

Figure 11.14 – % SiO_2 from Satmagan for 2007 through 2012 Field Duplicate Sample Pairs



Results for 160 samples where Davis Tube testes were completed on field-inserted Duplicates are shown on Figure 11.15. There are only two Davis Tube Field Duplicates for the 2012 program. Results for 160 sample pairs are generally well correlated and unbiased. There are however a number of samples where results are not closely equivalent.

Figure 11.15 – % DTWR results for field–inserted Duplicates

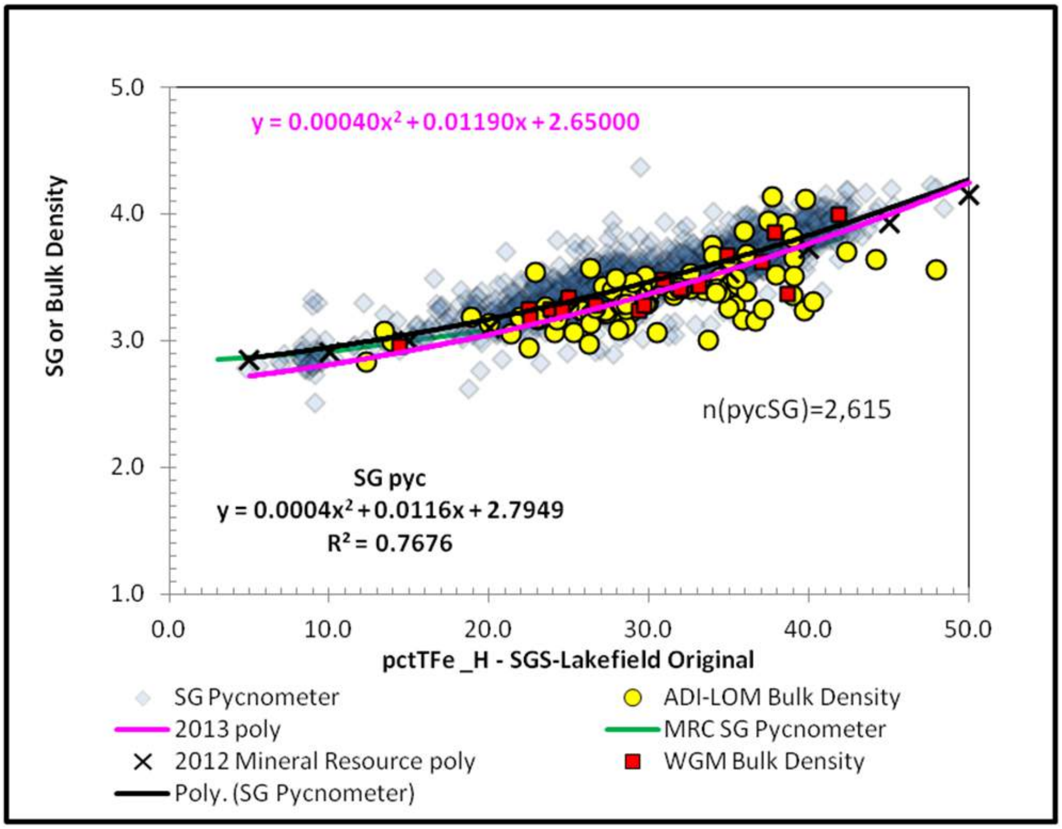


11.7 Specific Gravity and/or Density of Mineralization

As part of the sampling and assaying protocol (see Section 11.2) Adriana designated periodic samples for determinations of specific gravity. These determinations were completed using the gas comparison pycnometer method on prepared pulverized material at SGS. A total of 2,615 drill core samples from 2007 through 2013 had SG determinations completed. Adriana also requested bulk density determinations on a total of 8 samples. These determinations also done at SGS-Lakefield were done on entire ½ split core by weighing in air and weighing in water.

Figure 11.16 shows SG/Bulk density results for all of the 2,615 pycnometer and 87 ADI/LOM bulk density determinations graphed versus Head TFe. Also are shown are results for 17 samples collected by WGM from drill core and submitted to SGS-Lakefield for bulk density determinations, and several best fit trend lines including one (green) based on Check assay work completed at MRC in 2012 (see Section 11.9).

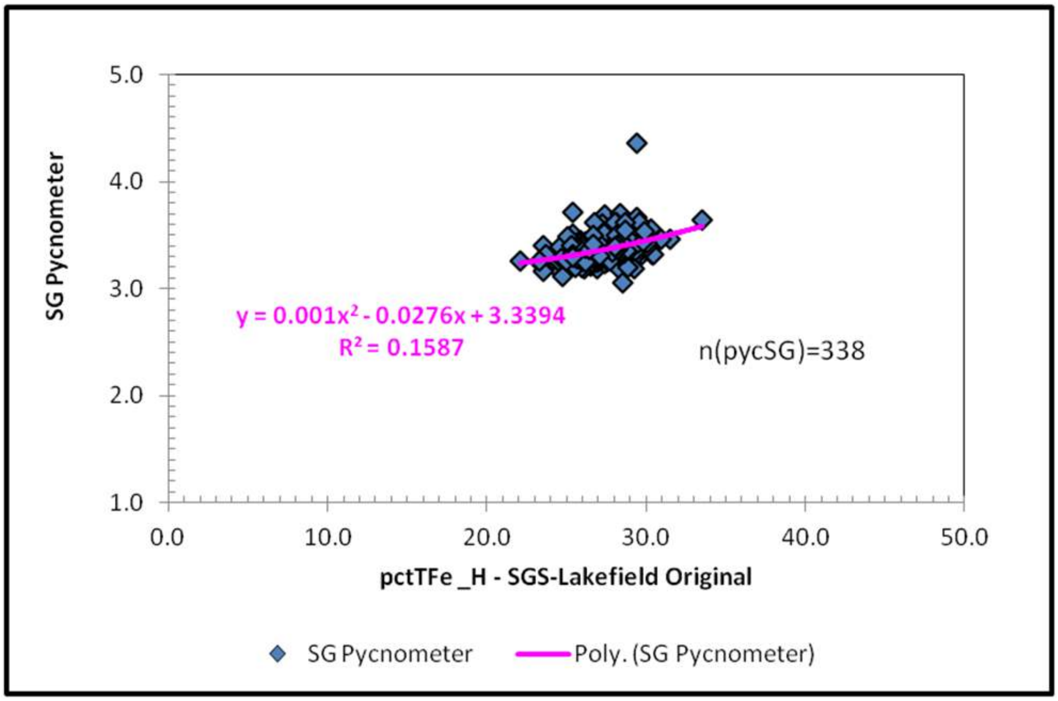
Figure 11.16 – Pycnometer SG or Bulk Density vs. % TFe for all samples



The black trend line which represents a trend line fit to the pycnometer SG results is closely coincident to the MRC (Section 11.9 of this report) trend line shown in green. The black trend line is similar but slightly different than the one defined in the previous WGM report. It also reasonably fits the various bulk density results but appears a little high with respect to the bulk density results. WGM has opted to revise slightly the SG/Density function used for the new Mineral Resource (SG/Density = 0.0004 $\%TFe\_H^2$ + 0.0119 %TFe_H + 2.65 to better fit the now more plentiful bulk density results which for unknown reasons are slightly lower than the pycnometer SGs.

WGM also reviewed the pycnometer SG results by sub-unit. Results for sub-unit 3b are shown as Figure 11.17. Results on the sub-unit basis are generally consistent with "all sample/all sub-unit patterns", however, there are a few outliers defined from departure from the best fit trend line. WGM previously recommended that some follow-up work should be undertaken to check results for some selected samples to determine if non-consistent results are the result of mineralogical differences or error and this work was included as a component of the Check assay program of summer 2013. This check assay work cleaned up a number of the outliers but a few still persist.

Figure 11.17 – Pycnometer SG vs. % TFe for Sub-Unit 3b Samples



11.8 In-Primary Lab (SGS-Lakefield) QA/QC

SGS-Lakefield inserts QA/QC materials and monitors results as part of their own internal QA/QC program. As part of every sample batch SGS-Lakefield prepares and assays Preparation Duplicates which it calls Replicates, Preparation Blanks, Analytical Duplicates, and it inserts Analytical Blanks and Certified Reference Standards into the analytical stream for assaying along with the samples submitted from the field. In 2007 Preparation Duplicates were not part of SGS-Lakefield's standard procedures so Preparation Duplicates were explicitly requested to be added to the analytical protocol. These samples have a routine project sample identifier but terminate in "B". Later SGS-Lakefield added its own Preparation Duplicates it calls replicates and the B samples were discontinued.

Figure 11.18 and Figure 11.19 show results for all Analytical Duplicates for TFe and MagFe from Satmagan 2007 through 2012 programs.

Figure 11.18 – TFe for Analytical Duplicates at SGS-Lakefield, 2007 – 2012 Programs

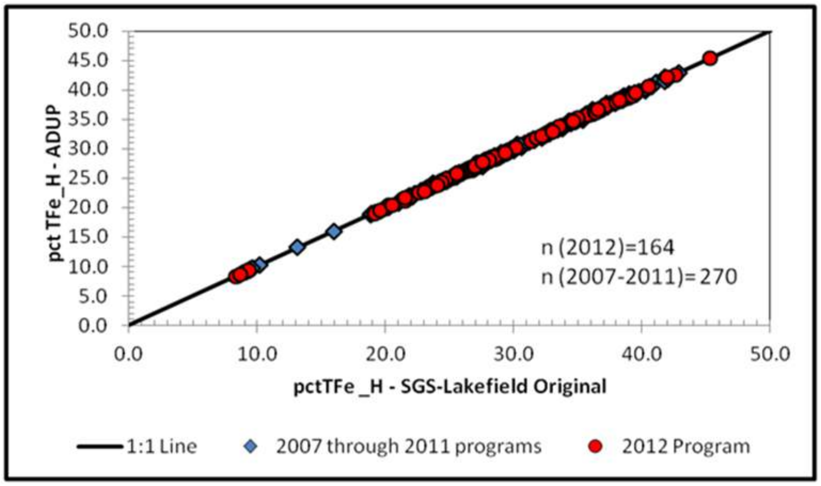
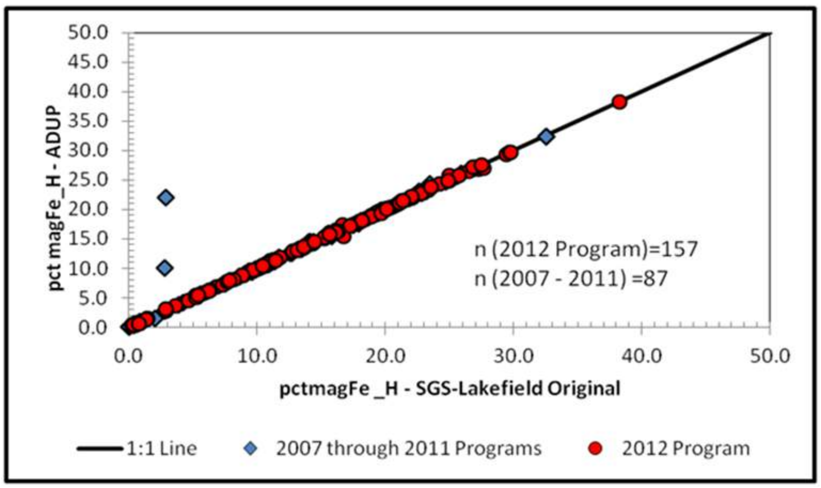


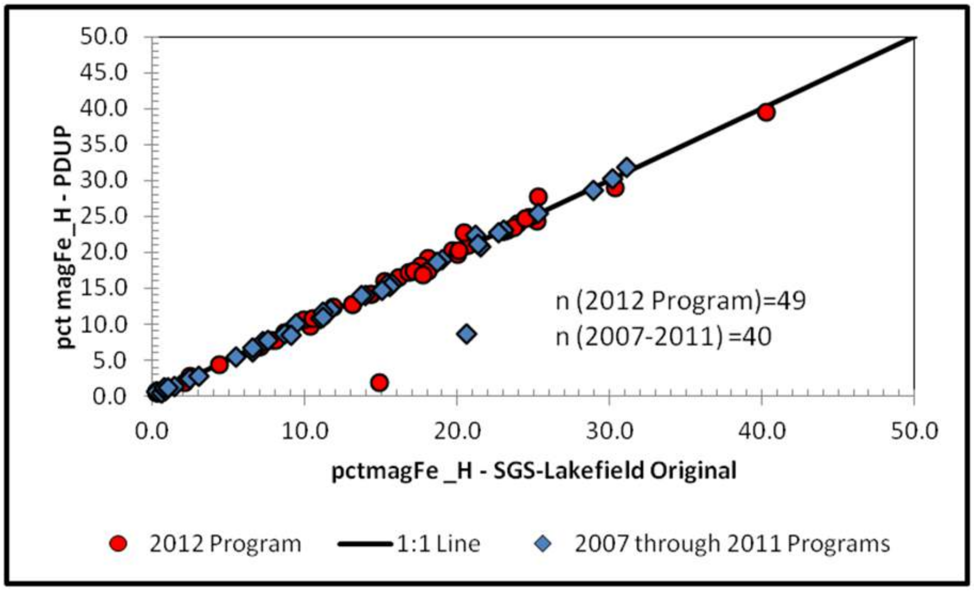
Figure 11.19 – TFe for Analytical Duplicates at SGS-Lakefield, 2007 – 2012 Programs



Results for TFe and Satmagan Analytical Duplicates are excellent except for two (2) samples.

Figure 11.20 shows MagFe from Satmagan results for Preparation Duplicates or Replicates at SGS-Lakefield.

Figure 11.20 – % MagFe for Preparation Duplicates at SGS-Lakefield, 2007 – 2012 Programs



Similarly, Preparation Duplicates generally are highly correlated with their originals. Results plotted here include only samples where the "original" of the pair was a Routine sample.

SGS-Lakefield completes numerous assays of Certified Reference Standards from various suppliers and internal Blanks as a part of its protocol. Different QA/QC materials are assayed depending on analytical protocol and the Adriana and LOM programs have generally comprised several analytical protocols including XRF-WR, Satmagan and SG pycnometer. Consequently there are analytical results for numerous Standards and Blanks.

One issue with these results is that all QA/QC results are not necessarily reported consistently on all Certificates of Analysis or entered consistently into the Project database by MRB. In particular, WGM notes SGS-Lakefield's own Satmagan Standards are incomplete in the database.

Table 11.2 summarizes assay results for SGS-Lakefield inserted Standards used since 2007 in terms of Head determinations of TFe.

Table 11.2 – Summary of Results for TFe in SGS-Lakefield Standards, 2007 through 2012 Programs

Standard ID	Standard or Certified Value TFe (%)	Count of Samples	Avg of pctTFe_H	Min of pctTlFe_H	Max of pctTFe_H
607-1	30.89	83	30.80	30.36	31.19
676-1	39.76	13	39.60	39.20	39.90
680-1	59.98	19	59.74	59.30	60.30
681-1	33.21	37	33.13	32.70	33.40
805-1	14.87	7	68.27	67.80	68.90
879-1	18.97	2	18.90	18.70	19.10
BCS-313/1	0.00839	1	0.01	0.01	0.01
GBM304-15		11	18.87	18.70	19.10
GBM904-15		4	14.33	14.20	14.40
GIOP-31	37.4	5	37.50	37.30	37.70
GIOP-32	30.2	1	30.30	30.30	30.30
GIOP-39	56.6	25	56.73	56.30	57.07
IPT 51	0.83	1	0.82	0.82	0.82
IPT 72	0.06	4	0.06	0.05	0.07
Lithium Blank XRF		282	0.00	0.00	0.01
Lkfd-Sample Prep BLK		107	2.50	0.12	5.99
NCS DC14004a		6	65.45	65.20	65.60
SARM-11	66.1	16	66.34	65.70	66.80
SARM-12	66.6	126	66.64	65.90	67.50
SARM-4	6.27	1	6.27	6.27	6.27
SARM-42	3.273	3	3.36	3.35	3.36
SARM-5	8.84	5	8.97	8.89	9.02
SARM-6	11.89	2	11.80	11.70	11.90
SCH-1	60.73	162	60.79	60.10	61.60
SiO_2 BLK		4	0.01	0.01	0.01
SY4	4.34	21	4.36	4.31	4.44
TILL4	3.97	9	4.02	3.98	4.07
Count = 27		**957**			

The table shows that approximately 27 different Standards and or Blanks were used by SGS-Lakefield for monitoring Head analysis for XRF-WR for the 2007 through 2012 programs. Results shown here, based on averages, minimums and maximum assays are all excellent; there are few or no outliers. Note the "Lkfd Sample Preparation Blk" is two different materials but grouped here as one so assay values are less indicative of performance.

Similarly Table 11.3 summarizes results for Standards analyzed as part of Satmagan work.

Table 11.3 – Summary of Results for MagFe in SGS-Lakefield Satmagan Standards, 2007 through 2012 Programs

Standard ID	Standard or Certified Value MagFe (%)	Count of Samples	Avg of pct MagFe_H	Min of pct maglFe_H	Max of Final MagFe_Sat
Lithium Blank XRF		1	0.72	0.72	0.72
Lkfd-Sample Prep BLK		96	0.42	0.01	0.87
Sat-001	0.7	77	0.80	0.65	1.16
Sat-005	3.6	83	3.58	3.33	3.91
Sat-010	7.2	31	7.36	7.16	7.60
Sat-025	18.1	78	18.12	17.50	19.69
Sat-050	36.2	62	36.32	34.25	38.81
Count = 7		**428**			

Again results are very good but as noted, all results may not be represented.

11.9 Secondary Laboratory Check Analysis

For 2007 and 2008 programs, WGM completed some Secondary Check assaying on behalf of Adriana. This work is described in this report, Section 12.3. No Secondary Check assaying was completed for the 2010 and 2011 drilling programs. For LOM's 2012 drilling program a Secondary Check assaying program was carried out at MRC.

LOM's geologists selected samples of SGS-Lakefield's assay sample reject material. LOM calls these samples "Umpire" samples. According to LOM's assessment report for 2012 there were 186 of these samples selected. SGS-Lakefield rifled off 100 g from each of the 2 mm rejects and sent the material to LOM. LOM forwarded the samples to MRC. MRC reports they received 178 samples including some duplicates. Assays are only available for 167. The Project database lists 172 of these "Umpire" samples but only 167 of the 172 have assays. The discrepancy is probably due to database input error or difference between samples listed for submission versus samples actually found and sent. MRC pulverized the samples to 100 % passing 325 mesh. MRC uses a multi-stage, mechanical mortar and pestle grinding method with dry screening between stages to reach the point where 100 % of the sample passes the prescribed screen.

Head assays for Fe and MagFe by Satmagan were completed on all of the samples. Specific Gravity measurements by Air Pycnometer were also completed on air dried as-received material. Davis Tube tests were completed using 20-g feeds and magnetic concentrates were analyzed for Fe and SiO_2.

For determining total Fe, MRC uses a non-mercury titration method using titanium chloride and titrating with potassium dichromate, following sample digestion using stannous chloride, HCL and HF. MRC determines silica by weighing sample residues before and after selective digestion and two stages of fusion in platinum crucibles.

Figure 11.21 shows MRC results for Fe on Heads versus original SGS-Lakefield assays for 154 sample pairs. Although MRC analysed 167 samples some of these samples are not

Routine samples. The remainder were LOM in-field QA/QC samples and these samples are not plotted.

Figure 11.21 – MRC Head Fe vs. SGS-Lakefield TFe Original

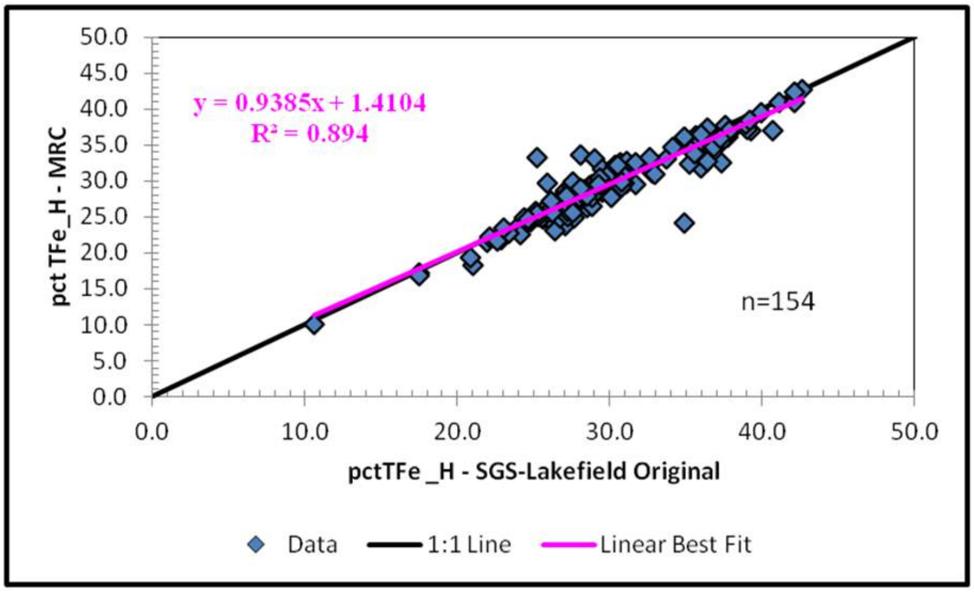


Figure 11.22 shows results MagFe by Satmagan between the two labs.

Figure 11.22 – MRC Head MagFe from Satmagan vs. SGS-Lakefield Originals

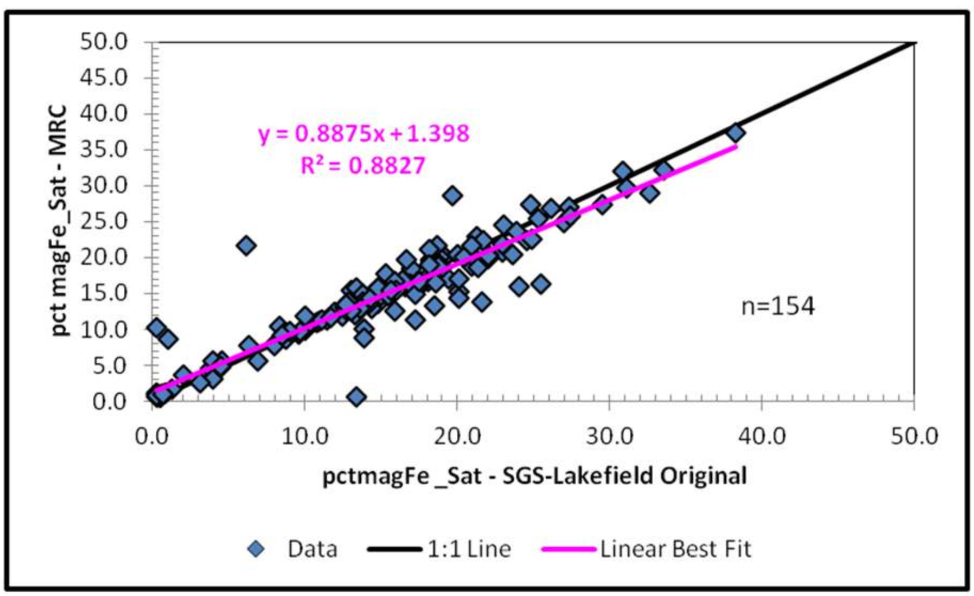
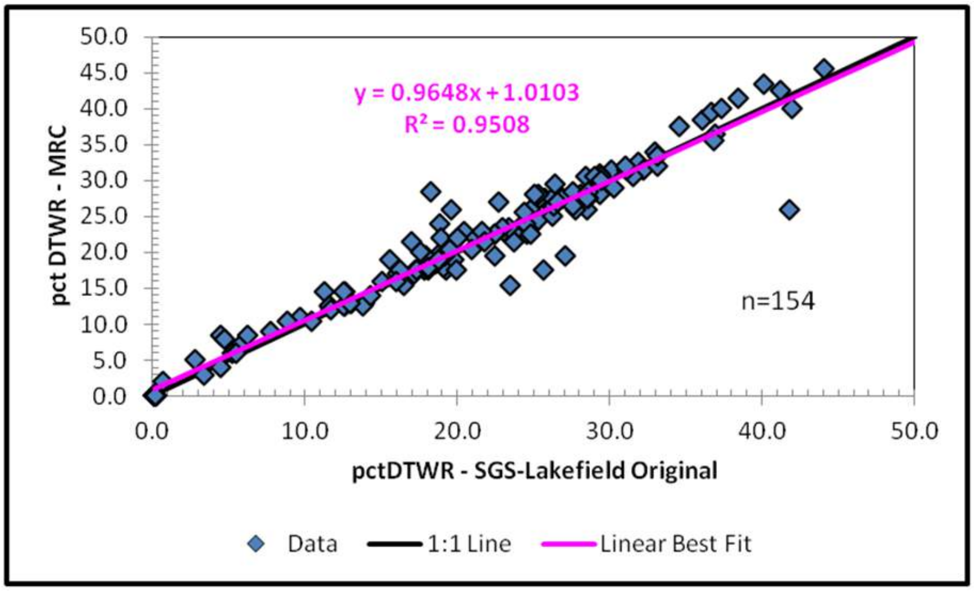


Figure 11.23 shows comparative results for DTWR at SGS-Lakefield and MRC.

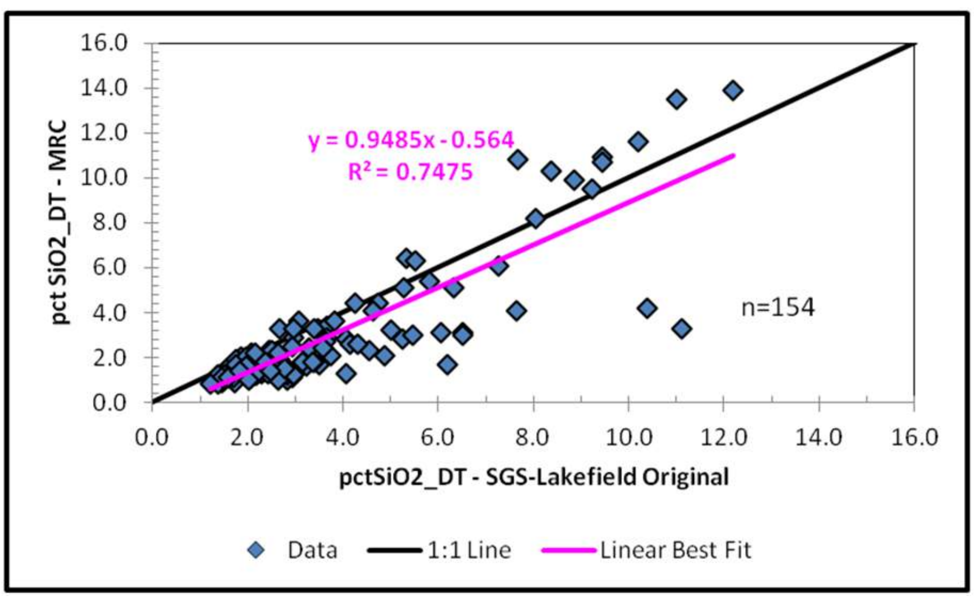
Figure 11.23 – MRC DTWR vs. SGS-Lakefield Originals



SGS-Lakefield and MRC results are generally comparable. MRC's Davis Tube concentrates are a little cleaner (lower silica) and higher Fe grade than those produced by SGS-Lakefield.

Figure 11.24 shows results for silica in Davis Tube concentrates.

Figure 11.24 – MRC SiO_2_DTC vs. SGS-Lakefield Originals



MRC also determined SG on each sample using a method similar to SGS-Lakefield. A comparison of SG results for 37 samples is shown in Figure 11.25. Figure 11.26 is a plot of MRC SG versus MRC Head Fe.

Figure 11.25 – MRC SG Pycnometer vs. SGS-Lakefield

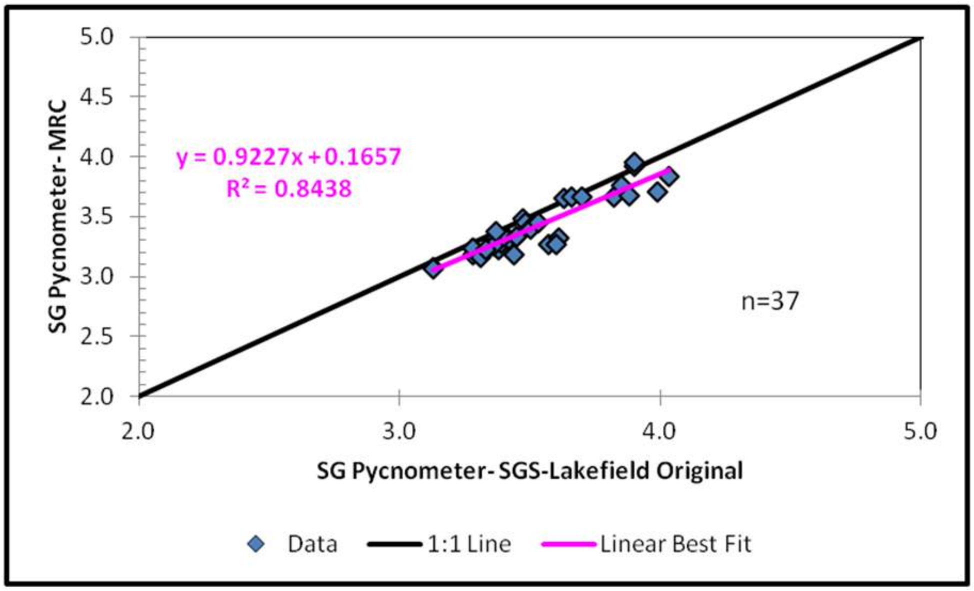
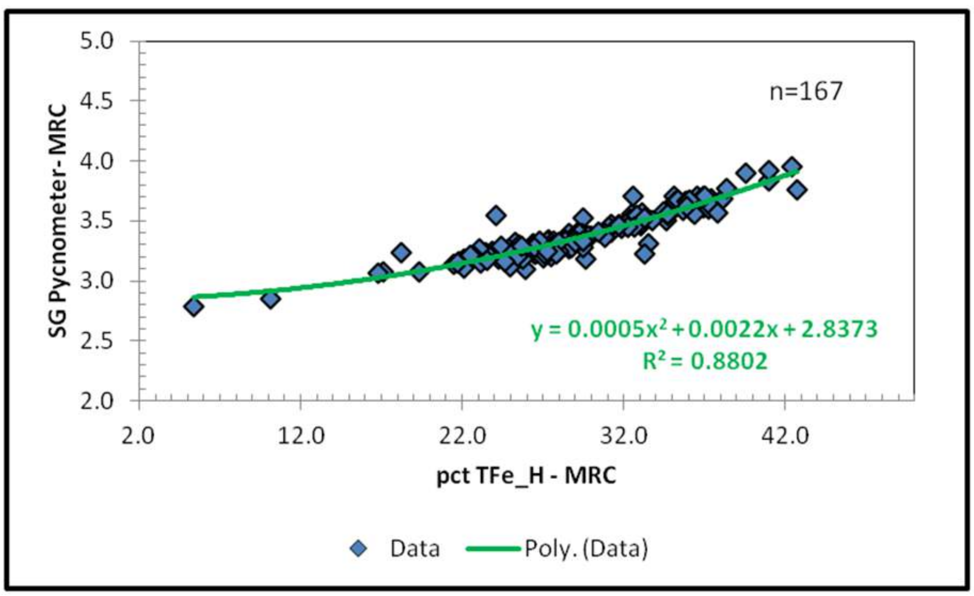


Figure 11.26 – MRC SG Pycnometer vs. MRC TFe_Head



11.10 2013 Check Assaying Program

The updating of sample assays for the Mineral Resource estimate started with correcting some Satmagan values per SGS-Lakefield certificate CA02729-JUL12 which addressed some erroneous values that were detected prior to the Check assaying program. The Check assay review was conducted following comments to LOM by WGM dated June 6th, 2013 and a letter report to LOM dated June 14th, 2013. This review included a review of past Satmagan results that was independently performed by SGS-Lakefield. SGS-Lakefield's work is summarized in an email to Frank Condon from Val Murphy, dated June 27th, 2013 which includes a certificate of analysis CA02900-JUN13 for the re-assays for 135 samples (plus eight lab inserted QAQC samples) and a table of values listing typos in assays found by SGS-Lakefield for 8 samples.

Gestion Otelnuk Inc.'s field personnel independently started a review of program assays (Gestion Otelnuk Inc., 2013). This review following WGM's guidelines included:

1. Checking assays against logged geology for reasonableness.
2. Checking Satmagan results against hand-held magnetic susceptibility.
3. Checking Satmagan results against DT results where samples had determinations by both methods.
4. Checking Satmagan results against TFe.
5. Checking TFe against SG.
6. Checking results for FBLKs and FDUPs.

As a result of this review new assays for 398 samples (excluding SGS-Lakefield QA/QC samples) were reported by SGS Lakefield in certificates CA03146-Sep13, CA02379-Sep13, CA02378-Sep13, CA02138-Sep13, CA03212-Jul13 and CA03215-Jul13). The requested samples included samples where a particular issue was suspected ("Issue Samples") but also included a number of samples that bordered or shouldered the "Issue" samples in the sampling sequence.

SGS-Lakefield was requested to retrieve the rejects of these selected samples from storage, prepare new pulps and assay each for XRF-WR, Satmagan and SG. SGS-Lakefield could not locate all of the selected samples. Most of the samples were re-assayed for XRF-WR, Satmagan and SG, but for several only Satmagan was determined. Five (5) of the samples were re-cut from 2012 program core towards resolving possible sample mix-up issues. A few samples intended for re-assay were not requested so Check assays for these samples were missed.

The new assays were reviewed by Gestion Otelnuk Inc.'s project personnel, compared against original assays and in each case a determination was made whether the new assay was to substitute for the original assay in the database as the final assay for a sample. One hundred and sixteen (116) of the new sample assays of the 398 Check assays completed for this component of the program became new final assays in the database replacing original assays.

The Check assayed program did result in improved assay quality as some erroneous assays were replaced with more accurate values. Some sample/assay issues remain unresolved due to conflicting results and missed samples and lost rejects and doubtless more Check assaying could have been undertaken. WGM advised ADI/LOM that for some of the outstanding cases additional immediate work to follow-up outstanding issues was not a high priority because the specific issues were outside of the Mineral Resource area. More re-sampling of previously split and sampled drill core would provide more unambiguous corrections.

The Project database was revised with the new assays. For WGM's Mineral Resource estimate Davis Tube results generally take precedence over Satmagan MagFe but for the revised Checked assays Satmagan results take precedence over original Davis Tube test results because no Davis Tube tests were done as part of the Check assays work.

11.11 Conclusions

WGM believes that sampling and assaying for LOM and Adriana's programs since 2007 have been performed reasonably well. The follow-up process including database development, QA/QC assessment, tracking and re-assaying of samples and interpretation of results could however have been handled much better. The database structure remains awkward with some data in separate MS Excel spreadsheet files and other data in Gemcom™ databases. Sample identifiers are not completely consistent between SGS-Lakefield and field/database records and drillhole identifiers have also been revised (slightly) in the database more than once through the project. The resulting database is not readily useable for tracking QA/QC issues and making revisions to sample assays and a small proportion of assays completed are not in the database.

Regardless of these shortcomings most sample assays are error-free, accurate and representative of mineralization. Improved proficiency for manipulating data in the field, improved understanding of the meaning of the assays and better software would have mitigated some of the difficulties. The database remains incomplete for some minor details. WGM's assay revisions based on Check assaying were done in part independently of LOM/ADI so for a very small proportion of samples WGM's final assays are different than what LOM/ADI has designated in their Project database.

12.0 DATA VERIFICATION

Drill core samples collected by LOM and Adriana from 2007 through 2012 were submitted on a routine basis by Adriana or LOM to SGS Lakefield. LOM in 2012 also operated a Check assay program which involved sending selected samples to MRC for assay. In addition to this sample assaying by Adriana and LOM, WGM in 2007 and 2008 collected Independent samples for assay. These WGM samples were also submitted to SGS-Lakefield. WGM also managed an assay program at MRC in 2007 and 2008. SGS-Lakefield is an accredited laboratory meeting the requirements of ISO 9001 and ISO 17025. MRC is not accredited, but is well respected. Although WGM has reviewed the assay results and a selection of Certificates generated by SGS-Lakefield and MRC believes they are generally accurate, WGM is relying on the laboratory as an independent expert in the field of analyses.

12.1 General

WGM geologists have made three visits to the Property but none recently. Mr. John Sullivan, P. Geo. (former-WGM geologist) visited the Property in September 2005 and viewed the Property and historic drill core in storage. Mr. Richard Risto visited the Property from August 2nd to August 31st, 2007 and again from September 13th to 16th, 2008. Mr. Risto's first visit was made during Adriana's first drill program, and drilling and core logging were in progress. At the time of Mr. Risto's 2008 site visit, drilling and core logging was finished for the season. Some core sampling was still to be completed, but at the time of the visit was in hiatus.

Ex-Senior WGM Associate Geologist, Mr. Buzz Neal, P. Eng., designer of the initial sampling and test work program for Adriana, was also a consultant for the historic programs on the Property in the 1970s and visited the Property at that time.

Mr. Sullivan's comments concerning the Property, prior to Adriana's drill programs are contained in WGM's first NI 43-101 report for Adriana, dated, November 24th, 2005 and available on SEDAR.

During Mr. Risto's first site visit in August 2007, WGM:

- observed core handling, core logging and sampling in progress and discussed field practice with geologists present;
- reviewed drill core and compared findings concerning geology and sampling against drill logs and sampling records;
- independently collected six (6) second half core samples;
- visited the drills in progress drilling the Property;
- viewed a number of drillhole collars of drillholes already completed; and
- checked drillhole collars locations with a hand-held GPS to validate collar locations.

During Mr. Risto's second site visit in September 2008, he again performed all of the foregoing, including the collection eleven (11) additional independent second half core samples, but could not view core logging and sampling in progress.

Geology and lithological units in drill core were found to match the drill core logs and sample intervals marked in core trays matched sampling records. WGM recommended more description in the core logs and better qualification of contacts between units. Coordinates for drillhole collars were found to reasonably match IOS records.

12.2 Independent Sampling by WGM

The second half core samples independently collected by WGM were placed in plastic sample bags closed with tamper evident factory numbered closures. The sample IDs were unique to the samples and did not report original sample numbers, or drillhole ID or meterage. Sample identities were consequently blind to both IOS/Adriana and SGS-Lakefield and only known to WGM. The samples were shipped to WGM's Toronto office where on arrival the bags and closures were inspected. The samples were then sent on to SGS-Lakefield for preparation, assay and test work following the flow sheet for routine samples (see Figure 11.4). Selected results for WGM's independent samples are shown in Table 12.1. Results for %TFe Heads, %TFe DTC and %SiO_2 DTC are plotted (Figure 12.1, Figure 12.2 and Figure 12.3).

Results for original and WGM independent second half core samples correlate well. Results indicate that IOS/Adriana sampling is reliable and no sample sequencing errors are apparent. The results also provide a measure of field sampling variance.

Figure 12.1 – % TFe_H (Original) versus WGM Independent Sample

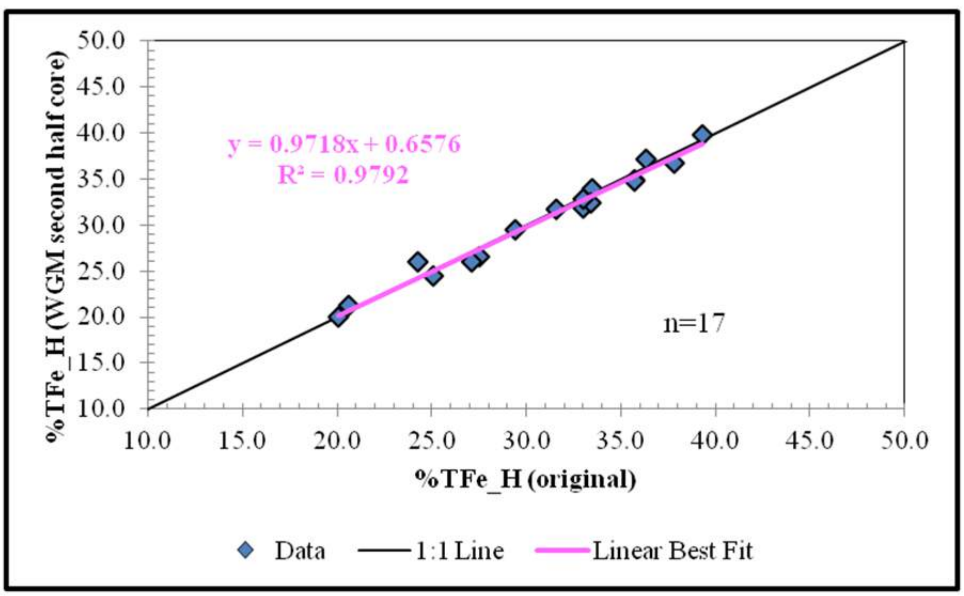


Table 12.1 – Summary of WGM Independent Sampling Results

Sample ID	WGM Sample ID	Heads										DT Magnetic Concentrates			
		Orig % Fe	WGM % Fe	Orig % SiO_2	WGM % SiO_2	Orig % Al_2O_3	WGM % Al_2O_3	Orig % Mn	WGM % Mn	Orig % DTWR	WGM % DTWR	Orig % Fe	WGM % Fe	Orig % SiO_2	WGM % SiO_2
62510076	WGMAD01	31.60	31.70	45.10	44.30	0.21	0.25	0.75	0.76	36.3	35.60	68.20	69.00	4.65	4.02
62510154	WGMAD02	27.50	26.50	45.80	46.20	0.02	0.05	1.11	1.04	15.8	15.00	64.10	64.20	6.91	6.70
62510137	WGMAD03	33.50	33.90	37.70	38.60	0.31	0.16	1.50	1.64	32.9	32.30	69.40	69.10	2.82	2.63
62510133	WGMAD04	27.10	26.00	44.80	43.70	0.005	0.005	0.65	0.73	12.1	12.60	69.90	71.00	2.88	2.14
62510116	WGMAD05	39.30	39.80	42.30	41.20	0.13	0.11	0.13	0.15	49.3	50.20	70.70	71.00	2.33	2.17
62510905	WGMAD06	37.80	36.80	36.20	37.20	0.07	0.07	0.73	0.73	24.1	23.50	71.00	71.50	1.45	1.46
194639	ADWGM-07	20.60	21.20	45.40	41.50	0.08	0.21	0.34	0.32	6.5	5.40	65.60	66.10	5.53	4.37
194668	ADWGM-08	24.30	26.00	46.10	41.60	0.16	0.15	0.57	0.59	13.7	15.30	68.80	68.10	2.57	2.71
194611	ADWGM-09	35.70	34.90	45.50	45.60	0.07	0.09	0.25	0.25	23.6	16.70	70.40	70.90	2.57	2.32
194510	ADWGM-10	36.30	37.20	33.40	30.80	0.16	0.17	1.15	1.21	30.7	29.70	68.10	68.60	2.39	2.60
194423	ADWGM-11	33.00	31.80	41.50	39.70	0.11	0.06	1.07	1.17	32.2	28.20	66.40	67.90	5.81	4.97
194363	ADWGM-12	25.10	24.40	46.60	48.20	0.08	0.005	1.10	1.17	13.5	12.50	68.60	67.20	3.13	3.55
194208	ADWGM-13	33.40	32.40	44.00	43.40	0.14	0.17	0.69	0.81	31.2	28.00	67.50	67.20	4.33	4.43
194182	ADWGM-14	33.00	32.90	39.90	38.30	0.15	0.19	1.54	1.55	6.7	6.40	67.30	67.90	4.50	5.29
194066	ADWGM-15	29.40	29.50	43.30	44.30	0.05	0.12	0.36	0.34	8.7	7.20	68.70	69.70	3.21	2.90
194778	ADWGM-16	35.70	34.80	44.20	43.70	0.26	0.23	0.38	0.43	19.8	19.20	69.10	68.80	4.17	4.47
193975	ADWGM-17	20.10	20.00	49.40	47.50	0.29	0.29	0.24	0.25	0.5	0.50	-	-	-	-

Figure 12.2 – % SiO_2_H (Original) versus WGM Independent Sample

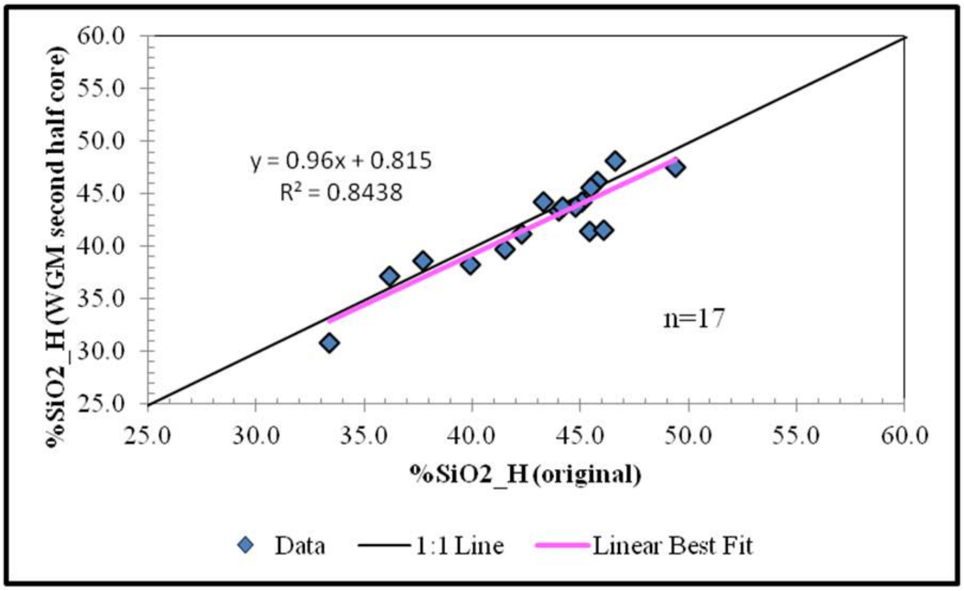
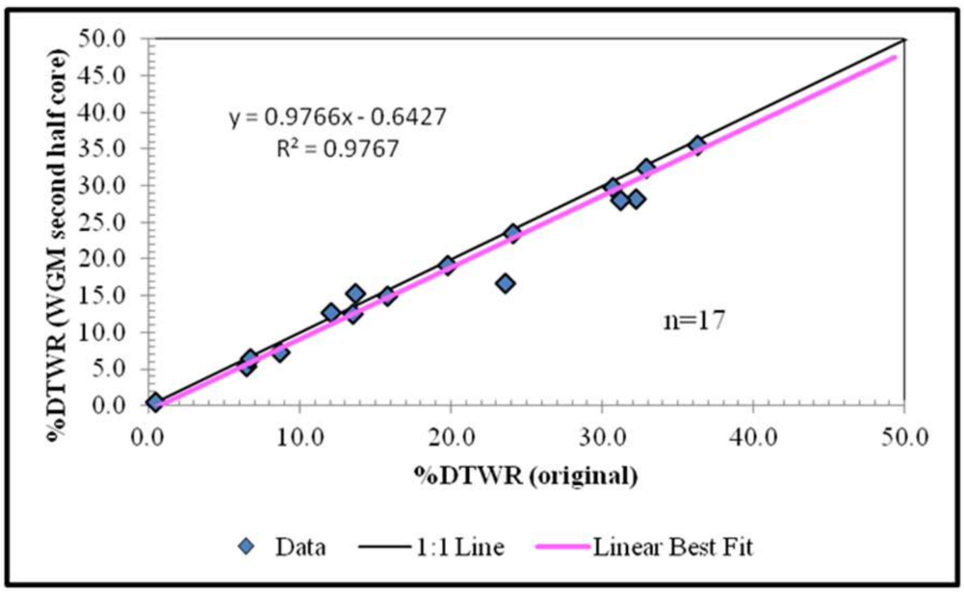


Figure 12.3 – % DTWR (Original) versus WGM Independent Sample



12.3 Check Assays at Midland Research Center 2007-2008

In early 2008 and again in early 2009, after all or almost all of the analytical work at SGS Lakefield had been completed for the 2007 and 2008 programs respectively, WGM independently selected samples for Check Assaying at MRC. The samples selected spanned different iron grades, DTWR and silica levels. From 2007 and 2008 archived materials in storage at SGS, -10 mesh rejects and magnetic DTCs were selected. For most of these samples, both fractions: -10 mesh rejects and DTC pulps were available, but for some samples, DTCs were unavailable because DTWRs were low. Some of the samples selected by WGM were selected purposely because DTWRs were low.

The list of samples selected by WGM was forwarded to SGS-Lakefield. SGS-Lakefield riffled out approximately 100g of -10 mesh material from rejects in storage and sent the remaining DTC pulp (each 2 to 5 g) to WGM. On receiving the samples, WGM checked the samples and forwarded them on to MRC.

12.3.1 Davis Tube Concentrates

MRC was requested to determine iron and silica in each of the magnetic DT concentrates, previously prepared by SGS-Lakefield, using its standard analytical methods. MRC uses wet chemical methods of analysis, as opposed to instrumental methods used at SGS-Lakefield. For determining Fe, MRC uses a non-mercury titration method using titanium chloride and titrating with potassium dichromate, following sample digestion using stannous chloride, HCL and HF. MRC determines silica by weighing sample residues before and after selective digestion and two stages of fusion in platinum crucibles.

Results for the Check Assaying by MRC on each of the DT magnetic concentrates, along with original SGS-Lakefield results, are listed in Table 12.2. Figure 12.4 and Figure 12.5 show a comparison of %TFe and %SiO_2 results between the two labs. Results for two samples requested from SGS-Lakefield and shipped to MRC (62510055 & 6251076) are not shown. These two samples were mixed up at MRC and results are not reported here.

Table 12.2 – MRC Check Assays on DTCs Made by SGS-Lakefield

Hole ID	Sample ID	SGS-Lakefield				MRC	
		% TFe H	% DTWR	% Fe DTC	% SiO_2 DTC	% Fe DTC	% SiO_2 DTC
LO-S-1003	62510060	28.2	25.9	70.4	1.4	70.75	1.30
LO-S-1003	62510070	29.3	33.5	69.8	2.6	70.30	2.36
LO-S-1003	62510075	30.6	18.0	69.8	2.9	69.48	2.76
LO-S-1007	62510098	36.4	41.3	68.6	2.8	68.95	2.78
LO-S-1007	62510100	28.0	34.0	70.5	1.8	70.90	1.70
LO-S-1007	62510102	23.9	30.2	67.6	5.7	67.46	5.50
LO-S-1007	62510104	26.8	27.1	69.7	1.8	70.45	1.62
LO-S-1007	62510109	30.2	35.3	70.0	2.6	70.30	2.40
LO-S-1005	62510115	34.4	38.9	70.1	2.1	70.75	2.12
LO-S-1005	62510118	35.4	42.8	69.9	2.5	70.15	2.50
LO-S-1005	62510121	33.4	26.0	69.6	2.1	69.25	2.08
LO-S-1005	62510126	25.3	27.0	69.5	2.8	69.25	2.78
LO-S-1005	62510129	28.7	27.5	70.3	1.7	70.53	1.72
LO-S-1009	62510154	27.5	15.8	64.1	6.9	63.86	6.88
LO-S-1009	62510157	27.4	31.1	68.4	4.9	67.75	4.88
LO-S-1009	62510159	22.9	25.5	67.6	5.0	65.96	8.21
LO-S-1011	62510180	29.7	32.4	68.2	3.2	69.40	3.02
LO-S-1011	62510183	25.4	25.5	67.0	4.7	67.60	4.64
LO-S-1011	62510186	27.3	27.5	69.5	2.2	70.15	1.96

Hole ID	Sample ID	SGS-Lakefield				MRC	
		% TFe H	% DTWR	% Fe DTC	% SiO_2 DTC	% Fe DTC	% SiO_2 DTC
LO-S-1011	62510188	28.2	25.7	68.1	4.1	68.95	4.42
LO-S-1011	62510192	27.4	29.6	69.2	2.3	70.45	1.86
LO-S-1013	62510218	28.2	25.1	69.4	1.9	70.72	1.00
LO-S-1013	62510220	27.4	26.1	69.3	1.8	70.15	1.36
LO-S-1013	62510226	24.9	21.6	71.0	1.8	70.53	1.46
LO-S-1009	62510903	24.7	28.9	66.1	7.2	65.06	6.92
LO-S-1011	62510907	36.9	42.6	67.3	3.2	68.20	3.01
LO-S-1032	193860	28.4	29.7	69.3	2.6	69.27	2.54
LO-S-1028	193881	37.1	45.1	68.4	4.2	68.13	4.04
LO-S-1029	193920	27.7	33.2	66.8	6.3	67.71	6.22
LO-S-1031	193953	33.9	17.2	70.5	2.0	69.85	1.82
LO-S-1035	194013	23.6	20.7	68.9	3.0	69.10	2.72
LO-S-1037	194075	30.8	33.5	69.1	2.8	69.63	2.70
LO-S-1038	194112	24.9	24.4	68.7	3.5	69.10	3.34
LO-S-1041	194196	28.6	26.1	68.4	3.0	68.95	2.88
LO-S-1042	194220	28.4	27.2	68.3	3.2	68.95	3.00
LO-S-1043	194245	26.9	21.3	69.3	2.2	69.93	1.94
LO-S-1045	194299	28.1	24.6	68.3	3.4	68.35	3.18
LO-S-1047	194356	24.8	22.8	67.4	4.7	67.46	4.63
LO-S-1050	194437	28.7	28.4	65.5	6.9	65.96	6.50
LO-S-1054	194516	29.3	25.7	68.3	2.8	69.03	2.80
LO-S-1054	194529	29.1	25.6	65.4	6.1	67.01	5.96
LO-S-1055	194541	35.6	41.4	68.1	3.3	69.03	3.14
LO-S-1056	194582	27.8	31.8	67.5	5.0	66.56	5.98
LO-S-1058	194669	29.4	17.8	68.1	4.3	68.50	4.16
LO-S-1059	194707	27.2	27.4	70.7	2.2	70.15	1.98
LO-S-1059	194721	27.7	19.0	68.3	3.2	69.55	3.30
LO-S-1060	194747	36.6	30.2	65.6	5.2	65.96	5.04
LO-S-1061	194793	27.5	25.9	67.2	5.5	67.16	5.62
LO-S-1063	194824	26.9	26.3	66.0	6.1	66.86	6.04
LO-S-1063	194826	30.1	29.4	67.4	5.0	68.05	5.08
LO-S-1003	62510060B	28.2	25.1	70.5	1.5	70.60	1.58
LO-S-1007	62510100B	28.2	33.0	70.9	1.8	70.90	1.74

Figure 12.4 for %TFe in DTCs indicates that results returned from MRC are on average slightly higher than SGS-Lakefield assays. The difference is, however, less than 0.5 % TFe. Iron assays remain strongly correlated and assays on Duplicates mostly fall within the ±1 % limit lines. This degree of scatter is closely similar, but a little less tight, than the scatter patterns for analysis replicates by SGS-Lakefield.

Figure 12.4 – MRC Check Assay Results for % TFe on Same DTC

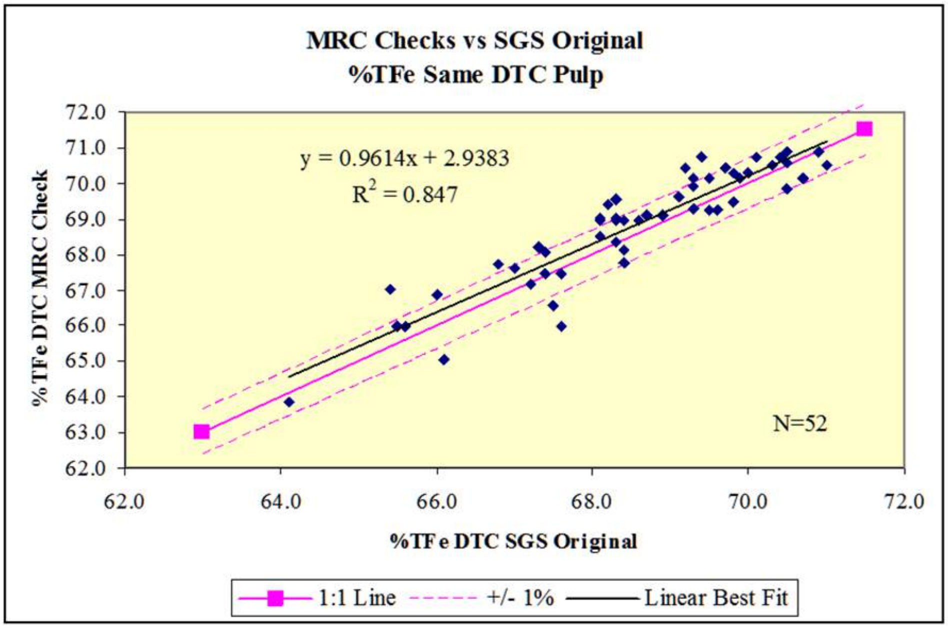
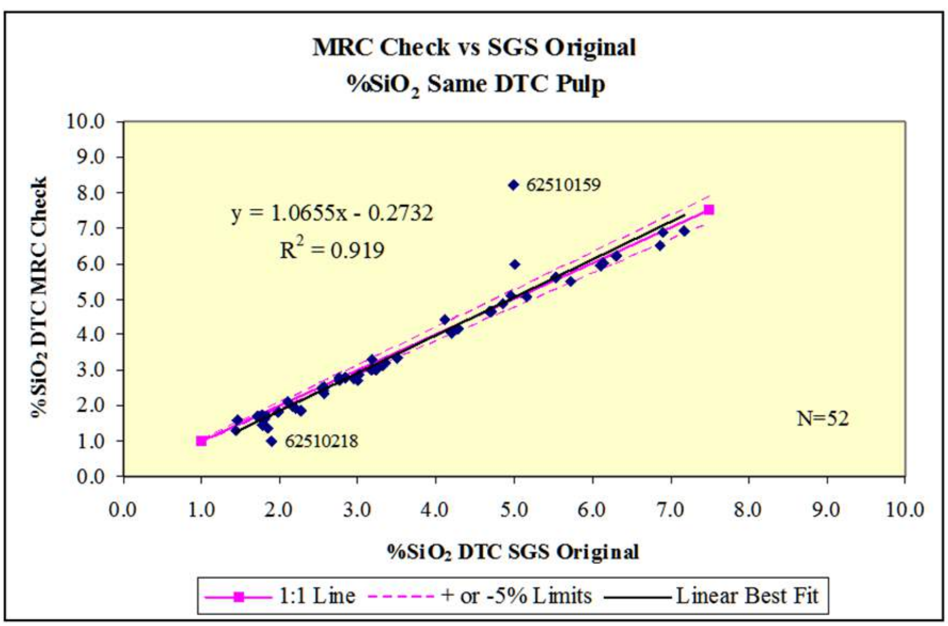


Figure 12.5 for %SiO_2 results for DTCs originally prepared and assayed by SGS-Lakefield indicates that no assay bias is apparent for silica. Most of the sample pairs plot within the ±5 % envelops similar to SGS-Lakefield replicate assays, but not quite as tightly. A couple of samples show poor correlation between MRC and SGS-Lakefield. The DTC for sample 62510218, provided to MRC was light at 0.8 g, and this is likely the explanation for this sample. No explanation for the poor correlation for sample 62510159 is known.

Figure 12.5 – MRC Check Assay Results for % SiO_2 on Same DTC



12.3.2 Minus 10 Mesh Rejects

MRC was requested to complete a determination of TFe on each Head and then make DTCs for each, and analyse each concentrate for %TFe and %SiO_2. For the 2007 program samples, WGM requested that MRC attempt to create DT feed samples that were 85 to 90 % -325 mesh. For the 2008 program samples, WGM requested that MRC pulverize each sample to 100 % -325 mesh as per their standard method. MRC uses a multi-stage, mechanical mortar and pestle grinding method with dry screening between stages to reach the point where 100 % of the sample passes the prescribed screen.

Results for the test work and analysis of the -10 mesh rejects are compiled in Table 12.3 and shown on Figure 12.6 through Figure 12.10. Figure 12.6 and Figure 12.7 for %TFe on Heads and %DTWR show that MRC and SGS-Lakefield assays are strongly correlated and no significant assay bias is apparent.

Figure 12.8 and Figure 12.9 show poorer correlation for both iron and silica in DT concentrates. This is not unexpected and is a function of the different grinding routines used by the two labs. At SGS-Lakefield, the samples as described in Section 11.3, were ground in a ring pulverizer for 90 seconds. At MRC, samples are pulverized in a mortar and pestle and multiple stages are used with over-size screened out and re-pulverized at each stage until all passes the 325 mesh screen. The two (2) methods of pulverization consequently generate DT feeds and concentrates with different particle size distributions. Figure 12.10 shows that iron concentrations in MRC prepared DT concentrates are on average slightly higher than for SGS-Lakefield prepared DTCs.

Table 12.3 – Check Assay Results for -10 Mesh Rejects

Hole ID	Sample ID	SGS-Lakefield Assays					MRC Assays				
		% Fe H	% DTWR	% Fe DTC	% SiO_2 DTC	% Mag Fe	% TFe H	% DTWR	% Fe DTC	% SiO_2 DTC	% Mag Fe
LO-S-1003	62510060	28.2	25.9	70.4	1.43	18.2	28.27	25.50	70.91	1.36	18.1
LO-S-1003	62510070	29.3	33.5	69.8	2.57	23.4	29.01	34.50	70.23	2.36	24.2
LO-S-1003	62510075	30.6	18.0	69.8	2.94	12.6	30.73	19.00	68.35	4.12	13.0
LO-S-1007	62510098	36.4	41.3	68.6	2.84	28.3	36.59	43.00	67.96	3.52	29.2
LO-S-1007	62510100	28.0	34.0	70.5	1.82	24.0	28.35	34.00	71.82	1.58	24.4
LO-S-1007	62510102	23.9	30.2	67.6	5.72	20.4	24.04	31.00	64.96	9.22	20.1
LO-S-1007	62510104	26.8	27.1	69.7	1.81	18.9	27.44	28.00	70.01	1.82	19.6
LO-S-1007	62510109	30.2	35.3	70.0	2.55	24.7	29.79	35.50	70.76	1.72	25.1
LO-S-1005	62510115	34.4	38.9	70.1	2.1	27.3	34.77	39.00	70.15	1.92	27.4
LO-S-1005	62510118	35.4	42.8	69.9	2.53	29.9	34.75	43.00	69.81	2.46	30.0
LO-S-1005	62510121	33.4	26.0	69.6	2.12	18.1	32.94	26.50	68.30	2.24	18.1
LO-S-1005	62510126	25.3	27.0	69.5	2.75	18.8	25.16	28.00	68.00	3.20	19.0
LO-S-1005	62510129	28.7	27.5	70.3	1.71	19.3	28.33	27.50	70.11	1.82	19.3
LO-S-1009	62510154	27.5	15.8	64.1	6.91	10.1	26.74	17.00	62.93	6.44	10.7
LO-S-1009	62510157	27.4	31.1	68.4	4.85	21.3	27.35	32.00	67.77	4.34	21.7
LO-S-1009	62510159	22.9	25.5	67.6	4.99	17.2	22.66	27.50	65.12	9.28	17.9
LO-S-1011	62510180	29.7	32.4	68.2	3.16	22.1	29.69	33.00	69.35	3.08	22.9

Hole ID	Sample ID	SGS-Lakefield Assays					MRC Assays				
		% Fe H	% DTWR	% Fe DTC	% SiO_2 DTC	% Mag Fe	% TFe H	% DTWR	% Fe DTC	% SiO_2 DTC	% Mag Fe
LO-S-1011	62510183	25.4	25.5	67.0	4.7	17.1	25.46	27.50	66.78	5.24	18.4
LO-S-1011	62510186	27.3	27.5	69.5	2.19	19.1	27.50	28.00	69.66	2.42	19.5
LO-S-1011	62510188	28.2	25.7	68.1	4.11	17.5	28.10	27.50	68.00	5.24	18.7
LO-S-1011	62510192	27.4	29.6	69.2	2.26	20.5	27.88	29.50	69.80	2.44	20.6
LO-S-1013	62510218	28.2	25.1	69.4	1.9	17.4	28.10	25.00	70.18	1.64	17.5
LO-S-1013	62510220	27.4	26.1	69.3	1.84	18.1	27.20	27.00	70.11	1.62	18.9
LO-S-1013	62510226	24.9	21.6	71.0	1.78	15.3	24.78	22.50	70.18	1.74	15.8
LO-S-1009	62510903	24.7	28.9	66.1	7.18	19.1	24.93	30.00	64.67	7.96	19.4
LO-S-1011	62510907	36.9	42.6	67.3	3.24	28.7	36.57	42.50	68.00	3.98	28.9
LO-S-1013	62510909	26.7	21.2	71.2	1.54	15.1	26.82	21.50	71.02	1.20	15.3
LO-S-1032	193860	28.4	29.7	69.3	2.58	20.6	28.18	30.50	70.08	2.26	21.4
LO-S-1028	193881	37.1	45.1	68.4	4.2	30.8	37.70	48.00	67.60	3.72	32.4
LO-S-1028	193902	41.2	0.5			0.0	40.92	1.00			0.0
LO-S-1031	193956	23.7	9.4	67.8	4.87	6.4	23.38	10.00	69.78	2.82	7.0
LO-S-1035	194013	23.6	20.7	68.9	3	14.3	23.83	20.50	69.85	1.98	14.3
LO-S-1037	194075	30.8	33.5	69.1	2.75	23.1	31.25	34.00	69.85	2.28	23.7
LO-S-1038	194112	24.9	24.4	68.7	3.5	16.8	25.26	24.50	70.90	1.58	17.4
LO-S-1040	194180	39.8	7.3	61.4	10.2	4.5	39.12	10.50	60.26	12.20	6.3
LO-S-1041	194196	28.6	26.1	68.4	3.01	17.9	29.16	27.00	70.90	1.38	19.1
LO-S-1043	194245	26.9	21.3	69.3	2.2	14.8	26.01	21.00	71.05	1.36	14.9
LO-S-1045	194299	28.1	24.6	68.3	3.35	16.8	27.51	23.50	70.45	1.78	16.6
LO-S-1047	194356	24.8	22.8	67.4	4.68	15.4	24.43	22.00	69.63	2.98	15.3
LO-S-1047	194362	19.2	9.6	65.6	5.43	6.3	19.19	9.00	69.55	2.08	6.3
LO-S-1048	194400	36.7	2.0			0.0	35.15	2.00	68.05		1.4
LO-S-1050	194437	28.7	28.4	65.5	6.87	18.6	28.26	27.50	69.85	2.78	19.2
LO-S-1051	194467	41.4	5.8	58.2	12	3.4	39.42	5.50	62.21	9.10	3.4
LO-S-1054	194516	29.3	25.7	68.3	2.82	17.6	29.68	26.00	70.75	2.56	18.4
LO-S-1054	194529	29.1	25.6	65.4	6.1	16.7	29.68	26.00	67.46	5.24	17.5
LO-S-1055	194541	35.6	41.4	68.1	3.32	28.2	36.13	41.50	70.15	2.12	29.1
LO-S-1057	194625	23.9	12.0	68.5	2.42	8.2	23.83	11.00	71.25	1.12	7.8
LO-S-1059	194707	27.2	27.4	70.7	2.17	19.4	26.98	29.50	71.20	1.30	21.0
LO-S-1060	194747	36.6	30.2	65.6	5.16	19.8	35.60	30.50	67.23	3.56	20.5
LO-S-1060	194759	24.5	6.4	69.9	2.76	4.5	24.28	6.50	71.35	1.20	4.6
LO-S-1061	194789	28.5	2.5	69.4	2.92	1.7	28.33	4.00	71.35		2.9
LO-S-1063	194824	26.9	26.3	66.0	6.14	17.4	26.31	31.50	67.38	3.56	21.2

Notes: 1. Magnetic Fe calculated by multiplying % Fe DTC by % DTWR.

Figure 12.6 – MRC Check Assay Results for % TFe

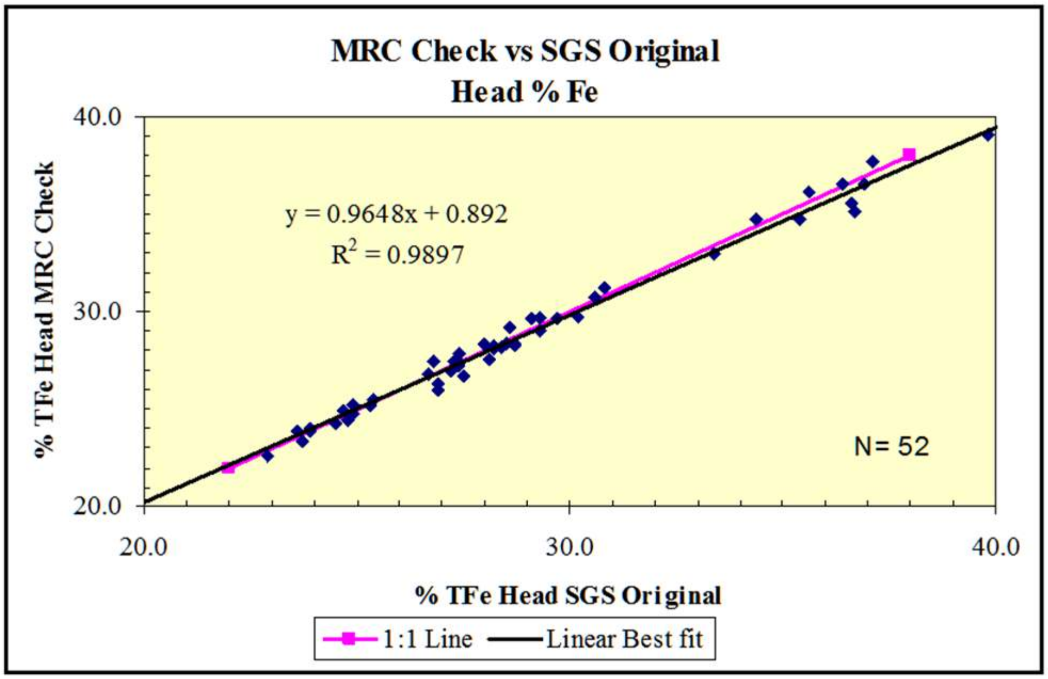


Figure 12.7 – MRC Check Assay Results for % DTWR

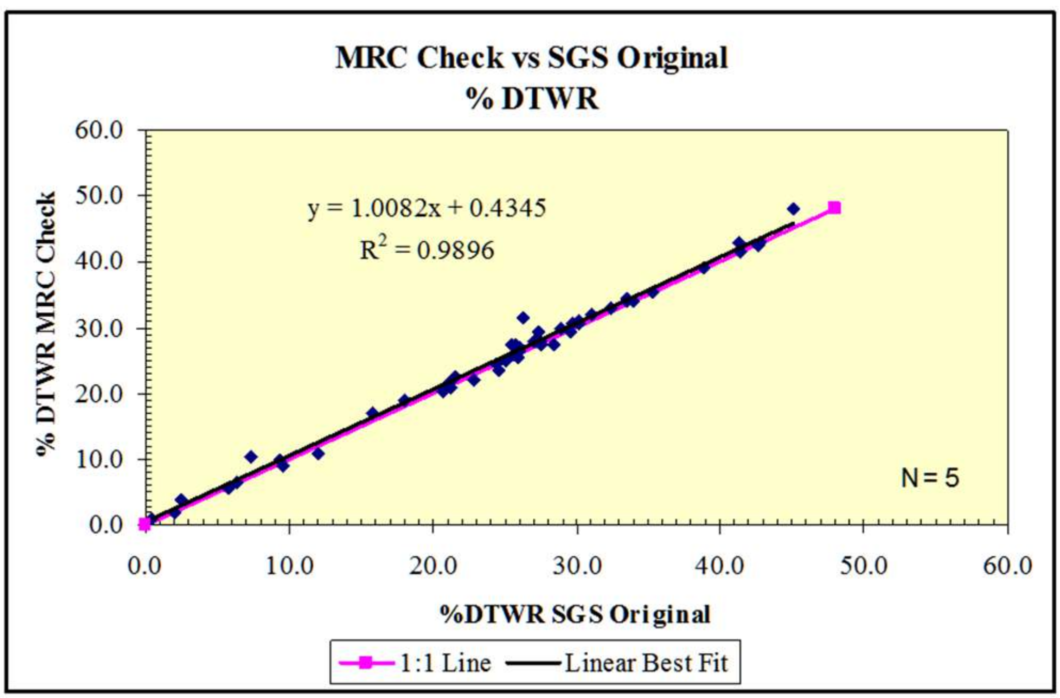


Figure 12.8 – MRC Check Assay Results for % TFe DTC

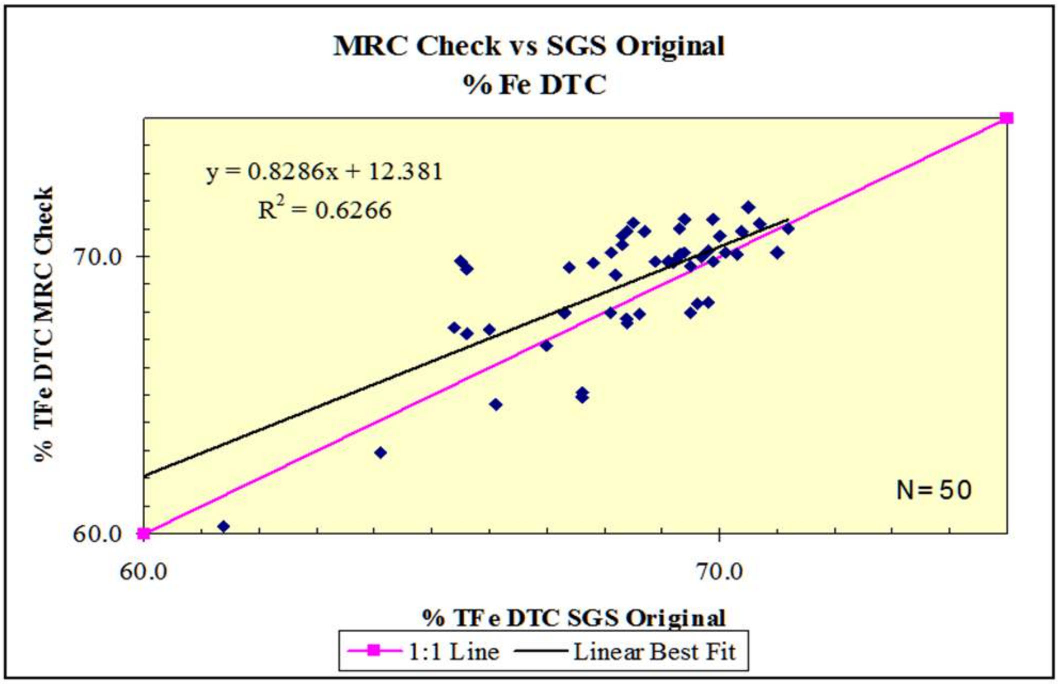


Figure 12.9 – MRC Check Assay Results for % SiO_2 DTC

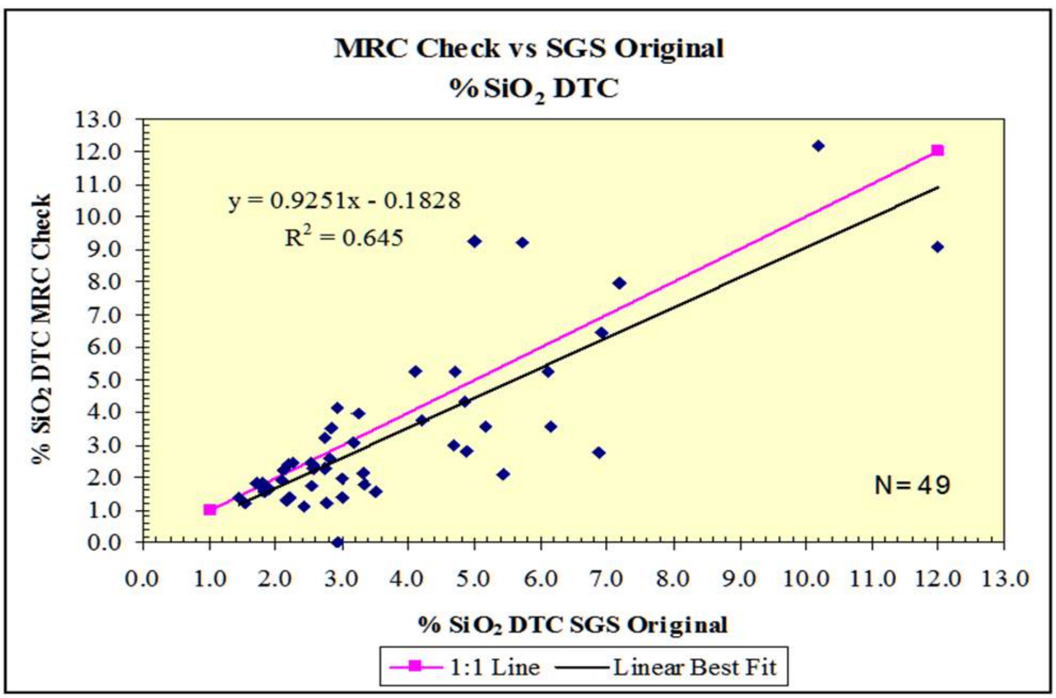
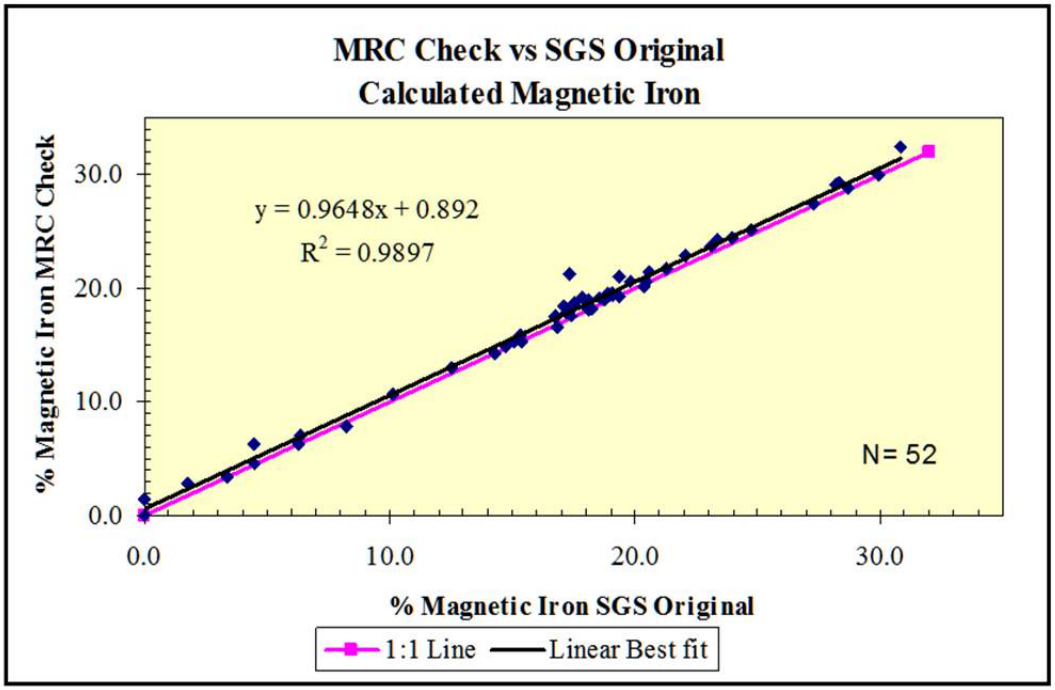


Figure 12.9 shows that silica is lower in MRC prepared DTCs than in SGS-Lakefield prepared concentrates. These results for iron and silica suggest that degree of liberation in MRC prepared concentrates is slightly better than SGS-Lakefield prepared concentrates. Again, this is not unexpected. MRC grinding adapts to the changing hardness of the rock. Results for calculated magnetic iron are shown on Figure 12.10. Again, results between the two labs are strongly correlated and no significant bias is evident.

Figure 12.10 – MRC Check Assay Results for Magnetic Fe



12.4 Other WGM Validation

1. Checked random assay certificates received directly from SGS-Lakefield against assays reported in the Project database.
2. Completed Section 11 of this report concerning assaying and QA/QC which included a large component of sample/assay data analysis and review.

13.0 MINERAL PROCESSING AND METALLURGICAL TEST WORK

This section addresses the mineral processing and metallurgical test work that has been carried out for the Lac Otelnuk deposit located in the Nunavik region of the province of Quebec, about midway north in the Labrador Trough iron range. Iron deposits in this part of the Trough are taconite, or weakly metamorphosed iron formation.

The Lac Otelnuk mineralization has been subjected to a series of metallurgical testing programs beginning in 1971. Mineral processing and metallurgical testing prior to 2009 are well documented and summarized by geological and mining consultants WGM reports in 2005 and 2009. These reports are available on SEDAR.

The process has evolved through various stage of development, the most recent one being a feasibility study (FS) for the production of 50 Mt of concentrate per annum completed by SNC-Lavalin in March 2015. The concentrate targeted is a pellet feed with a Fe grade over 68% and a SiO_2 grade under 4%.

13.1 Process Development Stages

This section of the Report outlines the evolution of metallurgical test work programs undertaken to date and the concurrent process development which has established the process flow sheet and design data for the Lac Otelnuk ore.

The beneficiation of taconite ores is based on a multi circuit crushing and grinding necessary to reach sufficient magnetite mineral liberation with intercalating low intensity magnetic separation stages in order to eliminate tailings as early as possible in the process and therefore reduce the capacity of the fine grinding stages before obtaining a final concentrate with an acceptable grade.

The following Table 13.1 summarises the history of the test work performed to date.

Table 13.1 – History of Metallurgical Test Work

Date	Project Owner	Laboratory	Brief Description	SNC-Lavalin Supervision
1971-1974	King Resources Company	Lakefield Canada	Laboratory test work on drill cores	No
1978	King Resources Company	SGA Germany	Pilot beneficiation and pelletizing tests	No
1981	King Resources Company	SGS Lakefield Canada	Pilot beneficiation on bulk sample	No
2007-2008	Adriana Resources Inc.	SGS Lakefield Canada Midland Research Center COREM	Laboratory characterization and grindability testing with Davis Tube QA/QC program and secondary laboratory check	No
2010	Adriana Resources Inc.	WISCO Kaisheng science & technology China	Laboratory beneficiation and grindability testing	No

Date	Project Owner	Laboratory	Brief Description	SNC-Lavalin Supervision
2011 (February)	Adriana Resources Inc.	SGS Lakefield Canada	Grindability on 39 samples	No
2011 (June)	Adriana Resources Inc.	SGS Lakefield Canada	Geometallurgical investigation	No
2011 (September)	Adriana Resources Inc.	SGS Lakefield Canada	Mineralogical Characteristics of Ninety Eight Ore Variability Samples	No
2013 (January)	Adriana Resources Inc.	SGS Lakefield Canada	The Characterisation of 15 composites from the Lac Otelnuk deposit	No
2013 (March)	Adriana Resources Inc.	SGS Lakefield Canada	SAG Power Index and Crusher Index Tests on 221 Core Samples	No
2013 (March)	Adriana Resources Inc.	SGS Lakefield Canada	Grindability Characterization of 16 PQ Core Samples	No
2013 (June)	Lac Otelnuk Mining Ltd	SGS Lakefield Canada	Beneficiation testing on a single composite	Test program Definition Feasibility Study Phase 1
2013 (June)	Lac Otelnuk Mining Ltd	SGS Lakefield Canada	Solid liquid separation and rheological testing of a concentrate sample from Lac Otelnuk deposit	Test program Definition Feasibility Study Phase 1
2013 (October)	Lac Otelnuk Mining Ltd	SGS Lakefield Canada	Beneficiation testing on 5 composites	Test program Definition Feasibility Study Phase 2
2014 (February)	Lac Otelnuk Mining Ltd	SGS Lakefield Canada	Pilot plant testing on a bulk sample from the Lac Otelnuk deposit.	Test program Definition and supervision with LOM Feasibility Study Phase 2
2014 (May)	Lac Otelnuk Mining Ltd	SGS Lakefield Canada	Davis Tube characterization of two composites from the Lac Otelnuk deposit.	Test program Definition Feasibility Study Phase 2
2014 (May)	Lac Otelnuk Mining Ltd	SGS Lakefield Canada	Sample preparation and rheology tests for two tailings samples from the Lac Otelnuk deposit.	Test program Definition Feasibility Study Phase 2

Extensive grinding and beneficiation test works have been performed on the Lac Otelnuk core samples. From this test work, two (2) points were highlighted to meet the final SiO_2 requirement of < 4 % in the concentrate: the grinding of the ore shall meet a very fine 80 % passing size of 38 µm and the LIMS concentrate shall be de-slimed.

Initially, early during the feasibility study, a high level review by SLI was conducted to evaluate three different coarse grinding options with a view to select the best process route for the project.

These three options were: (following the primary crushing stage)

- Secondary crushing and high pressure grinding rolls (HPGR);
- Semi autogenous grinding (SAG);
- Autogenous grinding (AG) and pebble crushing (PC).

HPGR appeared to be the most attractive technology from the NPV calculation but this technology was not considered as proven with an acceptable level of risk in terms of two identified critical flaws: ore moisture and freezing behaviour under sub-arctic conditions.

Therefore the SAG technology was selected for the feasibility study, with a recommendation to investigate both SAG and AG options during pilot trials.

A new testing program (which includes cobbing separation, fine grinding in two stages with roughing and cleaning magnetic separation stages, de-sliming stages, settling and filtration tests on concentrates) was developed by SLI early in the feasibility study to fill the gap left by previous studies.

Considering the very fine grind size of 38 µm obtained during the SGS laboratory test work, a process scheme with two stages of grinding with an intermediate LIMS was selected for further investigation. The slimes being liberated during the last fine stage of grinding had to be removed before the cleaner stage of LIMS in order to reach the SiO_2 requirement for the final concentrate.

Based upon a review of the test results, and the conclusions from metallurgical trade-off studies, SLI proposed the following two additional testing programs:

- Bench-scale test work on two master composites and three individual PQ cores: the first composite being representative of the whole 30 years mine life and the second being more representative of the first 10 years of operation. Three variability tests have been completed by using samples composed of PQ core samples taken from three zones of the deposit (North, South and Central) to assess the process response with different ore compositions.
- Pilot-plant test work of the whole selected process to be conducted on a bulk sample. The bulk sample was extracted at the surface by blasting to obtain a coarse product (minus 200 mm) consistent with a SAG/AG (semi-autogenous grinding / autogenous grinding) industrial feed.

The above two test programs have been completed during the feasibility study and are discussed in the section 13.6.

13.2 Ore Characterization

The mineralization of interest on the Lac Otelnuk property is the iron formation that consists mainly of magnetite (Fe_3O_4) and hematite (Fe_2O_3) in bands alternating with Chert carbonate or jasper. For the mineral resource estimate, WGM modelled the upper five geological sub-units (2A, 2B, 2C, 3A, 3B) of the Lac Otelnuk iron formation. From this, Met-Chem has developed a block model and a mining plan, which are described in the mining section of the Feasibility Study Report and discussed in Section 15 of this Report.

The above five geological (sub) units can be characterized using the following five (5) natural types of mineralization:

- Type I:

  Laminated Iron Ore. The iron mineral grain is over 70 % mainly magnetite, secondly hematite. The mineral grain morphology is complete and with strong metamorphism. The iron mineral grain which is larger than 32 µm accounts for 65.91 %. This mineralization belongs to a free milling ore type.

- Type II:

  Fragmented Iron Ore. The iron mineralization grain which is mainly concentrated at rudaceous and psammitic gangue is mainly magnetite, secondly hematite. The grain morphology is complete and with strong metamorphism. The iron mineral grain which is larger than 32 µm accounts for 59.88 %. This mineralization belongs to free milling ore type.

- Type III:

  Ball or Ring Iron Ore. The mineralization is ochre and the color is produced by fine-grain hematite, less than 2 µm, precipitated from ferrous carbonate phase grains. This type is mainly magnetite, secondly hematite with most morphology which is regular and with strong metamorphism. The iron mineral grain larger than 32 µm accounts for 47.93 %. The mineralization is free-milling but recovery is low, 14.1 %

- Type IV:

  Crystalline Iron Ore. The main feature is the fine iron mineral grain size; 16 µm or smaller is 78.91 %, which is mainly hematite

- Type V:

  Disseminated Crystalline Iron Ore. This ore is mainly in gangue and has many of the same features as Type IV.

The distribution of the five natural types of mineralization in each geological unit is as follows:

- Unit 2A is mainly Type II and secondly it is Type I. Together, Type II and Type I account for 82.8 % of the geological unit
- Unit 2B is mainly Type III and secondly Types I and II. Types I, II and III account for 97.8 % of the mineralization

- Unit 2C is mainly Type II and secondly it is Type I. Together, Type II and Type I account for 67 % of the mineralization, but Types IV and V account for 33 % and will have an obvious negative effect on iron recovery
- Unit 3A is mainly Type II and secondly it is Type I. Types I and II account for 96.7 % of the geological unit
- Unit 3B is mainly Type II and secondly it is Type I. Types I and II account for 98 % of the geological unit.

Given the above mineralogy distribution in the five geological units, it may be concluded that all five units belong to free-milling mineralization. Units 2B and 2C are complex and may be more problematic in the beneficiation process.

An investigation of an integrated sample of the raw mineralization demonstrated the following:

- The iron mineral is mainly present in the form of magnetite and may be recovered by low intensity magnetic separation. The magnetite is easily upgradable with low-intensity magnetic separation (LIMS), as long as the appropriate liberation grind is reached
- The iron in the form of iron carbonate mineralization (Fe content of 3.22 %) and hematite-limonite mineralization (Fe content of 3.64 %) accounts for 10.66 % and 12.05 % of the total iron
- After the removal of the magnetite, treating the non-magnetite iron with wet, high-intensity magnetic separation (WHIMS) results in a concentrate grade that is high in silica and therefore not up to the desired final product quality. This is because the hematite has a much finer liberation size, and therefore is not fully liberated.

13.3 Grinding Studies

A total of 221 variability samples from the Lac Otelnuk deposit were submitted for a grindability characterization study that involved the SPI® test and the modified Bond ball mill grindability (Mod Bond) test. Standard Bond ball-mill grindability test and static pressure tests (SPT) were conducted on a limited amount of samples (up to 20).

The main grindability statistics are summarized in Table 13.2, which assumes an equal weight for each sample.

Table 13.2 – Grindability Test Statistics

Statistics	CEET Ci	SPI® (Min)	BWI kWh/t	Mod Bond kWh/t	SPT (kWh/t) Eenergy	SPT (kWh/t) HPi
Number of Test	221	221	17	216	20	20
Average	**1.6**	**120.5**	**15.4**	**15.3**	**1.67**	**16.4**
Std. Dev.	1.4	25.3	0.7	1.0	0.10	1.0
Rel. Std. Dev.	88	21	4	6	6	6
Minimum	0.2	69.9	13.8	12.5	1.46	14.6
10th Percentile	0.7	93.1	14.5	14.0	1.58	15.4
25th Percentile	1.0	104.4	15.2	14.6	1.59	15.9
Median	**1.4**	**116.9**	**15.6**	**15.3**	**1.68**	**16.3**
75th Percentile	2.0	131.8	15.8	15.9	1.72	16.8
90th Percentile	2.5	148.9	16.1	16.4	1.80	18.0
Maximum	18.8	228.7	16.3	18.1	1.85	18.4

The samples tested were primarily classified as moderately hard to hard in terms of the SPI® test, averaging 120.5 minutes. Standard Bond ball-mill grindability tests and calibrated Mod Bonds, using a closing screen of 270 mesh (53 microns), classified most samples as medium to moderately hard and averaged 15.3 kWh/t (Mod Bond only). The SPT locked-cycle high-pressure indices (HPi) ranged from 14.6 kWh/t to 18.4 kWh/t, and averaged 16.4 kWh/t.

Table 13.3 – Overall Lac Otelnuk SPI and BWI Test Statistics

Statistics	SPI (minutes)						BWI (kWh/t)[3]					
Unit	2A	2B[1]	2C	3A	3B	Overall[2]	2A	2B1	2C	3A	3B	Overall[2]
Results Available	31	113	46	45	41	276	31	107	45	45	41	269
Proportion (%)	11	30	25	11	23	100	11	30	25	11	23	100
Average	**135.2**	**129.9**	**111.0**	**117.5**	**116.8**	**121.4**	**16.4**	**16.1**	**14.9**	**14.8**	**14.7**	**15.4**
Standard Dev.	32.1	30.0	21.9	23.0	28.9	28.9	0.9	1.1	0.7	0.9	0.9	1.1
Coeff. of Var. (%):	24	23	20	20	25	24	5	7	5	6	6	7
Min	93.5	67.5	69.9	71.5	79.7	67.5	14.8	13.8	13.3	13.0	12.5	12.5
10th Percentile	105.1	98.5	83.8	97.6	87.8	90.3	15.4	14.9	14.0	13.7	13.7	14.0
25th Percentile	113.9	109.9	100.7	103.5	97.6	103.5	15.7	15.3	14.5	14.3	14.1	14.6
Median	**132.4**	**125.6**	**109.4**	**110.9**	**113.9**	**116.2**	**16.4**	**15.9**	**14.9**	**14.9**	**14.7**	**15.3**
75th Percentile	141.6	140.3	121.3	123.8	127.9	132.2	17.2	16.5	15.5	15.5	15.3	16.0
90th Percentile	186.9	172.2	133.4	148.2	148.0	159.9	17.5	17.7	15.8	16.0	15.9	16.7
Max	228.7	238.8	190.0	182.5	203.8	238.8	18.1	19.0	16.2	16.8	16.4	19.0

[1] Includes Unit 2BC
[2] Statistics of the weighed results
[3] BWI results are mostly estimated BWI obtained from Mod Bond tests

The results in Table 13.3 above showed that units 2A and 2B are harder than the other units in the deposit. Units 2C, 3A and 3B depicted similar statistics in terms of hardness variability. The overall weighed average was 121.4 minutes in terms of SPI® and

15.4 kWh/t in terms of BWI. With an overall coefficient of variability of 24 %, the hardness variability was medium with respect to SPI®, while it was very low with respect to BWI (7 %).

13.4 Metallurgical Sampling

Based on the deposit modelling and the mining plan, SGS prepared different composite samples to complete bench-scale test programs and the pilot plant program. During the first stage of the DFS, the following sample was prepared:

- One master composite:
  Prepared with the 6.35 mm product remaining from the 14 composites currently stored at SGS under project 11727-005. This composite was used to perform test work during the first stage of the study.

For the FS, the following samples were prepared:

- 10 Y composite:
  One (1) 300-kg composite representative of the first ten (10) years of operation;
- 30 Y composite:
  One (1) 300-kg composite representative of the 30 years of the mine (Project life);
- One (1) 80-t bulk sample for the pilot plant:
  in this sample, the five geological sub-units are present but they were taken from outcrops sometimes outside the pit limits and the bulk sample cannot be considered representative.

The composites for the DFS were prepared with intervals taken from bulk BQ cores available from previous drilling exploration campaigns.

A total of 42 drill cores were necessary for the 10 Y composite and 127 drill cores for the 30 Y composite.

Three complete PQ cores were selected to assess the variability in the deposit and test the process with different ore composition.

Table 13.4gives the main head grade characteristics of the samples as analyzed by SGS.

Table 13.4 – Head Assay Characteristics of the Composite

Sample Name	Head Grade (%)			SG	Bond Work Index (kWh/t)		
	Fe	SiO_2	Sat	(g/cm^3)	BWI Head sample	BWI Cobber Concentrate	BWI Rougher Concentrate
Master Composite	28.4	46.4	28.6	3.29	14.5	15.4	20.1
10 Y Composite	29.5	45.9	27.4	3.40	14.2	14.7	17.1
30 Y Composite	29.9	45.0	27.0	3.39	13.8	14.8	17.5
PQ-1363	31.0	42.2	28.5	3.38	14.9	15.5	16.7
PQ-1380	29.8	44.2	26.5	3.32	14.0	14.9	17.3
PQ-1381	29.6	44.5	26.5	3.39	14.1	14.6	20.0

SG: Specific Gravity Sat: Satmagan Value

13.5 Metallurgical Laboratory and Pilot Testing

13.5.1 Metallurgical Testing Performed during the First Stage of the Feasibility Study (FS)

During the initial stage of the FS, a laboratory test-work program on core samples was defined and completed to fill the gaps remaining from previous studies and to finalize the flow sheet selection. This program included cobbing separation, fine grinding in two stages with roughing and cleaning magnetic separation stages, de-sliming stages, and settling and filtration tests on concentrates.

Three different process routes were evaluated:

- SAG option;
- AG + pebble crushing option;
- High pressure grinding rolls (HPGR) option.

The process flow design after the coarse grinding stage was identical for the three options.

Two optimal de-sliming locations in the flow sheet were investigated:

- After the first stage of grinding (80 % passing = 105 µm), using a two-stage hydrocycloning circuit;
- After the second stage of grinding (80 % passing = 38 µm), using a de-sliming thickener.

The front end of the testing program included coarse cobbing at three sizes (-6.35 mm, -3.35 mm and -1.18 mm), followed by a secondary grinding stage to a P_{80} of 106 microns. Next, two flow-sheet alternatives were tested:

- Flow sheet 1:
  Here, the ball mill product is de-slimed and then processed through a single stage of rougher LIMS. The rougher concentrate is reground to a P_{80} of 38 microns and then submitted for three stages of cleaner LIMS;
- Flow sheet 2:
  This flow sheet is similar to flow sheet 1, except for the de-sliming stage, which is placed before the final three-stage cleaner LIMS, instead of before the rougher LIMS stage.

The final concentrate was submitted for settling and filtration evaluation.

The test work led to the following conclusions and recommendations:

1. Flow sheet 2 was recommended, with two stages of grinding separated by the following:
   - A stage of LIMS to reduce the plant CAPEX and;
   - A de-sliming stage to allow a proper elimination of fine silica before the cleaner LIMS stage and to obtain a concentrate with acceptable grade and recovery.
2. Since the SAG option was selected for the FS, it was recommended to evaluate the economics of screening at 1 mm compared to 3 mm for the cobbing stage tailings

elimination and for the global energy balance. (SAG milling requires more energy but can lead to energy savings in the fine grinding section as the weight recovery of cobbing concentrate to be ground would be reduced by about 5 %).

3. It was recommended that during the second stage of the FS the following test work be implemented:
   - A laboratory bench-scale test work on two composites and three individual PQ cores and;
   - A pilot scale test work on an 80 t bulk sample;
   - These tests would confirm Phase 1 results and assess the consequences of any variation in grade that may occur in any potential revision of the mining plan.
4. It was concluded that since the iron contained in the Fe-oxydes represented only 53 % of the total iron in the tailings (due to the presence of other Fe-bearing minerals - ankerite, siderite and minnesotaite), the potential of a scavenging process to recover additional commercial Fe concentrate from the tailings is limited. Such a process would also be technically very challenging due to the complex mineralization of the tailings and the poor liberation of the Fe-oxides which are finely disseminated.

13.5.2 Metallurgical Testing during the Second Stage of the Feasibility Study

a) Bench-Scale Test Work

The objectives of this program were the following:

- Confirm that the process selected during Phase 1 could obtain good results on the composites representative of the new mining plan;
- Confirm that the process is stable enough to treat variable ore composition through the test of single PQ drill core samples.

The beneficiation flow sheet tested included coarse cobbing at 3.35 mm, followed by grinding to 100 µm, rougher LIMS, regrinding concentrate to 40 µm followed by a de-sliming stage and a section of three-stage-cleaning LIMS. The metallurgical performances of the five tested composites and the composite from Phase 1 test work (Master composite) are summarized in Table 13.5.

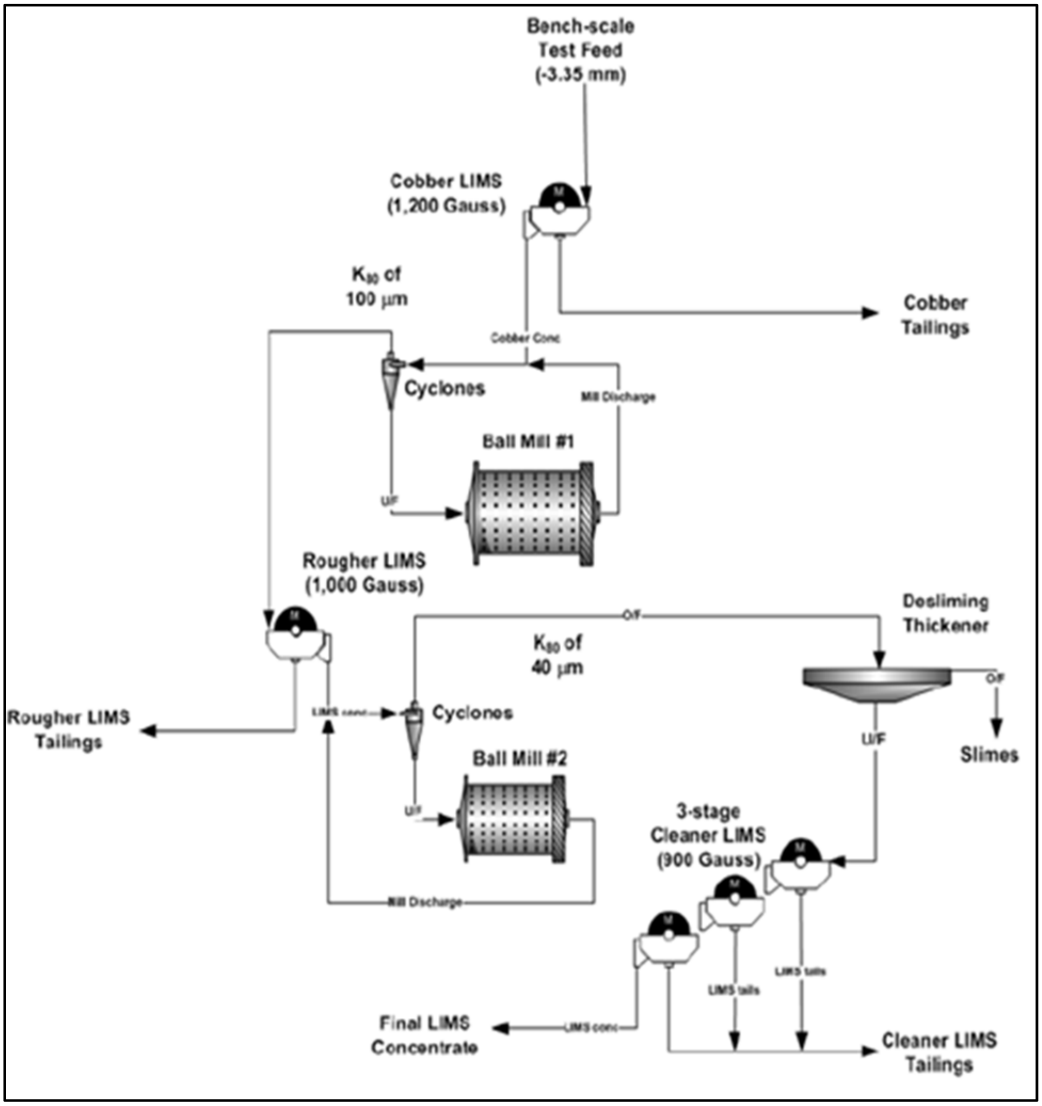
Table 13.5 – Metallurgical Results from Bench-Scale Test Work on Composites

Sample	Cobber Stage			Rougher LIMS Stage			3-stage De-Sliming			Cleaner LIMS		
	80 % passing in feed F_{80}	Recovery (%)		80% passing in rougher feed K_{80}	Recovery (%)		80 % passing in product P_{80}	Grade (%)		Recovery (%)		
	µm	Wt	Sat	µm	Wt	Sat	µm	Fe	SiO_2	Wt	Fe	Sat
10 Y Composite	2,354	73.7	98.0	117	46.4	97.5	44	68.4	3.48	27.6	63.9	92.9
30 Y Composite	2,505	71.6	97.8	119	46.6	97.3	39	69.1	3.08	27.6	64.6	95.6
PQ-1363	2,461	73.2	97.7	136	47.3	97.1	40	69.2	2.49	27.9	64.0	94.6
PQ-1380	2,400	73.4	97.5	115	46.5	96.8	43	68.6	3.44	27.4	63.3	94.8
PQ-1381	2,476	71.9	97.4	87	42.1	96.9	46	68.7	3.07	27.2	62.9	95.9
Average	2,439	72.7	97.7	115	45.8	97.1	42	68.8	3.11	27.5	63.7	94.7
Master Composite	2,188	73.6	98.1	84	47.3	97.6	40	68.9	3.64	27.5	66.0	94.9

The results among the two (2) composites, the three (3) PQ cores, as well as the master composite were generally very similar and the process proved to be robust. The final magnetite concentrate quality was very good, with a measured grading of 3.11 % SiO_2 and 68.8 % Fe on average. The average overall weight and iron recoveries were measured at 27.5 % and 63.7 % respectively, while the magnetite recovery was measured at 94.7 % on average.

The next step was to prove the flow sheet (shown in Figure 13.1) at a larger scale and on a continuous basis. A pilot plant operation was performed in SGS facilities to investigate different circuit configurations within the primary grinding circuit (SAG and AG + pebble crushing) and to allow optimization of the beneficiation circuit and data acquisition for the basic design of the industrial plant.

Figure 13.1 – Beneficiation Testing Flow-Sheet



Source: SGS

b) Pilot-Plant Test Work

Five bulk samples, representing the five main ore types from the Lac Otelnuk deposit and totaling about 80 tonnes of material, were received at the SGS Lakefield facilities to be tested in a grinding and beneficiation pilot plant. The five bulk samples were combined into a single composite for the pilot plant.

The pilot plant flow sheet included AG milling, followed by coarse cobbing and primary ball milling to about 100 microns, rougher LIMS, regrinding to about 50 microns, de-sliming, and 3-stage of cleaner LIMS. The pilot plant investigated two primary screen sizes (3.35 mm and 1.0 mm), as well as the AG mill circuit configuration (SAG with 10 % and 15 % of balls, and fully autogenous grinding (FAG) with pebble crushing).

The head characteristics of the bulk sample are presented in Table 13.6.

Table 13.6 – Bulk Sample Characteristics

Sample Name	Head Grade (%)			SG	SPI®	BWI	CEET
	Fe	SiO_2	Sat	(g/cm^3)	(Min)°	(kWh/t)	Ci
Bulk sample D SQ	29.9	44.1	24.8	3.38	139.1	16.2	10.3

SG: Specific Gravity
SPI®: SAG Power Index
BWI: Bond Ball-Mill Work Index
CEET: Comminution Economic Evaluation Tool
Ci: Crushing Index
Sat: Satmagan Value

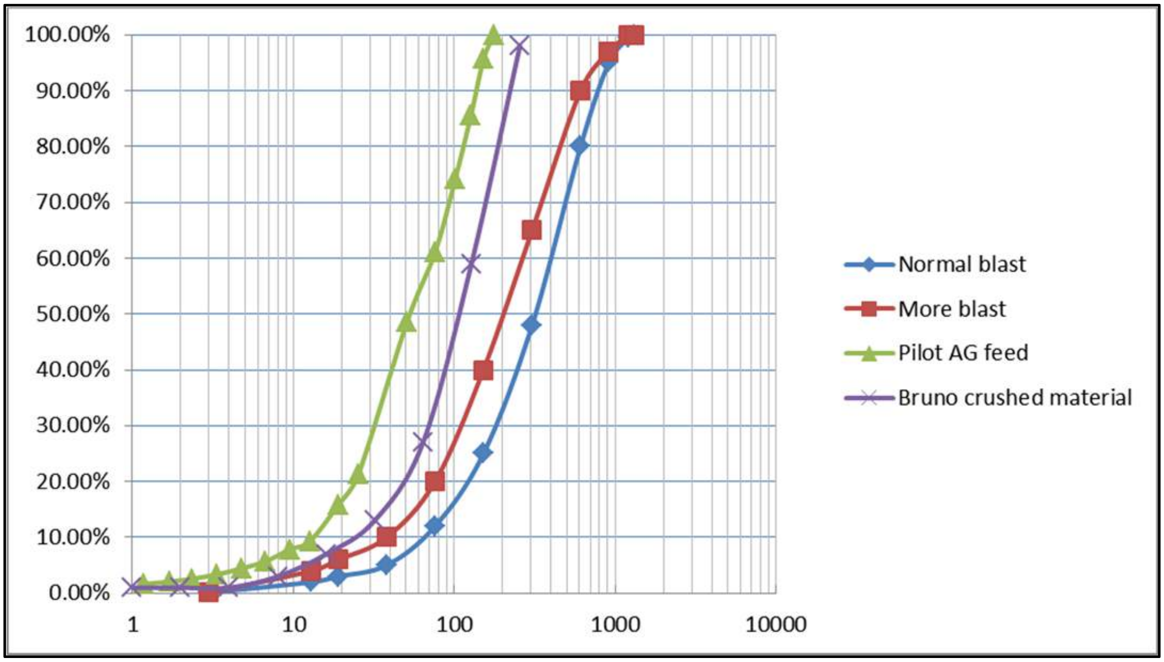
The bulk sample had a measured grade of 29.9 % Fe and 24.8 % magnetite. The weighted average iron grade was close to the iron grade measured in the previous composites tested, but the average magnetite grade was much lower (24.8 % vs. 27.5 %). Therefore, the average recoverable iron, which is a ratio of magnetic iron to total iron, was much lower from the bulk sample than from the previous composites tested (calculated at 58.5 % vs. 67 %).

The grade of MgO, CaO and MnO was quite variable amongst the samples but the weighted averages were in the same range than the previous composites tested.

The AG mill feed distribution, which had a F_{80} of 114 mm, was very sharp and did not contain a lot of fines.

Figure 13.2 below compares the particle size distribution in the pilot AG feed with a simulated ROM after blasting and crushing with an open side setting (OSS) at 200 mm. As the figure shows, the feed to the industrial plant is not expected to contain a lot of fines and the material is expected to be even coarser.

Figure 13.2 – Pilot Plant Feed Distribution and ROM Particle Size Distribution (µm) (Simulation with ®Bruno software by METSO)



The bulk sample was categorized as hard and moderately hard in terms of SAG power index (SPI®) and Bond ball-mill work index (BWI) respectively, compared to the SGS database. Davis Tube testing was performed on the five individual bulk samples.

Both products (concentrate and tailings) were submitted for whole rock analysis (WRA) assays. The Davis Tube results are summarized in Table 13.7. The silica grades measured in the Davis Tube concentrates ranged from 1.46 % to 3.17 % and averaged 1.98 %. The iron grade measured in the concentrate was fairly high, ranging from 69.3 % up to 70.8 %, averaging 69.9 %.

Table 13.7 shows that the measured weight recovery of the concentrate varied from 18.1 % to 38.6 %, averaging 23.6 %, which is consistent with the magnetite grade measured in the head samples.

Table 13.7 – Davis Tube Summary

Sample	Test	Head Grade		Concentrate Grade					Recovery		
		Fe[1]	Sat	Fe[1]	SiO_2	MgO	CaO	MnO	Wt	Fe	SiO_2
		%	%	%	%	%	%	%	%	%	%
Drum 1-2 (Zone 1)	DT-1	33.1	39.0	69.8	3.17	< 0.01	0.04	0.08	38.6	84.3	2.29
Drum 2-24 (Zone 2)	DT-2	39.5	26.9	70.8	1.46	0.12	0.12	0.13	26.9	47.7	1.16
Drum 3-46 (Zone 3)	DT-3	22.9	20.2	69.3	1.97	0.24	0.53	0.31	19.5	59.1	0.76
Drum 4-13 (Zone 4)	DT-4	22.9	19.3	70.0	1.89	0.12	0.46	0.09	18.1	55.8	0.67
Drum 5-45 (Zone 5)	DT-5	28.0	22.2	69.6	2.20	0.20	0.29	0.07	21.6	54.2	1.11
Weighted Average	-	29.8	24.1	69.9	1.98	0.16	0.30	0.15	23.6	56.3	1.08

[1] Fe grade calculated from the Fe_2O_3 WRA result

The following three types of grindability tests were performed on the bench scale master composite:

- SAG mill comminution (SMC) test:
  This abbreviated version of the standard JKTech drop-weight test was performed on the bench-scale master composite and the test results are presented in Table 13.8.

Table 13.8 – Preliminary SMC Test Results

Sample Name	A	b	A x b	Hardness Percentile	t_a[1]	DWI (kWh/m^3)	M_{ia} (kWh/t)	M_{ih} (kWh/t)	M_{ic} (kWh/t)	Relative Density
Preliminary SAG Feed	100	0.36	36.0	71	0.27	9.6	20.6	16.3	8.4	3.41

[1] The t_a value reported as part of the SMC procedure is an estimate

The sample was characterized as hard with respect to its resistance to impact (A x b).

- SPI test:
  The SAG feed sample was submitted for SPI and CEET crusher index measurements. The test results are presented in Table 13.9. The sample was categorized as hard with its SPI of 139 minutes, and it fell in the 78th percentile of the overall Lac Otelnuk results.

Table 13.9 – CEET Ci and SPI Test Results

Sample Name	SGS ID	CEET Crusher Index (Ci)	SPI (minute)	Hardness Percentile
PP-05 SAG Feed	1-13511	10.3	139.1	83

- Bond ball-mill grindability test:
  This test was performed at 120 mesh of grind (125 microns) on the master composite sample. The test results are summarized in Table 13.10. The sample was categorized as moderately hard with a BWI of 16.5 kWh/t, and fell in the 91th percentile of the Lac Otelnuk database, which comprised 239 tests.

Table 13.10 – Preliminary Bond Ball Mill Grindability Test Results

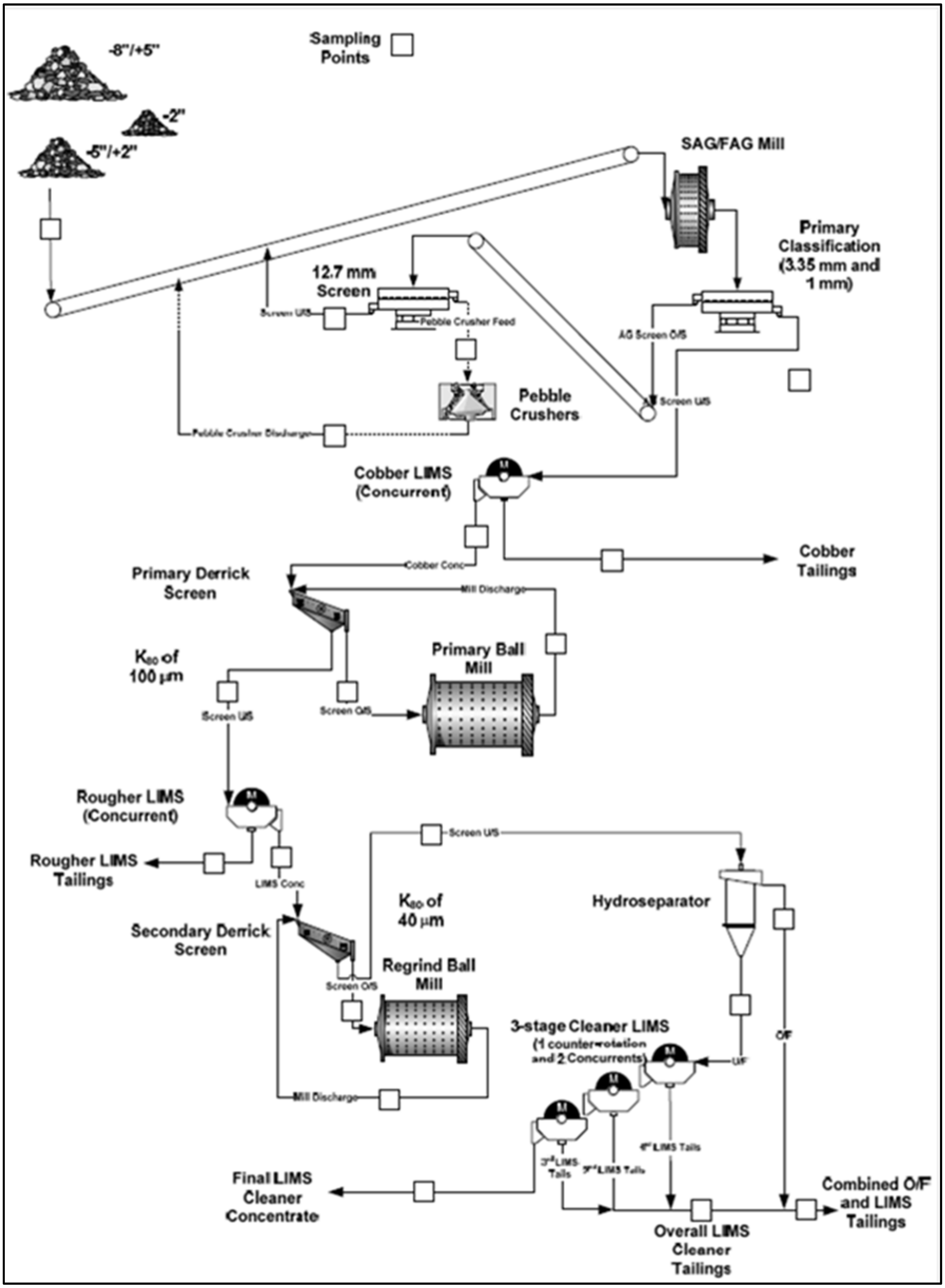
Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
Preliminary SAG Feed	120	2,638	92	1.19	16.5	73

Circuit Flow Sheet and Equipment

The simplified pilot plant flow sheet is presented in Figure 13.3. Sampling points are represented by squares. The pilot plant circuit operated as follows:

- The flow sheet included a 6' x 2' AG mill in closed circuit by a 3.35 mm or a 1.0 mm screen. In SAG mode, the oversize was returned to the mill through conveyor belts. In FAG (fully autogenous) mode, one 75-mm pebble port was open in the discharge grate to recycle the coarse pebbles to the pebble crusher circuit. The AG mill screen oversize (3.35 mm or 1.0 mm) discharged onto a 12.7 mm screen. The -12.7 mm was returned to the FAG mill, while the +12.7 mm rocks were crushed with the pebble crusher circuit prior to being returned to the FAG mill.
- The -3.35 mm (or 1.0 mm) was pumped to a cobber LIMS drum operated in concurrent mode. The cobber concentrate was screened to generate a screen undersize K_{80} of about 100 microns. The screen oversize was ground by the primary ball mill. The cobber tailings were pumped onto a 28 mesh (600 microns) scalp screen. The scalp screen oversize discharged directly on the floor, while the undersize was pumped to the tailings pond.
- The ground material (primary ball mill undersize) was pumped to a rougher LIMS drum that also operated in concurrent mode. The rougher LIMS tailings were pumped to the tailings pond, while the concentrate was screened to generate a screen undersize K_{80} of about 40 to 50 microns. The screen oversize was reground by the regrind ball mill.
- The reground material was pumped to a hydroseparator unit to de-slime the material. The hydroseparator underflow was submitted to three (3) stages of cleaner LIMS. The hydroseparator overflow was combined with the tailings from the three stages of cleaner LIMS, prior to being pumped to the tailings pond.

Figure 13.3 – Pilot Plant Flow Sheet



Source: SGS

The conditions of the pilot plant and the AG mill during the pilot plant test runs are summarized in Table 13.11 and Table 13.12 respectively.

Table 13.11 – Summary of Pilot Plant Conditions

Test Run	Circuit Configuration	AG Mill			Pebble Crusher	Primary Screen	Rougher LIMS	Regrind Screen
		Screen		Ball charge				
		mesh	mm	Vol%		µm		µm
PP-01	SAB-3.35mm	6	3.35	10	No	180	Yes	53
PP-02	SAB-1 mm	16	1.0	10	No	180	Yes	53&75
PP-03	SAB-1 mm	16	1.0	10	No	212	Yes	53&75
PP-04	SAB-3.35 mm	6	3.35	10	No	212	Yes	53&75
PP-05	SAB-3.35 mm	6	3.35	15	No	212	Yes	53&75
PP-06	SAB-3.35 mm	6	3.35	15	No	212	No	75
PP-07	SAB-1 mm	16	1.0	15	No	212	No	75
PP-08	FABC-1 mm	16	1.0	0	Yes	212	No	75
PP-09	FABC-3.35 mm	6	3.35	0	Yes	212	Yes	75

Table 13.12 – Summary of AG Mill Conditions

Test Run	Circuit Configuration	Mill Speed	Feed	Recycle	Pebble Recycle	Mill Load	AG Power	T_{80} Transfer	AG Density
		%	Dry						
		CS	kg/h	%	%	%	kWh/t	µm	%solid
PP-01	SAB-3.35mm	-	-	-	-	-	-	-	-
PP-02	SAB-1 mm	79.7	715	40.3	-	29.3	14.6	271	72.9
PP-03	SAB-1 mm	79.3	717	45.7	-	32.1	14.5	253	76.2
PP-04	SAB-3.35 mm	79.5	819	32.7	-	30.3	12.4	764	69.4
PP-05	SAB-3.35 mm	78.7	908	28.6	-	29.7	12.4	744	69.2
PP-06	SAB-3.35 mm	78.8	943	24.1	-	35.8	12.1	626	68.1
PP-07	SAB-1 mm	78.9	783	55.6	-	30.0	14.3	354	69.7
PP-08	FABC-1 mm	75.5	595	189	50.7	22.0	9.4	-	-
PP-09	FABC-3.35 mm	75.3	739	61.7	38.1	31.3	9.2	1075	76.2
Average for 1 mm screen				47.2		30.4	14.5	293	72.9
Average for 3.35 mm screen				30.6		30.0	12.4	754	69.3

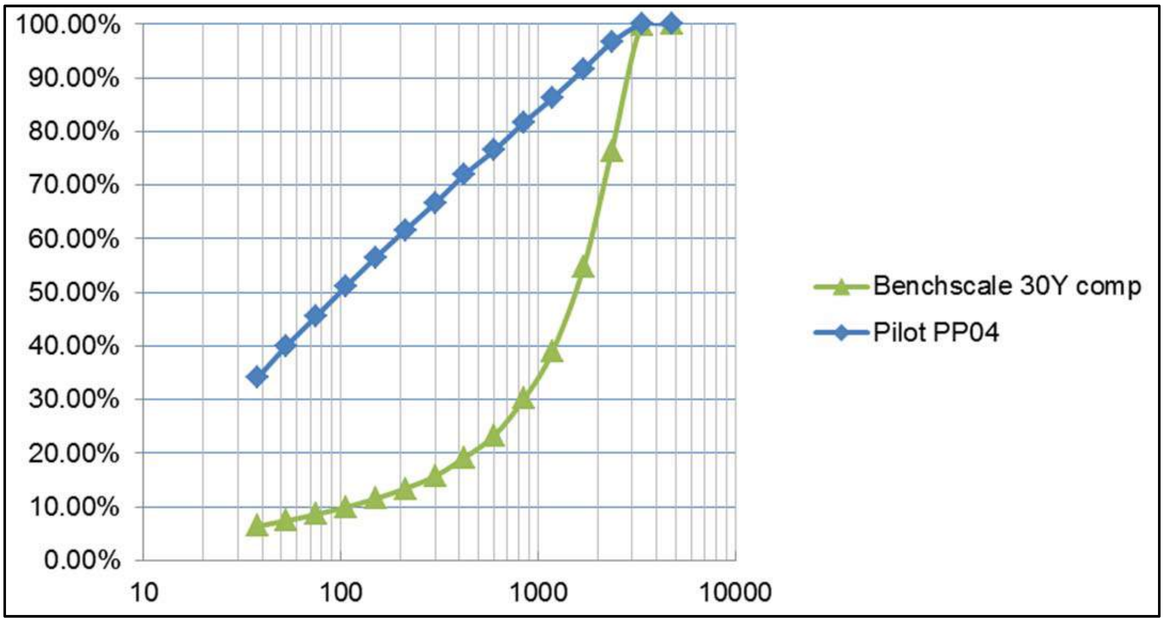
SAB and FABC: names of configuration circuits

As shown in Table 13.12 – Summary of AG Mill Conditions, the primary screen size had a significant impact on the SAG mill performance, whereas the impact of the ball charge in the SAG mill was marginal. The net power requirement of the SAG mill was higher at 1.0 mm (14.5 kWh/t) than at 3.35 mm (12.4 kWh/t). The circulation load was also much higher at 1.0 mm (47.2 %) than at 3.35 mm (30.6 %).

The particle size distribution of the AG screen undersize was much finer than the 3.35 mm bench-scale material used in previous Lac Otelnuk beneficiation-testing projects.

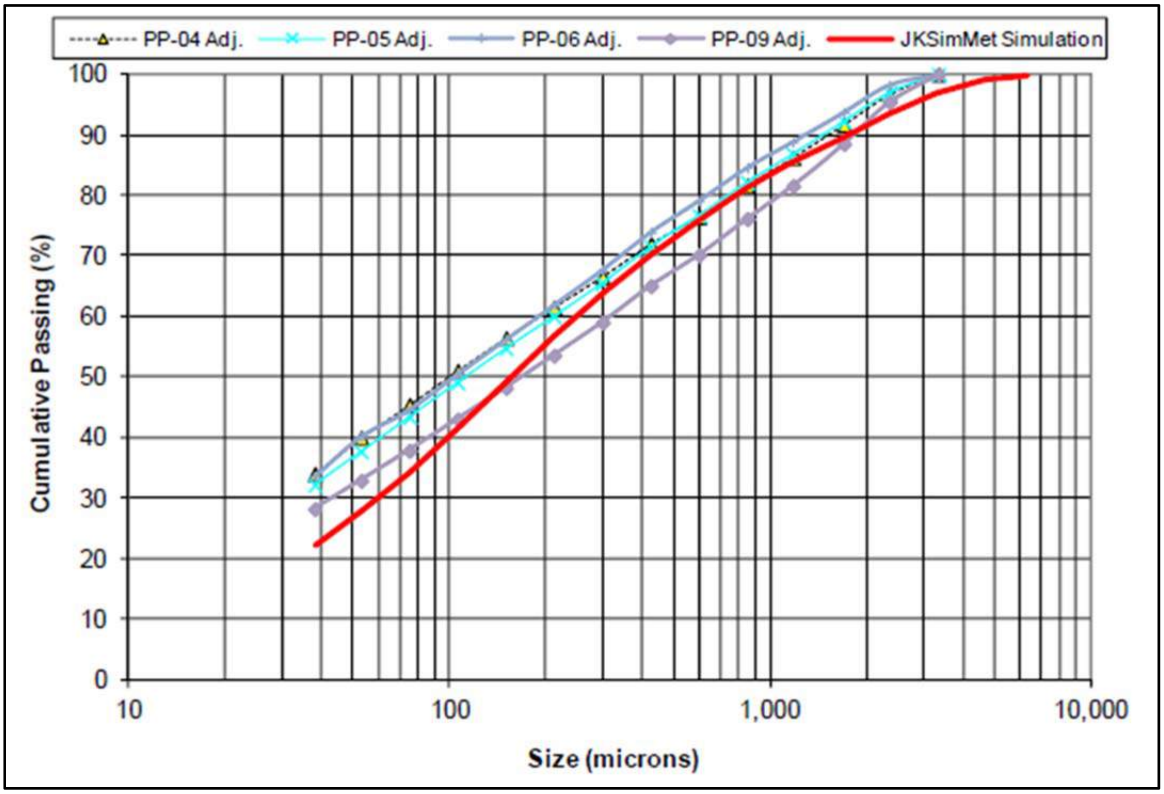
As shown in Table 13.12, the average transfer sizes (T_{80}) were approximately 290 microns and 750 microns for the 1.0 mm and 3.35 mm screens, respectively, with associated cobber weight recoveries of about 42 % and 47 % respectively. These recoveries are much higher than the bench-scale weight recovery of 28.8 % and this is mainly due to the difference of the particle size distribution of the cobber feed as shown in Figure 13.4.

Figure 13.4 – Distribution of Cobber Feed Particle Size (µm)



A computer simulation of the grinding process was made using JKSimMet software. The simulation was done using a 40' x 24' (effective grinding length - EGL) SAG mill, filled with 10 % balls and closed with a 3.35 mm screen. The simulation used the feed size distribution measured in the pilot plant, along with the sample hardness also measured on the pilot plant bulk sample. The simulation was done using the default JKSimMet SAG mill model, typically used for greenfield designs. The simulated T_{80} was 780 µm, which is close to what was measured in the pilot plant. A comparison between the simulated screen undersize and the -3.35 mm products measured during the pilot plant is presented in Figure 13.5.

Figure 13.5 – Simulated and Tested – 3.35 mm Products



Source: SGS

Of note is that the FAG mill power requirement was low at 9.2 kWh/t, even though the power requirement from the pebble crusher is not included in this value. The amount of work done by the pebble crusher was much higher than typical due to a very high pebble recycle of 38.1 % (from a total recycle of 61.7 %). Therefore this FAG power requirement should be used with care. For the primary ball mill, a ball charge of 20 % was used for most of the pilot plant testing. An attempt was made to reduce the ball charge but the very low ball charge used (12 %) resulted in a very inefficient grinding.

As shown in Table 13.13 below, the measured net specific power consumption, based on the primary circuit feed rate (cobber concentrate), ranged from 10.6 kWh/t up to 16.3 kWh/t. The measured grind size from the primary ball mill circuit ranged from 84 to 109 microns.

Table 13.13 – Summary of Primary Ball Mill

Test Run	Circuit Configuration	Mill Speed	Cobber Concentrate		CL	Ball Charge	F_{80}	P_{80}	Net Power	Wio	Mill Density
		%	Dry	w/w		Vol					
		CS	kg/h	%	%	%	µm	µm	kWh/t	kWh/t	%solid
PP-01	SAB-3.35 mm	-	-	-	-	20	-	-	-	-	-
PP-02	SAB-1 mm	83.4	298	41.8	46.1	20	353	84	15.9	28.5	46.4
PP-03	SAB-1 mm	83.2	287	40.0	42.5	20	337	87	16.3	31	26.9
PP-04	SAB-3.35 mm	78.5	395	48.3	69.1	20	1061	96	12.4	17.4	44.1
PP-05	SAB-3.35 mm	80.4	423	46.6	72.5	20	1030	100	11.1	16.2	45.1
PP-06	SAB-3.35 mm	81.3	440	46.7	71.8	20	826	101	10.6	16.4	43.4
PP-07	SAB-1 mm	80.7	342	43.8	56.0	20	420	97	13.7	26	31.5
PP-08	FABC-1 mm	80.1	272	45.6	81.7	12	-	107	9.1	-	-
PP-09	FABC-3.35 mm	83.7	348	47.1	84.8	18.1	1408	109	11.3	16.3	56
Average for 1 mm screen			309	41.9	48.2	20	370	89	15.3	28.5	34.9
Average for 3.35 mm screen			409	47.4	70.8	20	1046	98	11.8	16.8	44.6

CL: circulating load Wio: operating work index CS: critical speed

For the regrind ball mill, a ball charge of 17 % was used for most of the pilot plant testing. The measured net specific power consumption, based on the regrind ball mill feed rate (rougher concentrate in most of the cases), ranged from 10.2 kWh/t to 13.4 kWh/t.

The regrind ball-mill conditions are summarized in Table 13.14.

Table 13.14 – Summary of Regrind Ball Mill

Test Run	Circuit Configuration	Mill Speed	Rougher Concentrate		CL	Ball Charge	F_{80}	P_{80}	Net Power	Wio	Mill Density
		%	Dry	w/w		Vol					
		CS	kg/h	%	%	%	µm	µm	kWh/t	kWh/t	%solid
PP-01	SAB-3.35 mm	-	-	-	-	17.0	-	-	-	-	-
PP-02	SAB-1 mm	82.4	202	28.3	112	17.0	96	47	13.2	30.1	34.6
PP-03	SAB-1 mm	77.2	199	27.7	90	17.0	97	47	13.8	31.2	26.7
PP-04	SAB-3.35 mm	76.5	242	29.6	203	17.0	107	48	10	20.8	41.1
PP-05	SAB-3.35 mm	75.8	238	26.2	57	17.0	102	51	10.4	25.6	22.1
PP-06	SAB-3.35 mm	-	-	-	-	17.0	-	-	-	-	-
PP-07	SAB-1 mm	75.5	342	-	62	17.0	97	50	7.2	18.0	24.1
PP-08	FABC-1 mm	76.8	272	-	92	9.0	107	56	5.8	15.6	-
PP-09	FABC-3.35 mm	76.9	203	27.5	196	9.0	119	63	7.4	21.1	33.6
Average for 1 mm screen			248	28.0	88	17.0	97	48	11.3	26.4	28.5
Average for 3.35 mm screen			240	37.9	130	17.0	104	49	10.2	23.2	31.5

CL: Circulating load
Wio: Operating Work Index

Table 13.15 below summarizes the cobber assays, first in terms of Cobber LIMS reconciled assays and then in terms of Cobber LIMS overall distribution.

Magnetite recovery from the cobber concentrate was slightly higher (0.4 % on average) at 1.0 mm than 3.35 mm due to the lower losses of coarse middling particles in the cobber tailings. The magnetite recovery ranged from 97.6 % to 97.9 %, averaging 97.9 % at 1.0 mm, and varied from 97.0 % to 97.6 %, averaging 97.3 % at 3.35 mm. Conversely, the iron recovery was slightly lower at 1.0 mm, averaging 68.7 % compared to 71.6 % at 3.35 mm due to the higher amount of hematite liberated at 1.0 mm.

The cobber concentrate was generally about 30 % coarser than the transfer size (T_{80}) and almost twice coarser than the cobber tailings. The cobber tailings were generally very fine. For the test trials with the 3.35 mm screen, the cobber tailings contained typically less than 10 % of +1.0 mm material.

Table 13.15 – Summary of Cobber Assays

Test Run	Circuit Configuration	Head		Cobber LIMS Reconciled Assays (%)						
				Concentrate				Tailings		
		Fe	Sat	Weight	Fe	SiO_2	Sat	Weight	Fe	Sat
PP-01	SAB-3.35mm	-	-	-	-	-	-	-	-	-
PP-02	SAB-1 mm	29.9	26	41.8	49.6	23.8	60.9	58.2	15.8	0.9
PP-03	SAB-1 mm	29.6	24	40.0	49.6	24.1	58.5	60.0	16.2	1.0
PP-04	SAB-3.35 mm	30.2	25.5	48.3	45.2	28.1	51.5	51.7	16.1	1.2
PP-05	SAB-3.35 mm	28.7	23.9	46.6	43.3	28.9	49.8	53.4	16.0	1.2
PP-06	SAB-3.35 mm	29.8	25.1	46.7	46.0	27.0	52.2	53.3	15.7	1.4
PP-07	SAB-1 mm	30.2	25	43.8	48.3	25.0	55.7	56.2	16.2	1.1
PP-08	FABC-1 mm	-	-	-	-	-	-	-	-	-
PP-09	FABC -3.35 mm	28.9	23.5	47.1	43.9	30.3	48.6	52.9	15.4	1.1

Test Run	Circuit Configuration	Head		Cobber LIMS Overall Distribution (%)						
				Concentrate				Tailings		
		Fe	Sat	Weight	Fe	SiO_2	Sat	Weight	Fe	Sat
PP-01	SAB-3.35mm	100	100	-	-	-	-	-	-	-
PP-02	SAB-1 mm	100	100	41.8	69.2	22.2	97.9	58.2	30.8	0.6
PP-03	SAB-1 mm	100	100	40.0	67.1	21.3	97.6	60.0	32.9	0.4
PP-04	SAB-3.35 mm	100	100	48.3	72.4	30.9	97.6	51.7	27.6	0.7
PP-05	SAB-3.35 mm	100	100	46.6	70.1	30.3	97.2	53.4	29.9	0.9
PP-06	SAB-3.35 mm	100	100	46.7	71.9	28.8	97.0	53.3	28.1	-
PP-07	SAB-1 mm	100	100	43.8	69.9	25.1	97.6	56.2	30.1	-
PP-08	FABC-1 mm	-	-	-	-	-	-	-	-	-
PP-09	FABC -3.35 mm	100	100	47.1	71.7	31.3	97.4	52.9	28.3	1.0

Table 13.16 below summarizes the rougher assays.

The overall weight recovery of the rougher concentrate varied from 26.2 % to 29.6 %, for a feed grind size varying from 84 to 109 microns. The rougher concentrate from the pilot plant tests measured a grade of between 7 % and 9 % SiO_2 compared to 23 % to 28 % SiO_2 in the bench-scale programs.

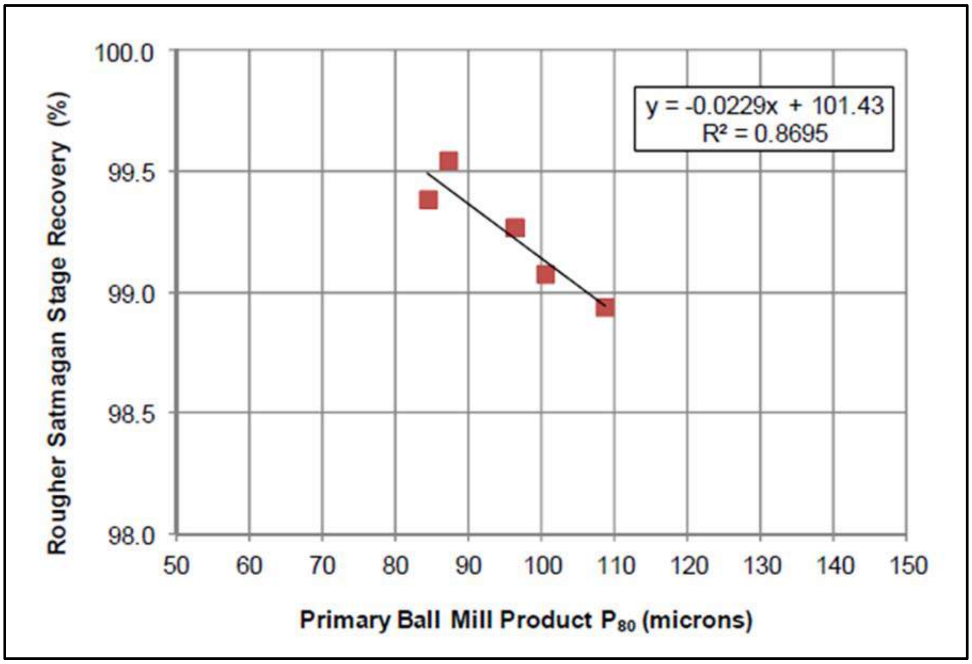
Table 13.16 – Summary of Rougher Assays

Test Run	Circuit Configuration	Head		Rougher LIMS Reconciled Assays (%)						
				Concentrate				Tailings		
		Fe	Sat	Weight	Fe	SiO_2	Sat	Weight	Fe	Sat
PP-01	SAB-3.35mm	-	-	-	-	-	-	-	-	-
PP-02	SAB-1 mm	49.6	60.9	28.3	65.2	6.96	89.5	13.5	16.9	1.2
PP-03	SAB-1 mm	49.6	58.5	27.7	64.0	7.98	84.2	12.3	17.0	0.9
PP-04	SAB-3.35 mm	45.2	51.5	29.6	63.3	9.10	83.4	18.7	16.6	1.0
PP-05	SAB-3.35 mm	43.3	49.8	26.2	64.5	8.39	87.9	20.4	16.0	1.0
PP-06	SAB-3.35 mm	46.0	52.2	-	-	-	-	-	-	-
PP-07	SAB-1 mm	48.3	55.7	-	-	-	-	-	-	-
PP-08	FABC-1 mm	-	-	-	-	-	-	-	-	-
PP-09	FABC -3.35 mm	43.9	48.6	27.5	63.5	9.36	82.2	19.6	16.4	1.2

Test Run	Circuit Configuration	Head		Rougher LIMS Overall Distribution (%)						
				Concentrate				Tailings		
		Fe	Sat	Weight	Fe	SiO_2	Sat	Weight	Fe	Sat
PP-01	SAB-3.35mm	-	-	-	-	-	-	-	-	-
PP-02	SAB-1 mm	69.2	97.9	28.3	61.5	4.4	97.3	13.5	7.6	0.6
PP-03	SAB-1 mm	67.1	97.6	27.7	60.0	4.9	97.2	12.3	7.1	0.4
PP-04	SAB-3.35 mm	72.4	97.6	29.6	62.1	6.1	96.9	18.7	10.3	0.7
PP-05	SAB-3.35 mm	70.1	97.2	26.2	58.8	4.9	96.3	20.4	11.4	0.9
PP-06	SAB-3.35 mm	71.9	97.0	-	-	-	-	-	-	-
PP-07	SAB-1 mm	69.9	97.6	-	-	-	-	-	-	-
PP-08	FABC-1 mm	-	-	-	-	-	-	-	-	-
PP-09	FABC -3.35 mm	71.7	97.4	27.5	60.6	5.7	96.4	19.6	11.1	1.0

Unlike in the cobber stage, the weight recovery of the rougher stage was not only dependent on the feed size (primary ball mill product) but more related to the magnetite grade of the cobber concentrate. The magnetite recovery was inversely proportional to the primary grind size as shown in Figure 13.6.

Figure 13.6 – Effect of Grind Size on Rougher Stage Magnetite Recovery



Source: SGS

For most of the pilot plant testing, a hydroseparator was used to de-slime the ground rougher LIMS concentrate. Because the rougher LIMS concentrate was already relatively clean, the amount of slimes generated during the regrind process was very low and the hydroseparator could not be used to its full potential.

As shown in the summary of the hydroseparator assay in Table 13.17, the weight rejected to the overflow was typically low, ranging from 1.0 % to 2.7 %, corresponding to a stage rejection of 3.6 % to 9.3 % of the hydroseparator feed weight. The magnetite stage recovery was very good (> 99.5 %), with the exception of run PP-04.

The silica grade at the hydroseparator underflow varied from 3.68 % to 7.55 % SiO_2 and was much higher during PP-07 when the rougher LIMS was bypassed, at 13.4 % SiO_2.

For most of the runs, the hydroseparator feed percent of solids ranged from 18.9 % to 32.9 %, and the rising water rate varied from 2.4 $m^3/h/m^2$ to 6.3 $m^3/h/m^2$.

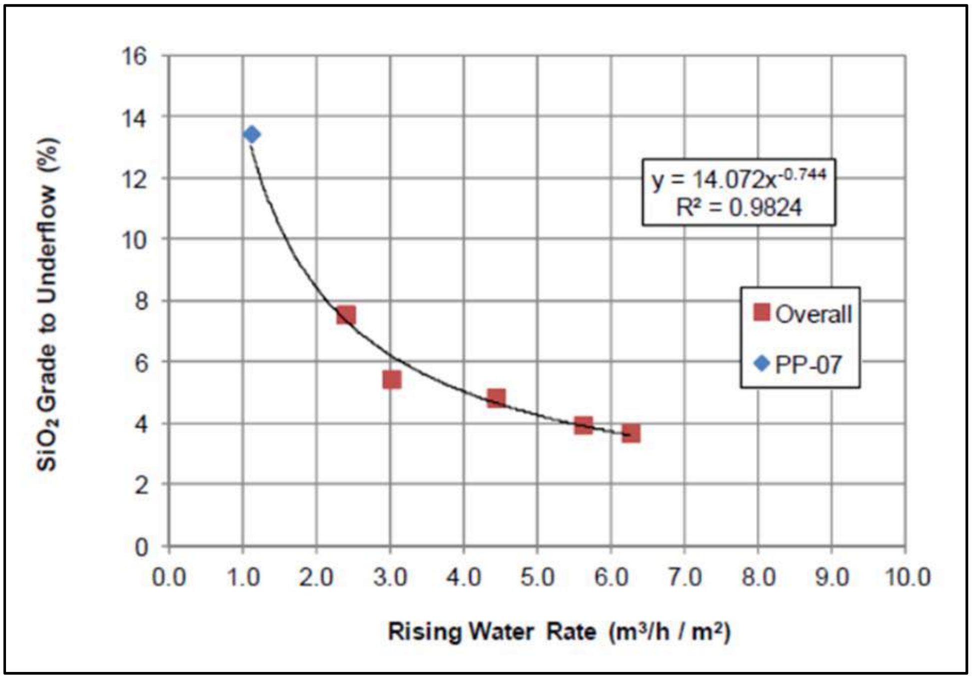
Table 13.17 – Summary of Hydroseparator Assay

Test Run	Circuit Configuration	Head Rough Conc.		Hydroseparator Reconciled Assays (%)						
				Underflow				Overflow		
		Fe	Sat	Weight	Fe	SiO_2	Sat	Weight	Fe	Sat
PP-01	SAB-3.35mm	-	-	-	-	-	-	-	-	-
PP-02	SAB-1 mm	65.2	89.5	26.5	68.3	3.68	95.1	1.8	17.5	4.5
PP-03	SAB-1 mm	64.0	84.2	25.5	67.9	3.94	91.0	2.2	19.0	5.1
PP-04	SAB-3.35 mm	63.3	83.4	26.8	67.3	4.83	90.0	2.7	24.9	19.3
PP-05	SAB-3.35 mm	64.5	87.9	24.6	67.5	5.44	93.1	1.5	17.4	3.4
PP-06	SAB-3.35 mm	-	-	-	-	-	-	-	-	-
PP-07	SAB-1 mm	-	-	32.4	59.8	13.4	74.7	11.4	15.6	1.7
PP-08	FABC-1 mm	-	-	-	-	-	-	-	-	-
PP-09	FABC -3.35 mm	63.5	82.2	26.5	65.3	7.55	85.2	1.0	14.9	1.2

Test Run	Circuit Configuration	Head Rough Conc.		Hydroseparator Overall Distribution (%)						
				Underflow				Overflow		
		Fe	Sat	Weight	Fe	SiO_2	Sat	Weight	Fe	Sat
PP-01	SAB-3.35mm	-	-	26.5	60.5	2.2	97.0	-	-	-
PP-02	SAB-1 mm	61.5	97.3	25.5	58.6	2.2	96.7	1.8	1.0	0.3
PP-03	SAB-1 mm	60.0	97.2	26.8	59.9	3.0	94.8	2.2	1.4	0.5
PP-04	SAB-3.35 mm	62.1	96.9	24.6	57.8	3.0	96.1	2.7	2.3	2.1
PP-05	SAB-3.35 mm	58.8	96.3	-	-	-	-	1.5	0.9	0.2
PP-06	SAB-3.35 mm	-	-	32.4	64.1	10.0	96.8	-	-	-
PP-07	SAB-1 mm	-	-	-	-	-	-	11.4	5.9	0.8
PP-08	FABC-1 mm	-	-	26.5	60.1	4.4	96.3	-	-	-
PP-09	FABC -3.35 mm	60.6	96.4	26.5	60.5	2.2	97.0	1.0	0.5	0.1

The relationship between the rising water rate and the grade of silica measured at the hydroseparator underflow is shown in Figure 13.7.

Figure 13.7 – Effect of Rising Water on Silica Grade in Hydroseparator Underflow



Source: SGS

As shown in the summary of the cleaner assay in Table 13.18, the silica grade in the final cleaner LIMS concentrate varied from 2.47 % to 4.05 %, with a regrinding mill circuit product size (P_{80}) ranging from 47 µm to 63 µm. The corresponding final concentrate grind size was found to be in the range of 48 µm to 64 µm. The relationship between the size and the silica grade is presented in Figure 13.8.

The overall weight recovery was in the range of 23.3 % to 25.9 %, averaging 24.9 %, while the overall magnetite recovery varied from 94.7 % to 96.9 %, averaging 96.1 %. The overall iron recovery averaged 58.3 %, which is very close to the calculated recoverable iron of 58.5 %.

As for the hydroseparator, the stage weight recovery was typically high when the rougher LIMS was in operation.

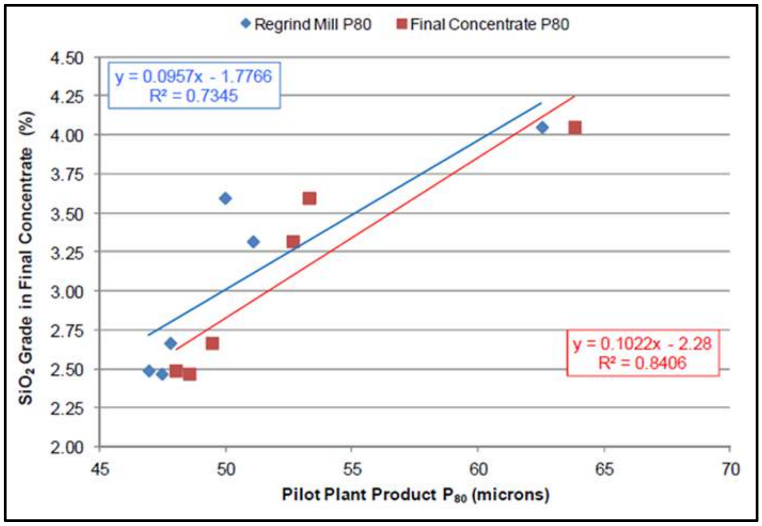
Table 13.18 – Summary of Cleaner Assay

Test Run	Circuit Configuration	Head Hydr. U/F		Cleaner LIMS Reconciled Assays (%)							Final Conc.
				Concentrate				Tailings			P_{80}
		Fe	Sat	Weight	Fe	SiO_2	Sat	Weight	Fe	Sat	µm
PP-01	SAB-3.35mm	-	-	-	-	-	-	-	-	-	-
PP-02	SAB-1 mm	68.3	95.1	25.7	69.6	2.49	98.2	0.9	28.3	3.4	48
PP-03	SAB-1 mm	67.9	91.0	24.4	69.6	2.47	94.8	1.1	30.1	4.5	49
PP-04	SAB-3.35 mm	67.3	90.0	25.2	69.8	2.67	95.6	1.6	27.9	2.5	49
PP-05	SAB-3.35 mm	67.5	93.1	23.3	69.8	3.32	98.1	1.3	25.8	2.1	53
PP-06	SAB-3.35 mm	-	-	-	-	-	-	-	-	-	-
PP-07	SAB-1 mm	59.8	74.7	25.9	69.5	3.60	93.3	6.5	21.1	1.0	53
PP-08	FABC-1 mm	-	-	-	-	-	-	-	-	-	-
PP-09	FABC -3.35 mm	65.3	85.2	24.3	69.2	4.05	92.8	2.2	22.9	1.7	64

Test Run	Circuit Configuration	Head Hydr. U/F		Cleaner LIMS Overall Distribution (%)						
				Concentrate				Tailings		
		Fe	Sat	Weight	Fe	SiO_2	Sat	Weight	Fe	Sat
PP-01	SAB-3.35mm	60.5	97.0	-	-	-	-	-	-	-
PP-02	SAB-1 mm	58.6	96.7	25.7	59.7	1.4	96.9	0.9	0.8	0.1
PP-03	SAB-1 mm	59.9	94.8	24.4	57.5	1.3	96.5	1.1	1.1	0.2
PP-04	SAB-3.35 mm	57.8	96.1	25.2	58.4	1.5	94.7	1.6	1.5	0.2
PP-05	SAB-3.35 mm	-	-	23.3	56.7	1.7	96.0	1.3	1.2	0.1
PP-06	SAB-3.35 mm	64.1	96.8	-	-	-	-	-	-	-
PP-07	SAB-1 mm	-	-	25.9	59.5	2.1	96.5	6.5	4.5	0.3
PP-08	FABC-1 mm	60.1	96.3	-	-	-	-	-	-	-
PP-09	FABC -3.35 mm	60.5	97.0	24.3	58.3	2.2	96.2	2.2	1.8	0.2

Hydr: Hydroseparator

Figure 13.8 – Size vs. Silica Grade in Cleaner LIMS Concentrate



Source:SGS

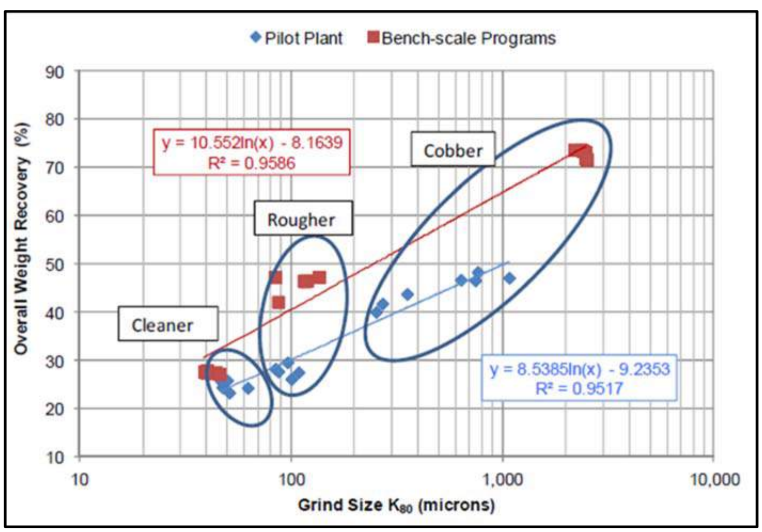
c) Pilot Plant vs. Bench-Scale Metallurgical Results

The pilot plant metallurgical results were not in agreement with the bench-scale test results.

The test results differed in three main aspects:

- First, for the same grind size (K_{80}), the overall weight recovery was higher in the bench-scale tests. This is shown in Figure 13.9, where the blue curve connects all the points from the pilot plant and the red curve corresponds to all the bench-scale tests conducted on six different composites. It can be extrapolated from the curves that for a T_{80} of 750 µm in a sample behaving as the previously tested bench-scale samples, a cobber concentrate weight recovery of about 62 % could be expected.

Figure 13.9 – Grind Size vs. Overall Weight Recovery



Source: SGS

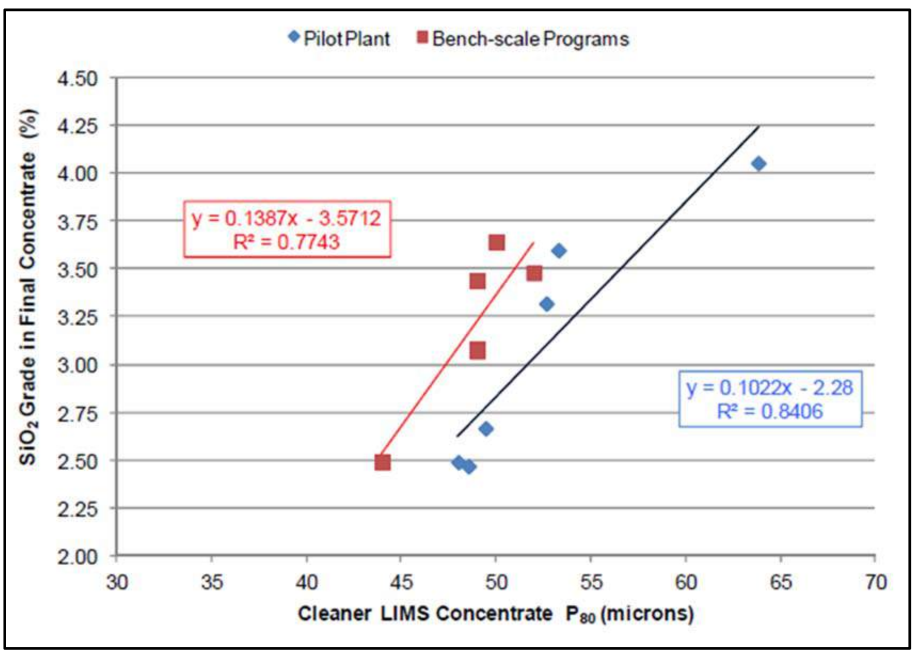
- Second, the final grind size of the pilot plant and bench-scale results was different, as presented in Table 13.19. In the pilot plant, an average grind size achieved by the regrind mill was 51 microns, the P_{80} of the final concentrate was 53 microns (corresponding to a silica grade of 3.10 %), and the average amount of silica in the regrind product (rougher concentrate) was 11.1 %. During the bench-scale testing programs, a finer grind (42 microns) was required on average to generate a final concentrate grade of 3.29 % SiO_2. Because of the higher SiO_2 grade in the bench-scale rougher concentrate, the bench-scale samples required a much finer regrind product P_{80} than the pilot plant bulk sample to generate a final concentrate with a P_{80} = 49-53 microns.

Table 13.19 – Pilot Plant and Bench Scale Final Grind Comparison

Type of Test	Average P_{80} (µm)			Average SiO_2 Grade (%)	
	Regrind Product	Final Conc.	Diff.	Regrind Product	Final Conc.
Pilot Plant	51	53	1.7	11.1	3.10
Bench-scale	42	49	6.8	25.4	3.29

- Third, when the silica grade is plotted against the P_{80} of the final cleaner LIMS concentrate, the trends for the pilot plant and bench-scale tests are parallel, but slightly shifted. This is shown in Figure 13.10.

Figure 13.10 – Pilot Plant and Bench-Scale Grind vs. Silica Grade in Cleaner LIMS Concentrate



Source: SGS

To better explain the differences between the pilot plant results and the bench scale results, SGS performed a Davis Tube characterization of the pilot-plant bulk sample and the 30 Y composite. The objective was to confirm whether the 30-year composite, prepared as part of the bench-scale testing, and the bulk composite used during the pilot test work would behave similarly with respect to Davis Tube performance at different grind sizes: (P_{100}) 1000, 840, 297, 125, 75 and 44 microns.

Table 13.20 presents the Davis Tube (DT) test summary from the pilot plant sample (PP-05 SAG Feed) and the 30 Y composite bench-scale. The difference of weight recovery between the two (2) composites is mostly due to the different magnetite grade in each sample.

Table 13.20 – Davis Tube Summary at P_{80} = 42 µm

Sample	Head assays		Test	P_{80}	Rec.	DT Concentrate Grade				
	Fe[1]	Sat[2]		µm	Wt	Fe[1]	SiO_2	Al_2O_3	MgO	CaO
	%	%			%	%	%	%	%	%
30Y Comp	29.6	27.0	DT-4	43	29.2	68.5	3.67	0.02	0.29	0.29
PP-05 SAG Feed	29.8	24.1	DT-9	42	24.0	69.5	2.97	0.05	0.16	0.23

[1] Fe grade calculated from the Fe_2O_3 WRA result

[2] Satmagan grade expressed as Fe_3O_4

The David Tube test results are presented in Table 13.21. The difference between the pilot plant and bench-scale cobber weight recoveries was mostly due to the feed size (F_{80}) to the cobber stage, but also due to the lower magnetite content of the pilot-plant bulk sample.

Table 13.21 – Davis Tube Test Results

Sample	Test	P_{80}	Rec.	DT Concentrate Grade				
		µm	Wt	Fe[1]	SiO_2	Al_2O_3	MgO	CaO
			%	%	%	%	%	%
30Y Comp	DT-11	751	56.1	39.7	33.6	0.15	1.93	2.20
30Y Comp	DT-1	319	46.0	47.5	25.3	0.20	1.46	1.65
30Y Comp	DT-2	168	37.2	55.8	16.7	0.07	1.03	1.17
30Y Comp	DT-3	61	30.0	66.0	6.15	0.01	0.46	0.47
30Y Comp	DT-4	43	29.2	68.5	3.67	0.02	0.29	0.29
30Y Comp	DT-5	26	27.5	69.7	2.66	0.01	0.17	0.18
PP-05 SAG Feed	DT-12	752	49.0	41.9	31.4	0.07	1.28	1.90
PP-05 SAG Feed	DT-6	496	42.8	45.1	28.0	0.08	1.15	1.81
PP-05 SAG Feed	DT-7	153	28.8	59.0	13.2	0.05	0.67	1.02
PP-05 SAG Feed	DT-8	64	24.0	67.6	4.76	0.02	0.24	0.39
PP-05 SAG Feed	DT-9	42	24.0	69.5	2.97	0.05	0.16	0.23
PP-05 SAG Feed	DT-10	26	23.1	70.4	1.84	0.02	0.09	0.16

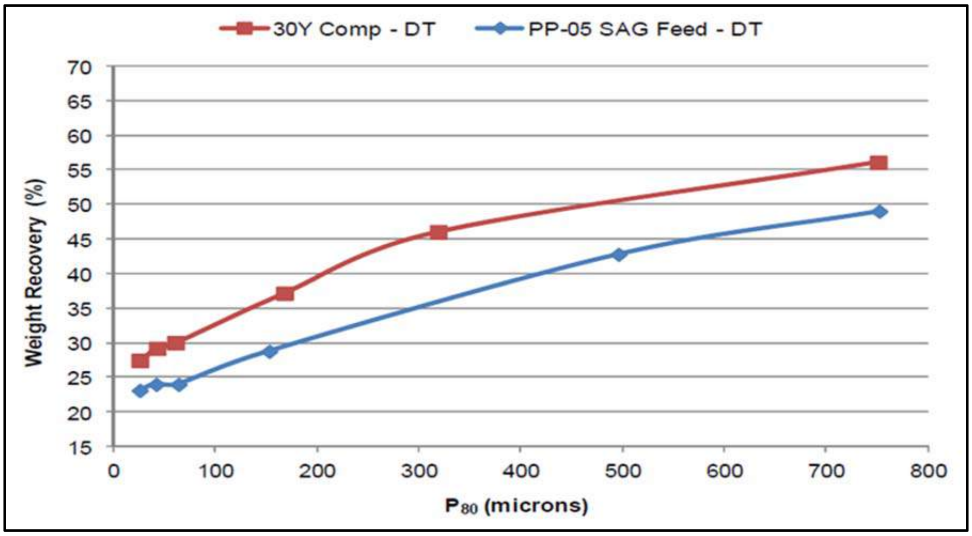
[1] Fe grade calculated from the Fe_2O_3 WRA result

The difference in the weight recovery between the pilot-plant bulk sample and the 30 Y composite ranged from 7-8 % at coarse size (750 µm) down to 4-5 % at fine size (43 µm).

The pilot-plant bulk sample generated a cleaner final concentrate than the 30 Y Comp for the same grind size.

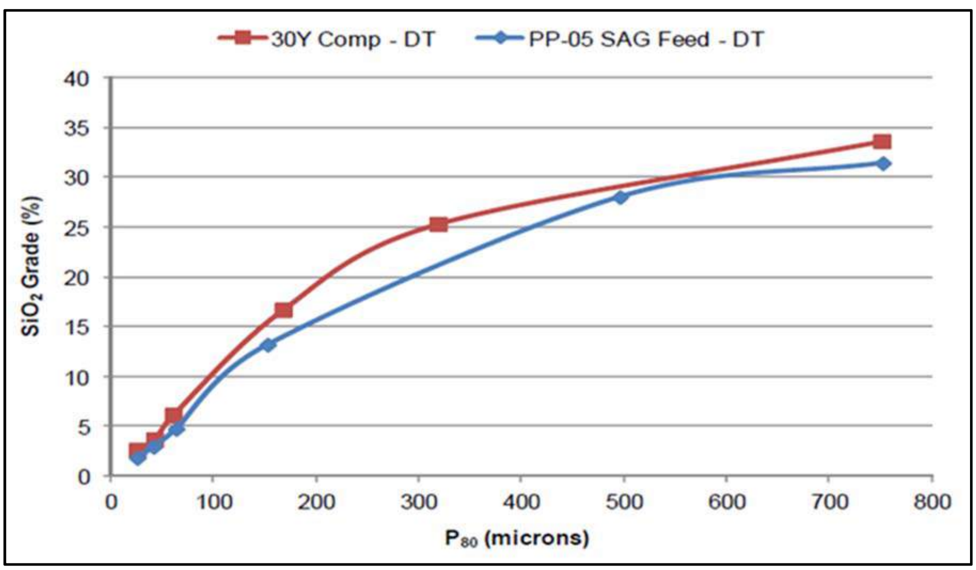
The mapping of the Davis Tube weight recovery vs. size and the Davis Tube concentrate grade vs. size is shown Figure 13.11 and Figure 13.12, respectively.

Figure 13.11 – Davis Tube Weight Recovery vs. Size



Source:SGS

Figure 13.12 – Davis Tube Concentrate SiO_2 Grade vs. Size



Source: SGS

13.6 Vendor Testing and Technology Testing

Several sub-samples were collected during the pilot plant testing to be sent to various vendors for testing or to be stored for future testing.

As part of the technological test work, the following investigations were conducted:

- Filtration using ceramic disc filters was tested to assess the filter productivity at different pulp temperature and density for the targeted concentrate moisture of 8 %;
- Bench-scale testing was conducted to evaluate filter cloth selection, filter cake thickness, filtration rate, moisture content of this cake, and cake handling characteristics. The testing results are summarized in Table 13.22.

Table 13.22 – Filtration Test Results Summary

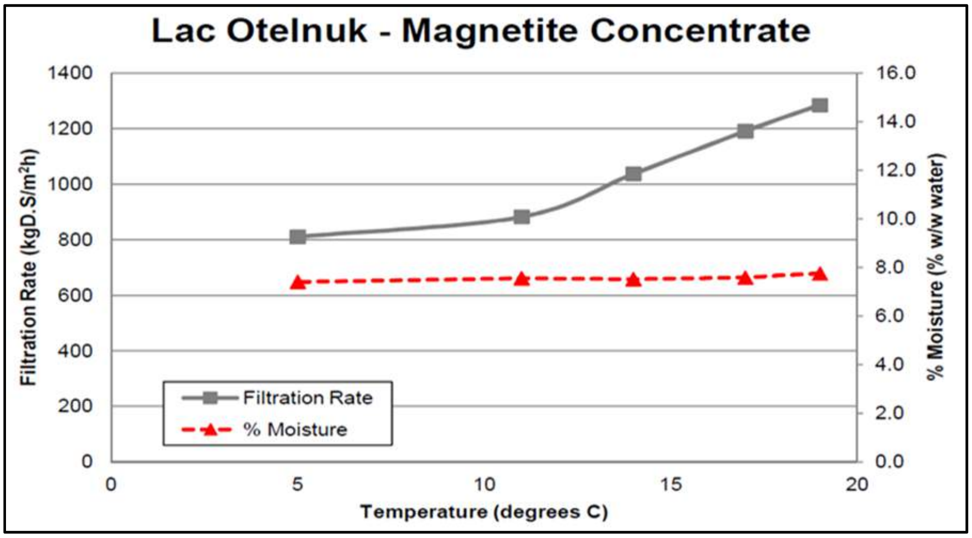
Sample - Unit	pH	Feed Solids	Feed Temperature (°C)	Cycle Time (sec)	Filtration Rate (kgDS/m²h)	Filter Cake Moisture (%w/w water)	Filter Cake Thickness (mm)
Magnetite – Ceramec Leaf Tester	8	62	9	30-65	773-1934	7.2-10.3	5-12
	8	63	5-19	30	811-1285	7.4-7.8	4-6
	8	70	8-9	30-65	1644-4342	9.4-12.9	6-20

- Tests were performed to evaluate the effects of temperature on the filtration rates. Five tests were performed at a feed concentration of 63 % w/w solids, using a range of slurry temperatures and a fixed cycle time. The testing results are given in Table 13.23 and plotted in Figure 13.13.

Table 13.23 – Effect of Temperature on Filtration Rate and Cake Moisture

Slurry Level from Shaft (mm)	Cycle Time (sec)	Temperature (°C)	Filtration Rate (kgD.S./m²h)	Cake Thickness (mm)	Moisture (% w/w water)
-600	30	5	811	4	7.4
	30	11	883	4	7.6
	30	14	1037	5	7.5
	30	17	1191	5	7.6
	30	19	1285	6	7.8

Figure 13.13 – Effect of Temperature on Filtration Rate and Cake Moisture



Source: OUTOTEC

- Rheological behavior of mixtures of coarse and fine tailings to assist in pump definition.
- Flow Property Tests:

  SGS asked Jenike & Johanson Ltd. to perform flow property tests on a sample of concentrate to determine the frozen strength and transportable moisture limit.

13.7 Forward Work Program

The work recommended during the basic engineering phase of the project is as follows:

- A final review of the options SAG vs. AG milling, considering:
  - Critical size build-up in SAG in relation with the ROM coarse size distribution; blasting and primary crushing optimization using simulation software in regard with potential SAG throughput limitation.
  - Grinding media consumption
  - Pebble crushing: potential loss of magnetite via the tramp metal rejection; plant footprint and layout.
- A review, during basic engineering, to examine whether tower mills should be used on the Project.
  - Tower mills have not been considered in the plant design because of space requirements, complexity of multiple units and the lack of a competitive market in the large size range required. However, the potential savings in electricity and grinding media costs would be significant.

- Evaluation of the metallurgical performance of the proposed flow sheet
  - A variability-testing bench-scale program should submit geographically dispersed samples from the first 10 years of the mining plan to a process that emulates the proposed flow sheet in order to evaluate the metallurgical performance of the proposed flow sheet in terms of assay quality and mass recovery.
- Mapping of the ore hardness in the mine plan
  - The block model shall include the ore SPI and BWI; perform a metallurgical model in regards to Fe grade and magnetite grade and calculate concentrate quantity and quality for each block.
- Development of a metallurgical model using METSIM® software and calculation of different mass balances for three qualities of ore (rich-average-poor) in order to validate the capacity of the plant and the sizing of the equipment.
- As soon as the site preparatory works are launched and the mine access roads are opened, a bulk sample of ore, about 200 tonnes, could be extracted in order to perform a continuous demonstration test run of the process.

14.0 MINERAL RESOURCES ESTIMATES

14.1 Previous WGM Mineral Resource Estimates

There have been three previous Mineral Resource estimates completed by WGM for Adriana/LOM. The first is the subject of a May 2009 NI 43-101 Report and is summarized in Table 14.1.

Table 14.1 – 2009 Categorized Mineral Resource Estimate for Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

Resource Classification	Tonnes (in billions)	% TFe Head	DTWR %	% SiO_2 DTC	% TFe DTC
Indicated	4.29	29.08	27.26	3.53	68.00
Inferred	1.97	29.24	26.55	3.51	68.12

WGM modelled the upper five geological sub-units (2a, 2b, 2c, 3a and 3b) of the Lac Otelnuk iron formation for the 2009 Mineral Resource estimate. Only drillholes completed by Adriana in 2007-08 were used for this Mineral Resource estimate and totalled 7,375 m in 67 holes covering approximately 9 km of strike length, referred to as the South Zone.

The second Mineral Resource estimate was an update to the 2009 estimate and was based primarily on additional infill drilling. A total of 43 holes was completed in the South Zone (up to drillhole LOS-1112) and the South Zone has now been renamed the Main Zone. WGM re-modeled the upper five geological sub-units (2a, 2b, 2c, 3a and 3b) of the Lac Otelnuk iron formation based on the results of the new drilling. The resultant Mineral Resource estimate after the infill drilling in the Main Zone is shown in Table 14.2.

Table 14.2 – 2011 Categorized Mineral Resource Estimate for Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

Resource Classification	Tonnes (in billions)	% TFe Head	DTWR %	% SiO_2 (DTC)	% TFe (DTC)
Measured	4.40	29.1	27.4	3.4	68.4
Indicated	0.49	28.3	26.3	3.2	68.5
Total M&I	**4.89**	**29.0**	**27.3**	**3.4**	**68.4**
Inferred	1.56	29.6	27.1	3.6	68.0

The infill drilling program showed that the iron formation units show excellent continuity of geology/geometry and TFe grade (the magnetic Fe grade is more variable) and was successful in upgrading the categorization of the Mineral Resources, which was the main goal of this program. No updated NI 43-101 Report was issued for this 2011 estimate, as it was not deemed to be a material change and hence did not trigger the requirement to publish a new Technical Report.

WGM prepared an updated Mineral Resource estimate for the Lac Otelnuk Iron Property in 2012 and as previously done, WGM re-modeled the upper geological sub-units of the Lac Otelnuk iron formation (2a, 2b, 2c, 3a and 3b), but also added a newly defined transitional sub unit (2b-c) identified by the Adriana Project Geology Team. It was decided to carry this sub-unit in the current Mineral Resource estimate as separate and distinct until more test work has been completed.

The 2012 Mineral Resource estimate was completed using an Inverse Distance to the power of one method with a block size of 50 m x 50 m x 5 m. As with the 2011 Mineral Resource estimate, Measured Resources are defined as blocks being within 350 m of a drillhole intercept, Indicated Mineral Resources are defined as blocks from 350 m to 500 m from a drillhole intercept and Inferred Mineral Resources are defined as blocks more than 500 m distance from a drillhole intercept and interpolated out to a maximum of approximately 1,000 m where the drilling is more sparse. This categorization was used specifically in the Main Zone area of the deposit. More widely spaced drilling directly to the north (the North Zone) and the south (the South Zone) of the Main Zone for about 3 km strike length were all classified as Indicated. Any Mineral Resources beyond the 3 km extension on either side of the Main Zone was classified as Inferred, due to even more widely spaced drilling. The deeper intersections of mineralization, predominantly on the northeastern down-dip extension of the deposit, generally lie beneath 70 m or more of cover rock and this mineralization was re categorized as Inferred.

The 2012 Mineral Resource estimate included the new drilling results from the 2010 and 2011 exploration programs and estimate uses of total of 213 drillholes. A summary of the 2012 Mineral Resources is provided in Table 14.3.

Table 14.3 – 2012 Categorized Mineral Resource Estimate for Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

Resource Classification	Tonnes (in billions)	TFe Head %	DTWR %	Magnetic Fe %
Measured	5.51	29.2	26.8	18.4
Indicated	5.84	28.7	25.3	17.5
Total M&I	**11.35**	**28.9**	**26.0**	**17.9**
Inferred	12.39	30.4	26.0	17.8

Notes: The previous Mineral Resource estimates from 2009 to 2012 are no longer current and should not be relied upon.

14.2 Exploration Data Analysis

14.2.1 Drill Hole Data

Data used to generate the Mineral Resource estimate originated from digital spreadsheet / database files supplied to WGM by Adriana technical personnel and MRB. A Gemcom™ project was established to hold all the requisite data to be used for any manipulations necessary and for completion of the Mineral Resource estimate.

The current Gemcom™ drillhole database consisted of 448 drillholes, which includes 36 holes drilled in the 1970s, primarily in the North Zone, 21 NQ hydrogeological and geotechnical holes and 21 PQ holes to collect material for metallurgical test work. No assaying of the PQ holes was completed and were twins to existing BQ core holes that are used for the Mineral Resource estimate.

Only drillholes completed by Adriana from 2007 to 2012 were used for the current Mineral Resource estimate and totalled 370 drillholes (46,150 m) covering approximately 36 km of strike length. The primary objective of the 2012 drilling program was to complete the grid pattern drilling along the 36 km strike length of the Lac Otelnuk deposit at a drillhole spacing of 600 m by 500 m and to bring the sub-units to surface wherever possible. The early drilling was limited to in-fill drill sites previously prepared in the Main Zone; the drilling then expanded to the north and south as field conditions improved.

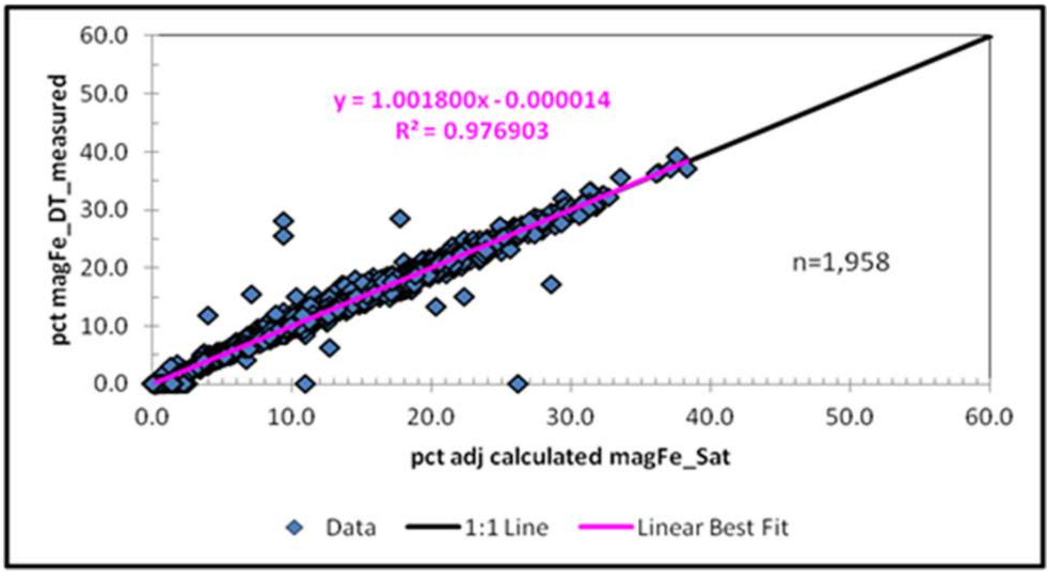
Most of the drilling was focused on completing the grid pattern on the "North Zone" of the deposit from Line 30 S north to Line 250 N. The North Zone had previously been tested on a much wider grid pattern and this allowed for an upgrade of categorization of the Mineral Resources for this report. As predicted in the previous WGM report, the pre-fix LOS and LON were discarded for the most recent drilling program; the South Zone and North Zone are no longer referred to by LOM as these zones are continuous and it makes little sense to retain this nomenclature. The drilling was all vertical and penetrated the entire iron formation stratigraphy.

The drillholes in the database contains geological codes for each unit and sub-unit and multi element assay data for Head, Davis Tube concentrate analyses and Satmagan determinations for the sampled intervals used for the Mineral Resource estimate. The sampled intervals totalled 8,973; 7448 intervals were within the sub units that were modelled for the estimate. The range of sample lengths was 0.2 m to 9.6 m, with an average length of 4.1 m. Almost 90 % of the assayed intervals are between 3.0 m and 5.0 m in length. Additional information, including copies of the geological logs, summary reports and internal geological interpretations were supplied to WGM digitally or as hard copies.

14.2.2 Changeover from DTWR to Satmagan

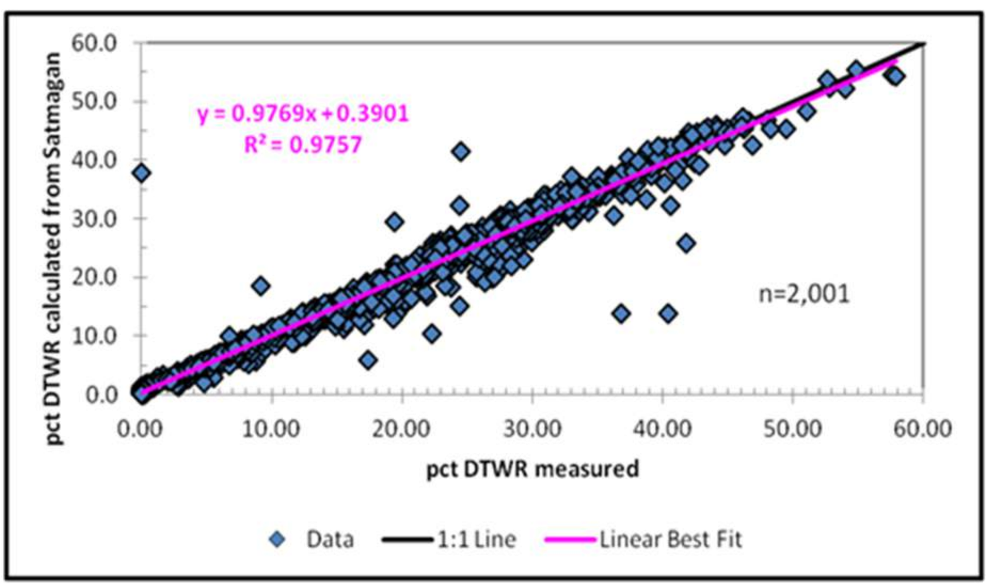
From 2007 through 2010, Adriana completed Davis Tube tests to estimate DTWR and MagFe as described in Section 11. In 2011, Adriana although still maintaining Davis Tube tests for selected samples, switched to using Satmagan as its preferred methodology for assessing MagFe. Using samples where both Davis Tube tests and Satmagan determinations were completed on the same samples, MagFe estimated from Satmagan was found to be statistically slightly higher than MagFe from Davis Tube tests (See Mineralization Section). The difference may be due to Satmagan calibration issues or small losses of magnetite from the Davis Tube. In order to complete the Mineral Resource estimate using all the data together starting in 2007 to present, WGM has reduced the Satmagan MagFe results slightly, normalizing them to the Davis Tube results. To perform this normalization WGM applied the equation of the linear best fit function that relates the Satmagan MagFe results to the Davis Tube MagFe results where samples were tested both ways. Figure 14.1 shows the results of the normalization applied to the samples.

Figure 14.1 – % Adj_MagFe_Sat



In addition, WGM completed a projection of DTWR from Satmagan results so that DTWR could be reported in the Mineral Resource estimate to enable a clearer comparison of the current Mineral Resource estimate with the previous estimates (for consistency). To complete this projection of DTWR from Satmagan, WGM used the relationship between raw Satmagan MagFe results and DTWR for the samples where both Satmagan and Davis Tube tests were completed. Figure 14.2 shows the results of applying this function to the samples and shows calculated DTWR from Satmagan versus measured DTWR from Davis Tube. The high quality of the correlation is illustrated on this figure.

Figure 14.2 – % Measured DTWR



14.2.3 Data Validation

Upon receipt of the data, WGM performed the following validation steps:

- ✓ checking for location and elevation discrepancies by comparing collar coordinates with the copies of the original drill logs received from the site;
- ✓ checking minimum and maximum values for each quality value field and confirming/modifying those outside of expected ranges;
- ✓ checking for inconsistency in lithological unit terminology and/or gaps in the lithological code;
- ✓ spot checking original assay certificates with information entered in the database; and
- ✓ checking gaps, overlaps and out of sequence intervals for both assays and lithology tables.

The database tables contained some minor errors and these were corrected and confirmed by Adriana technical personnel before proceeding with the Mineral Resource estimate. The gaps or missing intervals identified were due to unsampled / unassayed intervals outside of the mineralized zones. A number of revisions to assays were also made on the basis of the 2013 Check assaying program. In general, WGM found the database to be in good order and accurate and no errors were identified that would have a significant impact, therefore WGM found the database to be appropriate for Mineral Resource estimation.

14.2.4 Database Management

The drillhole data were stored in a Gemcom™ multi-tabled workspace specifically designed to manage collar and interval data. The line work for the geological interpretations and the resultant 3D wireframes were also stored within the Gemcom™ Project. The Project database stored cross section and level plan definitions and the block models, such that all data pertaining to the project are contained within the same Project database.

14.3 Geological Interpretation

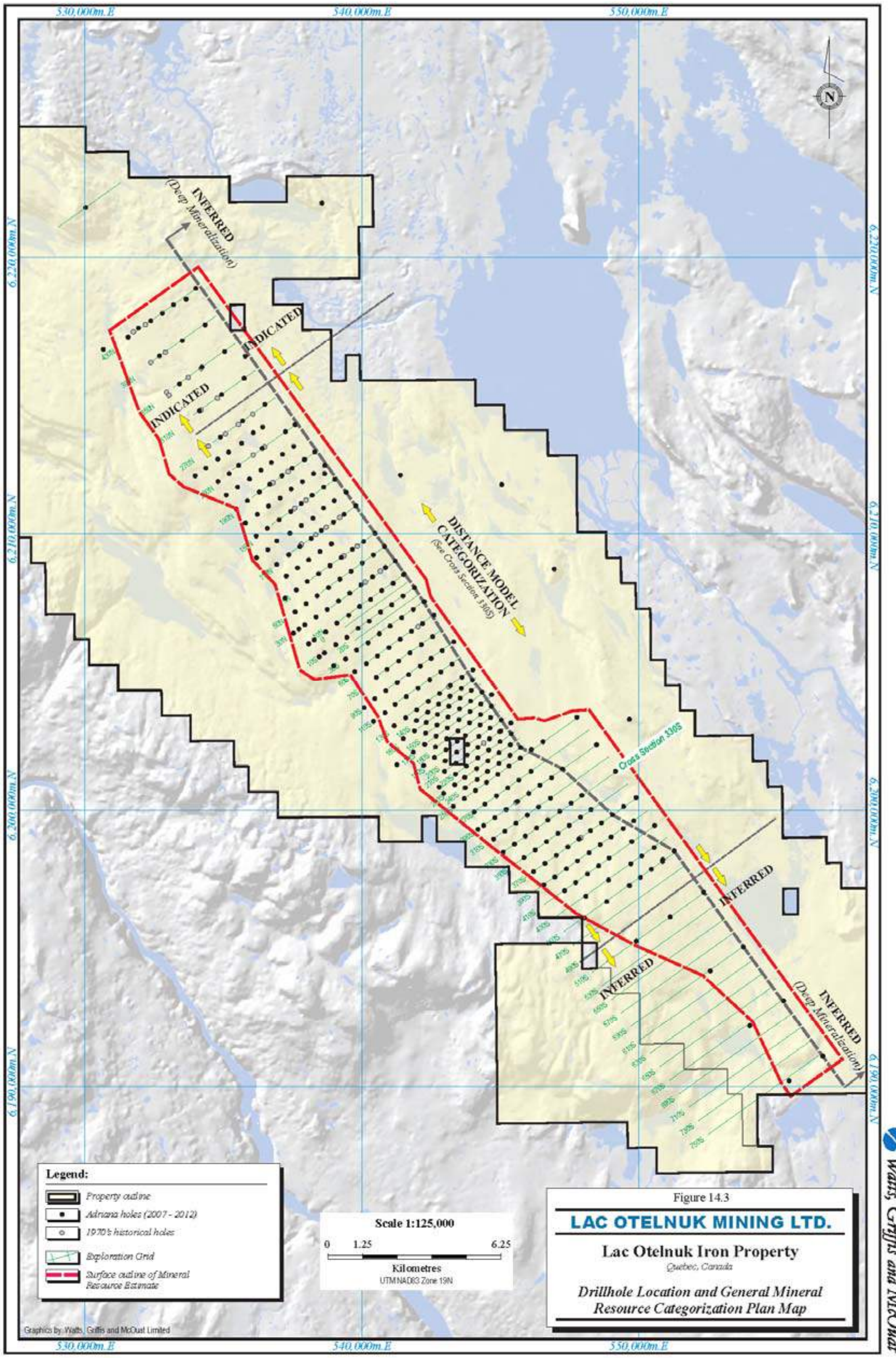
14.3.1 Cross Section Definition

More than 50 vertical cross sections were defined for the Property at a nominal spacing of approximately 600 m along the drillhole lines. Where drillhole spacing was closer than that due to infill drilling (predominantly in the Main Zone), the distances between cross sections were reduced to about half that. In the north and south areas of the Property, the distances between cross sections could reach 1,200 m, or about double the nominal cross section separation to accommodate more widely spaced drilling along the strike of the deposit. In the extreme south of the Property the cross section spacing is even wider. In general, each cross section contained six to ten holes on a nominal 500 m spacing (varying from about 450 m to 550 m) on the well drilled sections, and a 1,000 m to 1,500 m spacing on the more widely drilled cross sections in the north and south parts of the deposit. Figure 14.3 shows the drillhole plan and the cross section locations.

14.3.2 Geological Interpretation and 3D Wireframe Creation

WGM used our previous geological interpretations and knowledge of the deposit as a guide to redefine the boundaries of the mineralized sub-zones / sub-units. WGM's zone interpretations of the mineralization were digitized into Gemcom™ and each polyline was assigned an appropriate rock type and stored with its section definition. The digitized lines were "snapped" to drillhole intervals to anchor the line which allows for the creation of a true 3D wireframe that honours the 3D position of the drillhole interval. Any discrepancies or interpretation differences between Adriana's unit definitions from logging and/or the supplied database and WGM's final interpretations were discussed with Adriana technical personnel. This involved much back and forth with WGM and the field personnel (with associated re assaying and re-logging) before completion of the geological model for the Mineral Resource estimate, as complications arose with the interpretation of some units during the final modeling stage.

Figure 14.3 – Drillhole Location and General Mineral Resource Categorization



Zone (sub-unit) boundaries were digitized from drillhole to drillhole that showed continuity of strike, dip and grade, generally from 500 m to 600 m in extent, and up to a nominal 1,000 m on the ends of the zones and at depth where there was no drillhole information, if the interpretation was supported by drillhole information on adjacent cross sections. Internally, the continuity of the sub-units was excellent, so WGM had no issues with extending the interpretation beyond the 600 m distance, as long as there was supporting data from adjacent sections. This extension was taken into consideration when classifying the Mineral Resources and these areas were given a lower confidence category.

Using the results of the most recent drilling data, WGM re-modeled the upper geological sub units of the Lac Otelnuk iron formation that were previously defined (2a, 2b, 2c, 3a and 3b), keeping the transitional sub unit (2b-c) identified previously due to the thought that this unit has slightly different mineralogical and metallurgical characteristics. This sub-unit remains as a distinct sub-unit until additional confirmation test work has been completed.

Any of the other thinner sub-units logged as transitional were combined with the overlying sub units in an effort to simplify the interpretation somewhat and to eliminate thin intervals of transitional material, as this transitional material was not identified in every drillhole and could not be correlated between holes.

WGM also added an internal shale waste unit north of the old Main Zone. This waste unit has become better defined with additional drilling and is more prominent and thicker to the north. It directly underlies sub-unit 2c. There is some confusion on whether to identify this unit as shale or 3c, so these were used almost interchangeably to define this internal waste unit in the north part of the Property. It is not uncommon for this waste unit to reach thicknesses of 30 m to 50 m to the northwest, but it thins and pinches out down-dip and to the east the further south one goes until about Line 30S where it disappears completely.

WGM applied a cut-off of 10 % DTWR to define the top of sub-unit 2a, as it is a gradational boundary from the overlying U1 unit. This upper boundary is not easily visually logged, so the assay results (and magnetic susceptibility readings) were used as a guide to redefine the top of this sub-unit. The same cut-off was also applied to define the bottom of sub-unit 2c where it does not immediately grade into sub-unit 3b or the newly defined internal waste unit (including 3c) to the north. The 10 % DTWR is close to a natural cut-off and was deemed to be appropriate to redefine these sub-unit boundaries in specific cases.

Sub-unit 2a has an average thickness of approximately 12.2 m, 2b averages 21.4 m thick, 2b-c averages 8.8 m thick, 2c averages 19.5 m thick, 3a averages 10.2 m thick and 3b averages 16.8 m thick. The 2012 drilling program showed that the iron formation units have excellent continuity of geology/geometry and average thickness in the main part of the deposit, however, to the north and the south, the sub-unit thicknesses become more variable. Some sub-units are not present/identified in some holes as the units have either pinched out or been eroded away at surface on the up-dip extension of the stratigraphic package.

There appears to be some structural complexity to the northeast of the deposit where possible thrusting has occurred, but it was not followed up from the previous Mineral Resource estimate as this was not the focus of the 2012 drilling program. Future drilling could be done in these areas to get a better understanding of the nature of this complexity and how it affects the stratigraphic package. In general, the recent drilling program was successful in upgrading the categorization of the existing Mineral Resources and expanding the resources where continuity was not certain due to lack of drilling.

Figure 7.5, shown previously, illustrates a typical cross section (Line 330S) through the Lac Otelnuk stratigraphy and illustrates the definition of the mineralized sub-units for the Mineral Resource estimate.

14.3.3 Topographic and Overburden Surface Creation

As described in Section 9 of this report, Adriana contracted Eagle Mapping to carry out an airborne imaging survey to build a digital terrain model ("DTM") to cover the Property area. The topographic surface used for the 2013 Mineral Resource estimate did not change from the 2012 estimate.

Aerial triangulation used the airborne GPS data in conjunction with the ground survey coordinates, with a resultant aerial triangulation of a series of geo-referenced stereo models for topographic and feature collection in 3D. The mapping project was referenced to NAD83, UTM Zone 19N datum. Final maps were produced in AutoCAD at 1:1,000-scale with 1 metre contours and it is this DTM that was used for the current resource estimate.

An overburden contact (3D surface) was created from the logged intervals in each drillhole. In general, this overburden layer is quite thin and exceeds 10 m in only a few holes, averaging less than 4 m. The overburden surface predominantly mimics the topography however, where there is little drillhole information in the extreme northern and southern extensions of the mineralized zone, the 3D overburden triangulation created from the drillholes is not accurate and sometimes crosses the topography in areas of higher relief. The 2012 infill drilling (especially when tracing the up-dip extension of the sub-units to surface) aided in creating a more accurate overburden surface than previously generated in these areas.

In rare cases when the interpolated overburden surface crossed the DTM, WGM used the topographic surface as the bounding surface for the Mineral Resource estimate. Any remaining overburden overlap with the topography does not have a material effect on the estimate, but should be corrected during the next phase of more advanced studies.

14.4 Mineral Resource Block Modeling

14.4.1 General Mineral Resource Estimation Procedures

The block model Mineral Resource estimate procedure included:

- Importing / compiling and validation of digital data into Gemcom Software International Inc.'s ("Gemcom™") geological software package to create a Project database. The database was validated both within MS Access and Gemcom™;
- generation of cross sections and plans to be used for geological interpretations;

- basic statistical analyses to assess cut-off grades, compositing and cutting (capping) factors, if required;
- development of 3D wireframe models for zones with sufficient continuity of geology/mineralization, using available geochemical assays for each drillhole sample interval; and
- generation of block models for Mineral Resource estimates for each defined zone and categorizing the results according to NI 43-101 and CIM definitions.

14.4.2 Back-Coding of Rock Code Field

The 3D wireframes / solids that represented the interpreted mineralized sub-units (and the large internal waste zone in the northern part of the Property) were used to back-code a rock code field into the drillhole workspace, and these were checked against the logs and Adriana's interpretation and adjusted where required, either based on returned assay results, re-logging of core, or to simplify the 3D interpretation somewhat. Each interval in the original assay table was generally kept as logged, but WGM generated composite tables for the Mineral Resource estimate and a new rock code was assigned, if necessary, based on the rock type wireframe that the interval midpoint fell within. As previously mentioned, the thinner intervals that could not be correlated from one drillhole to another or the transition zones logged in the field (except for the sub-unit 2b-c) were either "absorbed" within an existing larger sub-unit or were combined with the sub-unit above them (if at the base) and the intervals were back-coded based on the new "combined" rock code. WGM was of the opinion that thinner sub-units that had very little consistency would add needlessly to a more complex interpretation and would have little value in the Mineral Resource estimate.

14.4.3 Statistical Analysis and Compositing

In order to carry out the Mineral Resource grade interpolation, a set of equal length composites of 3.0 m was generated from the raw drillhole intervals, as the original assay intervals were different lengths (see Section 14.4) and required normalization to a consistent length. A 3 m composite length was chosen so that more than one composite would be used for grade interpolation in sub-units that were thinner. Regular down-the-drillhole compositing was used, and WGM retained all the composites regardless of their length, as there was no indication that the last interval in any sub unit was lower or higher grade that the composite above it and, in general, the sub-units are gradational with each other and all the samples should be used. Only about 6 % of the 3 m composite lengths used for the Mineral Resource estimate were below 1 m (with 85 % being between 2.5 and 3.0 m) and WGM was of the opinion that discarding the last interval if it was below a certain threshold length (to keep them relatively constant) would have no significant effect on the grade interpolation or weighting.

Table 14.4 summarizes the statistics of the 3 m composites inside the defined sub-units for %Fe Head, %DTWR and %MagFe and Figure 14.4 to Figure 14.6 show representative histograms for sub-unit 2c. All sub-units showed comparable patterns of grade distribution but sub-unit 2c was used for illustrative purposes because it has the most samples.

Table 14.4 – Basic Statistics of 3 m Composites

Zone	Element	Number	Minimum	Maximum	Average	C.O.V.
Sub-unit 2a	% Fe Head	991	0.00	40.90	30.12	0.18
	% DTWR	991	0.00	59.70	29.20	0.38
	% MagFe	993	0.00	41.30	20.09	0.38
Sub-unit 2b	% Fe Head	2,131	0.00	42.20	33.68	0.13
	% DTWR	2,131	0.00	50.96	23.61	0.38
	% MagFe	2,131	0.00	35.26	16.29	0.38
Sub-unit 2b-c	% Fe Head	962	16.56	38.80	26.59	0.10
	% DTWR	962	1.10	46.38	22.27	0.33
	% MagFe	962	0.57	32.60	15.27	0.33
Sub-unit 2c	% Fe Head	2,314	0.00	37.20	27.57	0.10
	% DTWR	2,299	0.00	42.16	22.05	0.28
	% MagFe	2,299	0.00	29.15	15.15	0.29
Sub-unit 3a	% Fe Head	995	17.80	32.63	25.85	0.09
	% DTWR	995	0.44	36.79	16.03	0.48
	% MagFe	995	0.00	24.72	10.93	0.48
Sub-unit 3b	% Fe Head	1,777	11.82	39.40	27.64	0.05
	% DTWR	1,777	6.95	48.20	27.85	0.18
	% MagFe	1,777	0.00	33.30	19.12	0.19

Figure 14.4 – Normal Histogram, % TFe Head for Sub-Unit 2c

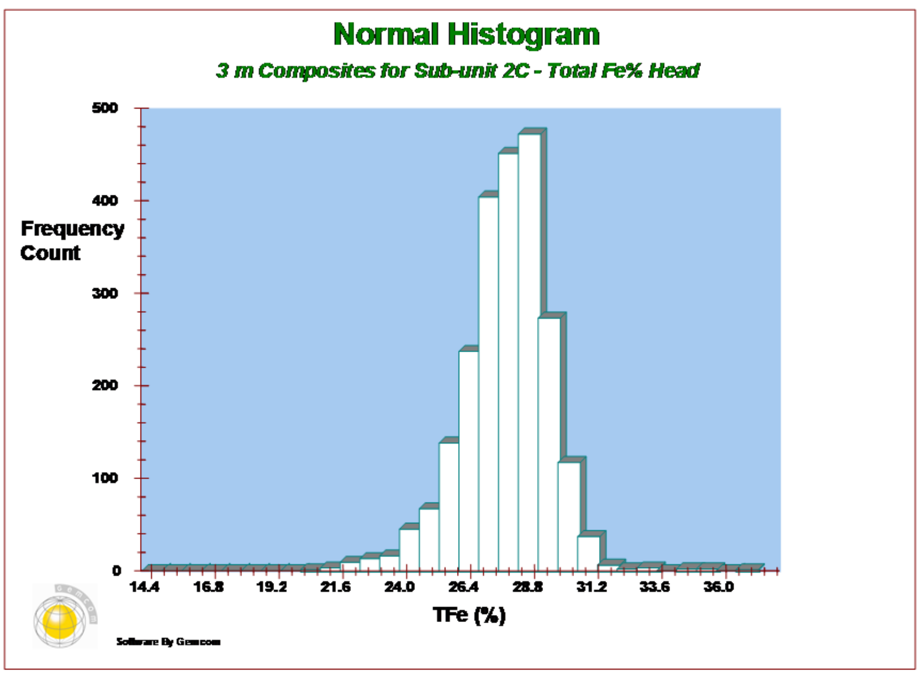


Figure 14.5 – Normal Histogram, % DTWR for Sub-Unit 2c

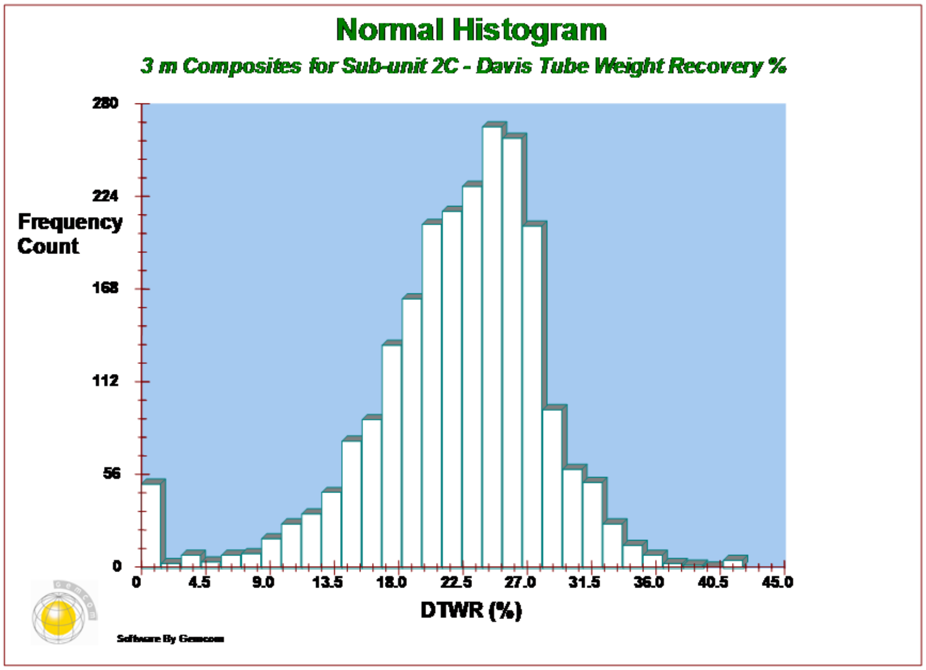
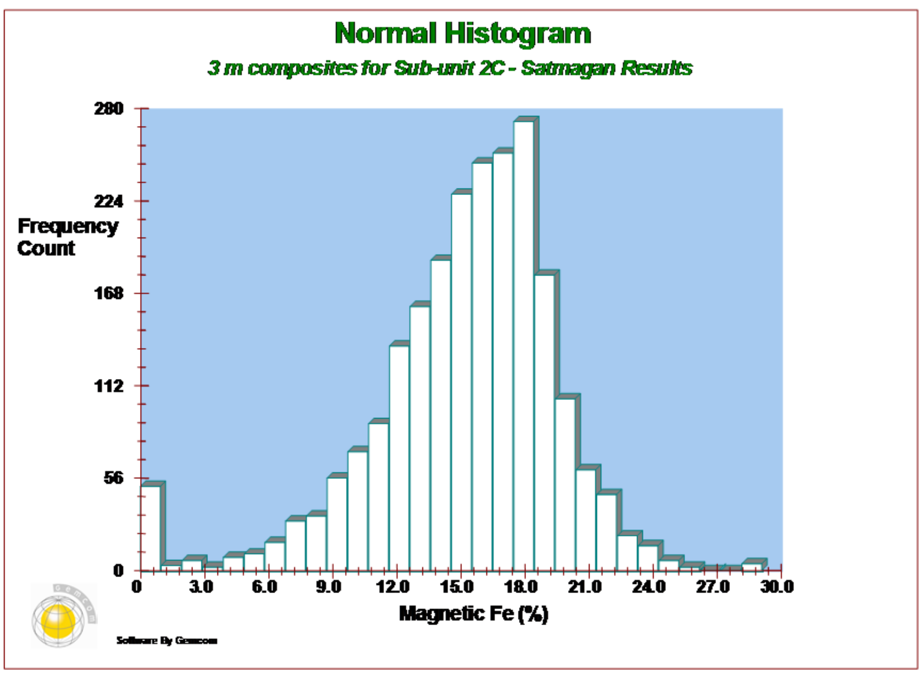


Figure 14.6 – Normal Histogram, Satmagan (% MagFe) for Sub-Unit 2c



14.4.4 Grade Capping

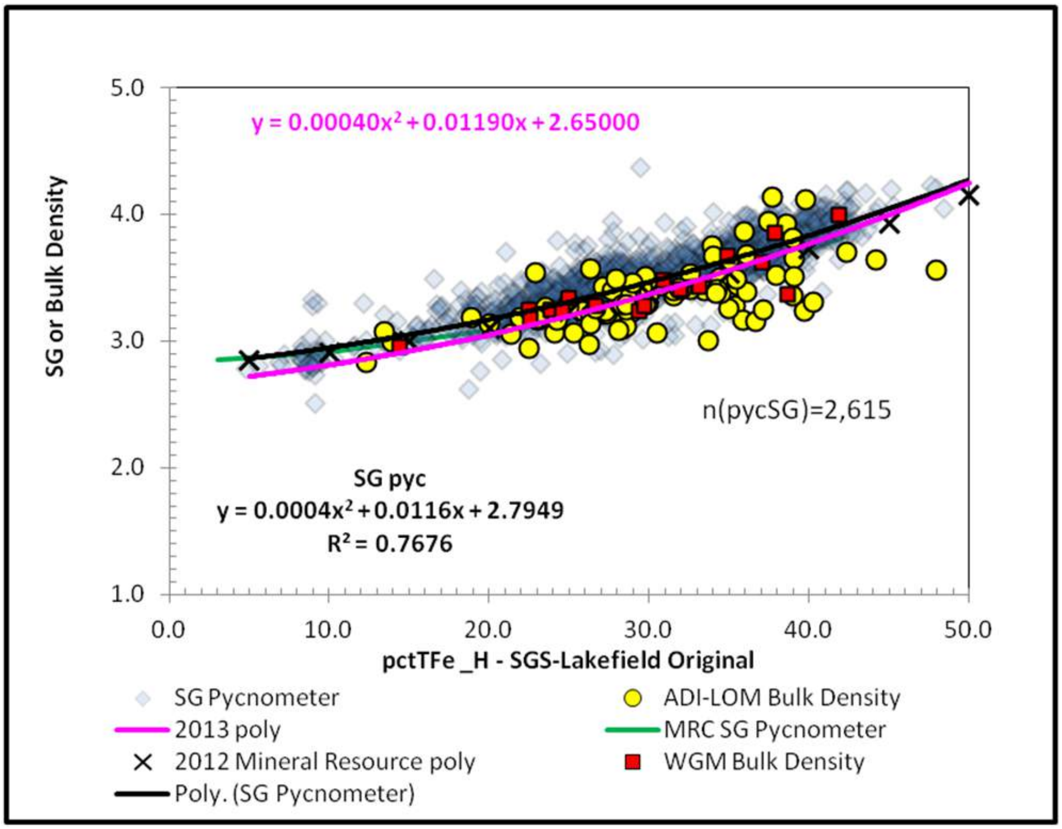
The statistical distributions of the modelled elements show good normal distributions for the sub-units; the thicker sub-units show the best normal distributions due to the abundance of samples. Sub-units 2a, 2b and 3b are the highest grade (with 2a and 3b having the highest magnetite content), but all sub-units exhibit similar behaviour of grade distributions, except for sub-unit 3a, which is magnetite-poor. Grade capping, also sometimes referred to as top cutting, is commonly used in the Mineral Resource estimation process to limit the effect (risk) associated with extremely high assay values, but considering the nature of the mineralization and the continuity of the zones, WGM determined that capping was not required for the Lac Otelnuk mineralization.

14.4.5 Density/Specific Gravity

For the initial Mineral Resource estimates on the Property, WGM used one average density value for each sub-unit for the Mineral Resource estimate. In 2009, WGM completed an assessment of over 300 samples and graphed TFe vs. specific gravity (SG) for each sub-unit. This has since been updated with new results and summarized earlier in this report. The results are similar to 2009 but are considered to be more representative due to the increased number of samples measured.

As aforementioned in this report as part of the sampling and assaying protocol, Adriana designated periodic samples for determinations of specific gravity which were completed by the gas comparison pycnometer method on prepared pulverized material. Adriana also requested additional bulk density determinations on a select number of samples that were completed on entire ½ split core by weighing in air and weighing in water. Figure 14.7 (shown previously and reproduced below) shows specific gravity/bulk density results for all of the available pycnometer and bulk density determinations graphed versus Head TFe. Also shown are results for 17 samples collected by WGM from drill core and submitted to SGS-Lakefield for bulk density determinations. There is significantly more data available than for the previous Mineral Resource estimate and WGM completed a similar assessment of TFe vs. SG.

Figure 14.7 – Pycnometer SG or Bulk Density vs. % TFe for All Samples



The above graph shows several best fit trend lines, including Check assay work completed at MRC in 2012. We found that the pycnometer and bulk density measurements were fairly consistent and defined the same trend therefore we were not overly concerned about internal porosity being an issue when determining SGs. The black trend line which represents a trend line fit to the pycnometer SG results is closely coincident to the MRC trend line shown in green. The black trend line is similar but slightly different than the one defined previously and does reasonably fit the various bulk density results so WGM has revised the SG function used for the new Mineral Resource estimate to the 2013 polyline in the graph above to better fit the now more plentiful bulk density results. The pattern shown is very typical for iron formation. The best fit curve describing the relationship between SG and TFe and modeled above was used to create a variable density model to estimate tonnage.

WGM also reviewed the pycnometer SG results by sub-unit and the results on a sub-unit basis are generally consistent with "all sample/all sub-unit patterns", however, there are a few outliers defined from departure from the best fit trend line. Follow-up work to determine the cause of these outliers was included as a component of the 2013 Check assay program.

14.5 Grade Interpretation Methodology and Block Model Parameters

14.5.1 General

The Lac Otelnuk Mineral Resource estimate was completed using a block modelling method and for the purpose of this study, the grades have been interpolated using an Inverse Distance ("IDW") to the power of one estimation technique. IDW belongs to a distance-weighted interpolation class of methods, similar to Kriging, where the grade of a block is interpolated from several composites within a defined distance range of that block. IDW uses the inverse of the distance (to the selected power) between a composite and the block as the weighting factor.

For comparison and cross checking purposes, the IDW^2 and IDW^{10} methods, which closely resembles a Nearest Neighbour ("NN") technique, was used. In the NN method, the grade of a block is estimated by assigning only the grade of the nearest composite to the block. All interpolation methods gave similar results, as the grades were well constrained within the sub-unit wireframes, and the results of the interpolation approximated the average grade of the all the composites used for the estimate. WGM's experience with similar types of deposits showed that geostatistical methods, like Kriging, gave very similar results when compared to IDW interpolation, therefore we are of the opinion that IDW interpolation is appropriate.

14.5.2 Block Model Setup and Parameters

The block model was created using the Gemcom™ software package to create a grid of regular blocks to estimate tonnes and grades. The deposit specific parameters used for the block modelling are summarized below.

The block sizes used were:

Width of columns:	50 m
Width of rows:	50 m
Height of blocks:	5 m

The specific parameters for each block model are as follows:

Easting coordinate of model bottom left hand corner:	551500.00
Northing coordinate of model bottom left hand corner:	6184800.00
Datum elevation of top of model:	530.00 m
Model rotation:	35.50
Number of columns in model:	200
Number of rows in model:	800
Number of levels:	110

14.5.3 Grade Interpolation

The geology and geometry of the sub-units is fairly well understood and consistent, so the search ellipse size and orientation for the grade interpolation were based on this geological knowledge. The following lists the grade interpolation parameters:

IDW Search Ellipsoid:

2,000 m in the East-West direction

2,500 m in the North-South direction

100 m in the Vertical direction

Minimum / Maximum number of composites used to estimate a block: 2 / 12

Maximum number of composites coming from a single hole: 3

Ellipsoidal search strategy was used with rotation about X, Y, Z of 0°, 4°, 0°.

The large search ellipse was used in order to inform all the blocks in the block model with grade, however, the classification of the Mineral Resources (see below) was based on drillhole density (or drilling pattern), geological knowledge and interpretation of the sub-units and WGM's experience with similar deposits.

Gemcom™ does not use the sub-blocking method for determining the proportion and spatial location of a block that falls partially within a wireframe object. Instead, the system makes use of a percent or partial block model (if it is important to track the different rock type's proportions in the block – usually if there is more than one important type) or uses a "needling technology" that is similar in concept, but offers greater flexibility and granularity for accurate volumetric calculations. In the needling technique, all the blocks that are inside the wireframe (the user specifies the %threshold) are coded and thus are assigned the appropriate rock code and the interpolated grade. During the volumetric calculation, Gemcom™'s needling process reports only the volume / tonnage of the block actually within the wireframe itself, but applies the interpolated grade to that portion of the block within the wireframe / solid. Since WGM did not use the percent model approach, a block height of 5 m was used to get better resolution on the geological coding for portions of the thinner sub-units.

14.6 Mineral Resource Classification

The classification of Mineral Resources used in this report conforms to the definitions provided in the final version of NI 43-101, which came into effect on February 1st, 2001, as revised on May 10th, 2014. WGM further confirms that, in arriving at our classification, we have followed the guidelines adopted by the Council of the Canadian Institute of Mining Metallurgy and Petroleum ("CIM") Standards. The relevant definitions for the CIM Standards/NI 43-101 are as follows:

- A ***Mineral Resource*** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

- An ***Inferred Mineral Resource*** is that part of a ***Mineral Resource*** for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An ***Inferred Mineral Resource*** has a lower level of confidence than that applying to an ***Indicated Mineral Resource*** and must not be converted to a ***Mineral Reserve***. It is reasonably expected that the majority of ***Inferred Mineral Resources*** could be upgraded to ***Indicated Mineral Resources*** with continued exploration.
- An ***Indicated Mineral Resource*** is that part of a ***Mineral Resource*** for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of ***Modifying Factors*** in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An ***Indicated Mineral Resource*** has a lower level of confidence than that applying to a ***Measured Mineral Resource*** and may only be converted to a ***Probable Mineral Reserve***.
- A ***Measured Mineral Resource*** is that part of a ***Mineral Resource*** for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of ***Modifying Factors*** to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A ***Measured Mineral Resource*** has a higher level of confidence than that applying to either an ***Indicated Mineral Resource*** or an ***Inferred Mineral Resource***. It may be converted to a ***Proven Mineral Reserv***e or to a ***Probable Mineral Reserve***.
- ***Modifying Factors*** are considerations used to convert ***Mineral Resources*** to ***Mineral Reserves***. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.
- A ***Mineral Reserve*** is the economically mineable part of a ***Measured*** and/or ***Indicated Mineral Resource***. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at ***Pre-Feasibility*** or ***Feasibility*** level as appropriate that include application of ***Modifying Factors***. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The reference point at which ***Mineral Reserves*** are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

- The public disclosure of a ***Mineral Reserve*** must be demonstrated by a ***Pre-Feasibility Study*** or ***Feasibility Study***.
- A ***Probable Mineral Reserve*** is the economically mineable part of an ***Indicated***, and in some circumstances, a ***Measured Mineral Resource***. The confidence in the ***Modifying Factors*** applying to a ***Probable Mineral Reserve*** is lower than that applying to a ***Proven Mineral Reserve***.
- A ***Proven Mineral Reserve*** is the economically mineable part of a ***Measured Mineral Resource***. A ***Proven Mineral Reserve*** implies a high degree of confidence in the ***Modifying Factors***.

The Mineral Resource classification is based on certainty and continuity of geology and grades. In most deposits, there are areas where the uncertainty is greater than in others. The majority of the time, this is directly related to the drilling density. Areas more densely drilled are usually better known and understood than areas with sparser drilling, which would be considered to have greater uncertainty, and hence lower confidence.

The block size chosen for the Mineral Resource estimate was kept the same as the previous estimate at 50 m x 50 m x 5 m high for better refinement of the thinner sub-units and due to the closer spaced infill drilling in the Main Zone. Additional analysis and further refinement was completed based on the new drilling since the 2012 Mineral Resource estimate for the categorization of the resources.

A significant diamond drilling program was carried out in 2012 and 157 delineation holes were drilled to expand and upgrade the Mineral Resource. The main goals of the program were to complete infill drilling in the north part of the defined Mineral Resource area (previously named North Zone) on a 600 m by 500 m grid to upgrade the resource categorization and to accurately define the limits of the Main Zone (previously named the South Zone) of the deposit by extending the up-dip mineralization to surface along the western margin of the mineralization.

Measured Resources are defined as blocks being within 400 m of a drillhole intercept, Indicated Mineral Resources are defined as blocks from 400 m to 600 m from a drillhole intercept and Inferred Mineral Resources are defined as blocks more than 600 m distance from a drillhole intercept and interpolated out to a maximum of approximately 1,000 m where the drilling is more sparse, predominantly in the deeper parts of the deposit. This categorization was used specifically in the previously named "Main Zone" area of the deposit and directly to the north of this area where more infill drilling was completed during 2012. This categorization is illustrated on Cross Section L330S (Figure 14.8); the same cross section as the type geological section in Section 7 of this report.

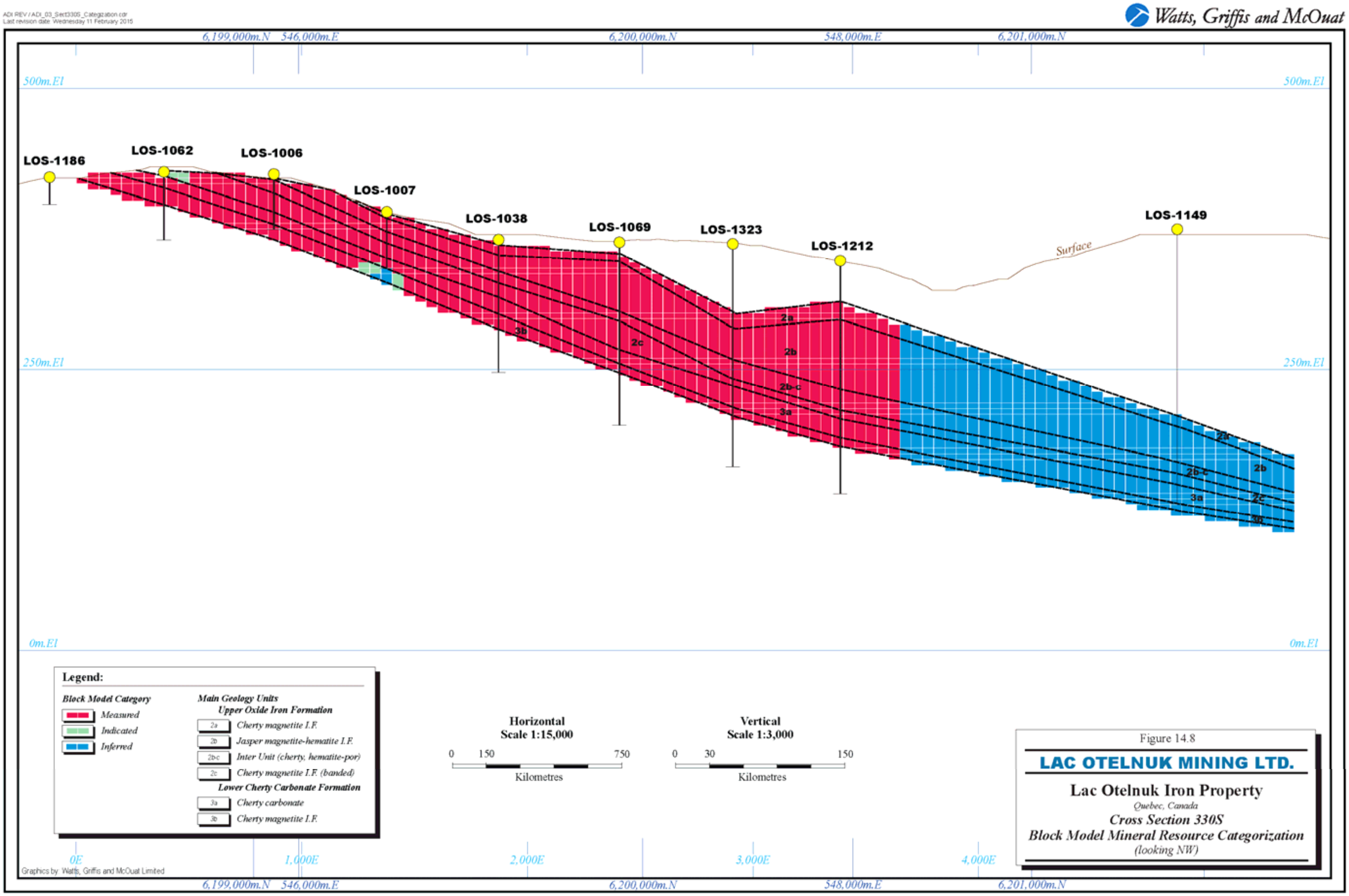
Mineralization defined by more widely spaced drilling north of Line 270 N has been classified as Indicated and Mineral Resources south of Line 490 S were classified as Inferred, due to even more widely spaced drilling. The deeper intersections of mineralization, predominantly on the northeastern down-dip extension of the deposit, generally lie beneath 70 m or more of cover rock. This mineralization was re-categorized as Inferred, regardless of the distance to a drillhole, to account for the uncertainty that would be associated with a higher stripping ratio. These general areas for the Mineral

Resource categorizations not based strictly on search ellipse distances are shown previously on Figure 14.3.

Internally, the continuity of the sub-units was excellent, so WGM had no issues with extending the interpretation beyond the more densely drilled parts of the deposit, as long as there was supporting data from adjacent sections. This extension was taken into consideration when classifying the Mineral Resources and these areas were given a lower confidence category. Variograms were also generated along strike and across the deposit in support of these distances.

Because the search ellipses were large enough to ensure that all the blocks in the model were interpolated with grade, WGM generated a "Distance Model" (distance from actual data point to the block centroid) and reported the estimated Mineral Resources by distances which represented the category or classification. The blocks in some of these resource categories that were based on search ellipse distances alone were re-categorized, as described above. The Measured and Indicated portion of the current Mineral Resource extends over a strike length of about 27 km. The Inferred portion of the deposit south of the Measured and Indicated Mineral Resource, adds an additional strike length of about 9 km to the deposit for a total strike length of approximately 36 km.

Figure 14.8 – Cross Section 330S Showing Block Model Mineral Resource Categorization



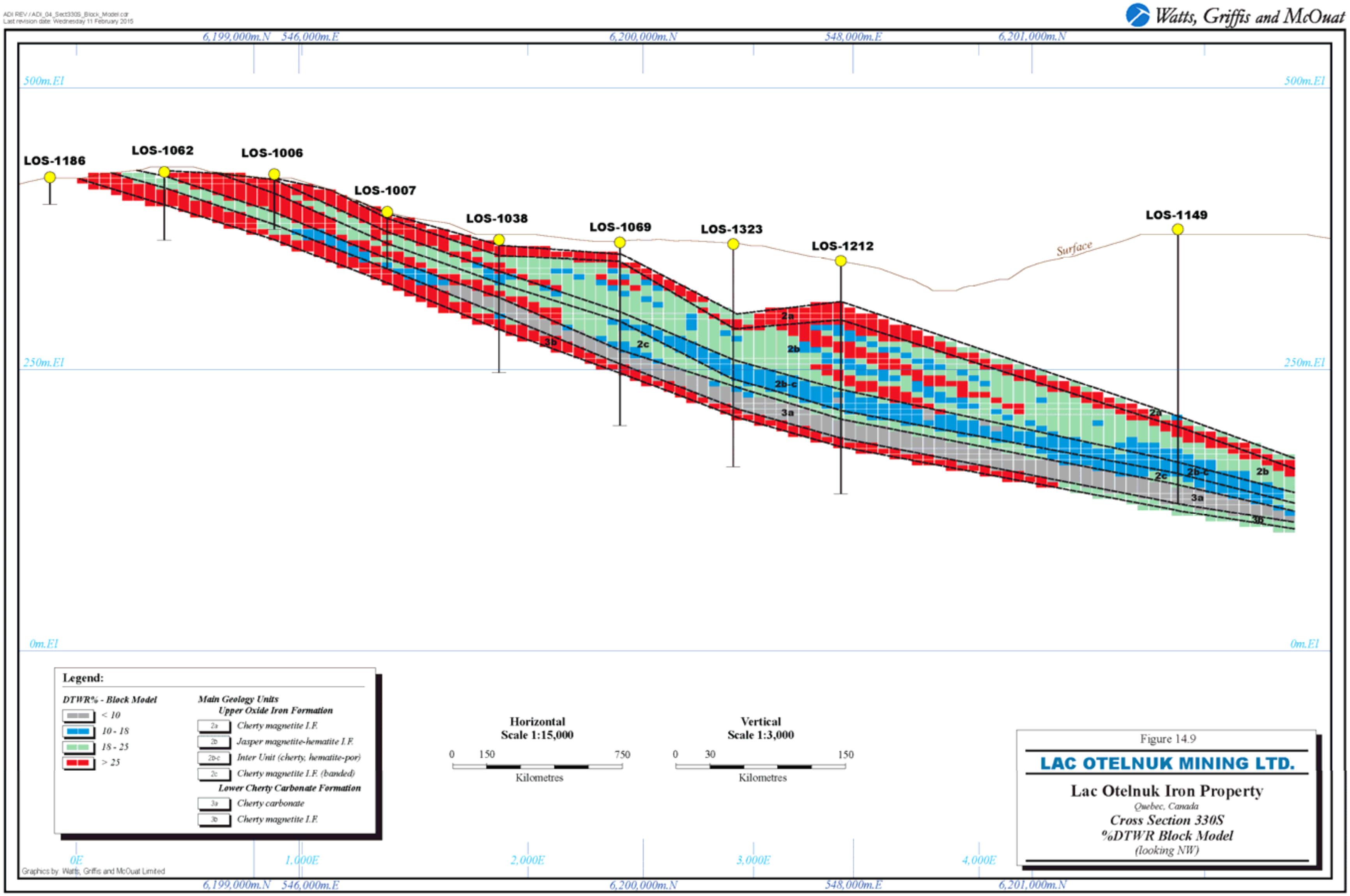
The average distances per category for most of the sub-units were similar (especially for the Measured) and are shown in Table 14.5. If the Mineral Resources were categorized solely on the Distance Model, then the average distance for Measured would be about 220 m, Indicated would be about 490 m and Inferred would be 860 m. However, since the Distance Model was re categorized in the north and south parts of the resource area, the average distances as reflected in the table below, were about 210 m for Measured, about 470 m for Indicated and 570 m for Inferred.

Table 14.5 – Average Interpolation Distance for Resource Categorization

Zones	Average Distance Measured	Average Distance Indicated	Average Distance Inferred
Unit 2a	212 m	441 m	557 m
Unit 2b	214 m	452 m	525 m
Unit 2b-c	219 m	495 m	598 m
Unit 2c	209 m	446 m	572 m
Unit 3a	213 m	465 m	369 m
Unit 3b	211 m	468 m	475 m

Figure 14.9 on the next page shows the interpolated % DTWR blocks on Cross Section 330 S.

Figure 14.9 – Cross Section 330S - % DTWR Block Model



14.7 Mineral Resource Estimation

WGM has prepared a Mineral Resource estimate for the Lac Otelnuk Iron Property mineralized zones that have sufficient data to allow for continuity of geology and grades. As previously done, WGM re-modeled the upper geological sub-units of the Lac Otelnuk iron formation that were previously defined (2a, 2b, 2c, 3a and 3b), retaining the transitional 2b-c sub unit identified for the 2012 estimate. WGM also added an internal shale waste unit north of the old Main Zone, starting at about Line 30S. This waste unit is better defined with additional drilling and becomes more prominent and thicker to the north.

The 2013 Mineral Resource estimate included the new drilling results from the 2012 exploration program (an additional 157 holes from the 2012 Mineral Resource estimate). These holes were completed primarily as infill drilling in the north part of the previously defined Mineral Resource area to upgrade the categorization of the resources and to extend the up-dip mineralization to surface along the western margin of the mineralization. The 2013 estimate used the results from a total of 370 drillholes. A cut-off of 18 % DTWR was determined to be appropriate at this stage of the project (Table 14.6). This cut-off was chosen based on a preliminary review of the parameters that would likely determine the economic viability of a large open pit operation and compares well to similar projects in the area that are currently at a more advanced stage of study. This cut-off was the same as used for the previous Mineral Resource estimates.

Table 14.6 – 2013 Categorized Mineral Resource Estimate for Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

Resource Classification	Tonnes (in billions)	TFe Head %	DTWR %	Magnetic Fe %
Measured	16.21	29.3	25.8	17.8
Indicated	4.43	31.5	24.1	16.7
Total M&I	**20.64**	**29.8**	**25.4**	**17.6**
Inferred	6.84	29.8	26.3	17.8

Notes:
1. Interpretation of the mineralized zones were created as 3D wireframes/solids based on logged geology and a nominal 10 % DTWR when required.
2. Mineral Resources were estimated using a block model with a block size of 50m x 50m x 5m.
3. No grade capping was done. Tonnages and grades reported above are undiluted.
4. Assumed Fe price was US$ 110/dmt.
5. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues;
6. The quantity and grade of reported Inferred Mineral Resources in this estimation 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 upgrading them to an Indicated or Measured Mineral Resource category;
7. The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.

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.

Table 14.7 shows the Mineral Resource estimate at various cut-offs for comparison purposes.

Table 14.7 – Categorized Mineral Resources by %DTWR Cut-Off
Lac Otelnuk Iron Project

	Tonnage (in billions)	TFe Head %	DTWR %	Magnetic Fe %
No DTWR Cut-Off (all mineralization within the wireframe sub-units)				
Measured	19.15	29.2	24.1	16.6
Indicated	5.72	31.1	22.2	15.4
Inferred	10.33	29.1	21.2	14.4
12 % DTWR Cut-Off				
Measured	18.56	29.3	24.6	16.9
Indicated	5.65	31.2	22.4	15.5
Inferred	8.52	29.6	24.2	16.4
15 % DTWR Cut-Off				
Measured	17.88	29.3	25.0	17.2
Indicated	5.38	31.2	22.8	15.8
Inferred	7.91	29.8	25.0	17.0
18 % DTWR Cut-Off				
Measured	16.21	29.3	25.8	17.8
Indicated	4.43	31.5	24.1	16.7
Inferred	6.84	29.8	26.3	17.8
20 % DTWR Cut-Off				
Measured	14.52	29.4	26.6	18.3
Indicated	3.65	31.7	25.2	17.5
Inferred	6.06	29.8	27.2	18.5
22 % DTWR Cut-Off				
Measured	12.54	29.4	27.5	19.0
Indicated	2.80	32.0	26.5	18.4
Inferred	4.99	29.9	28.6	19.4

Table 14.8 shows the tonnage and grades in the three Mineral Resource categories for the sub-units at 18 % DTWR cut-off.

Table 14.8 – Categorized Mineral Resources by Sub-Unit
Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

	Tonnage (in millions)	TFe Head %	DTWR %	Magnetic Fe %
Measured				
Sub-unit 2a	1,649	31.0	31.0	21.4
Sub-unit 2b	3,539	34.1	25.3	17.5
Sub-unit 2b-c	1,519	26.5	23.6	16.3
Sub-unit 2c	4,397	27.8	23.5	16.2
Sub-unit 3a	856	26.3	21.9	15.0
Sub-unit 3b	4,247	27.8	28.3	19.5
Total Measured	**16,207**	**29.3**	**25.8**	**17.8**
Indicated				
Sub-unit 2a	497	29.2	25.4	17.6
Sub-unit 2b	2,192	34.7	24.6	17.1
Sub-unit 2b-c	530	27.7	23.6	16.3
Sub-unit 2c	838	28.7	21.1	14.6
Sub-unit 3a	28	26.1	21.4	14.7
Sub-unit 3b	344	27.1	27.6	18.6
Total Indicated	**4,429**	**31.5**	**24.1**	**16.7**
Inferred				
Sub-unit 2a	1,509	30.5	30.6	20.8
Sub-unit 2b	2,094	33.1	24.6	16.9
Sub-unit 2b-c	841	27.9	26.1	16.8
Sub-unit 2c	1,407	27.1	23.9	16.3
Sub-unit 3a	127	27.0	21.2	14.4
Sub-unit 3b	866	27.4	27.9	19.1
Total Inferred	**6,844**	**29.8**	**26.3**	**17.8**

Note: Numbers may not add up due to rounding

14.8 Block Model Validation

The validation of the Lac Otelnuk Mineral Resource estimate was carried out separately in two steps.

14.8.1 Visual Comparison

The visual comparison of block model grades with composite and raw assay grades shows a reasonable correlation between the values. No significant discrepancies were apparent from the cross sections and level plans reviewed. The interpolated grades on cross sections follow more or less the projection angles defined by the search ellipsoid which was oriented along the average dip of the sub-units. It is possible that refining the search ellipsoid orientation by adding an additional sub-domain in areas of steeper dip (particularly at depth) may provide an improvement in the local grade distribution, but

WGM does not believe it will have a material effect on the Mineral Resource estimate. The global validation of the block model results against the grade of the assay and composite intervals were confirmed using this visual comparison.

14.8.2 Statistical Comparison of Average Grades

For the second step, the average of the block grades were reported at 0 % TFe cut-off with blocks in all classifications summed. This average is the average grade of all blocks within the mineralized domains. The values of the interpolated grades for the block model were compared to the average grade of head assays and average grade of composites of all samples within the modeled domains (Table 14.9).

Table 14.9 – Comparison of Average Grade of Assays and Composites with Total Block Model Average Grades

	TFe Head %	DTWR %	Magnetic Fe %
Raw Assays	29.2	20.9	14.3
3 m Composites	29.2	20.8	14.2
Block Model	29.3	22.9	15.7

The comparisons above show the average grade of all the blocks in the constraining domains to be in close proximity of the average of all assays and composites used for grade estimation, and any variances observed were not considered to be material and can be explained by geological reasoning. This summary also indicates that there is no bias between the raw assays and the composited assays used for the grade modeling.

14.9 Conclusions

- WGM is satisfied that sampling and assaying for Adriana's programs since 2007 have been performed well and have been effective leading to the generation of a data set sufficient in quality to support the Mineral Resource estimate;
- Specific gravities for the 2013 Mineral Resource estimation of tonnage were completed using a variable density model based on the relationship generated by WGM between %TFe and measured densities, as WGM determined that a variable density model would more accurately define the local variations based on grade rather than using an average density on a per sub-unit basis;
- As with the previous Mineral Resource estimate, WGM built a relationship between the magnetic Fe determined by Satmagan and that determined by DT where both techniques were used to account for the changeover to Satmagan measurements to replace Davis Tube results during the most recent assaying programs. For consistency with previous Mineral Resource estimates, a %DTWR cut-off was retained based on this relationship. A Magnetic Fe% value was determined for each block (using inverse distance technique) and this is reported in the current Mineral Resource estimate along with the DTWR%;
- WGM re-modeled the upper geological sub-units of the Lac Otelnuk iron formation that were previously defined (2a, 2b, 2c, 3a and 3b) and retaining the transitional 2b-c

sub unit identified in the 2012 estimate. A new internal shale waste unit was also defined in the northern part of the Property. Internally, the continuity of the sub-units was excellent, so WGM had no issues with extending the interpretation beyond 600 m distance. This extension was taken into consideration when classifying the Mineral Resources and these areas were given a lower confidence category;

- The 2013 Mineral Resource estimate included the new drilling results from the 2012 exploration program (an additional 157 holes from the 2012 Mineral Resource estimate) and used results from a total of 370 drillholes. A cut-off of 18 % DTWR was used to determine the Mineral Resources (as summarized in Table 14.10) and was the same cut-off used for the previous Mineral Resource estimates:

Table 14.10 – 2013 Categorized Mineral Resource Estimate for Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

Resource Classification	Tonnes (in billions)	TFe Head %	DTWR %	Magnetic Fe %
Measured	16.21	29.3	25.8	17.8
Indicated	4.43	31.5	24.1	16.7
Total M&I	**20.64**	**29.8**	**25.4**	**17.6**
Inferred	6.84	29.8	26.3	17.8

- The drilling programs have illustrated that the iron formation units have excellent continuity of geology/geometry and TFe grades, with the magnetic Fe grades being more variable due to changes in the magnetite/hematite ratio within the sub-units. The average thickness of the units does not significantly change in the main part of the deposit, but are more variable to the north and south. There appears to be some structural complexity to the northeast of the deposit where possible thrusting has occurred but this was not further explored during the 2013 drilling program as it was not the focus of the campaign.

15.0 MINERAL RESERVE ESTIMATES

The Mineral Reserves for the Lac Otelnuk deposit were prepared by Jeffrey Cassoff, Eng., Lead Mining Engineer with Met-Chem Canada Inc. and a Qualified Person. The Mineral Reserves have been developed using best practices in accordance with CIM guidelines and National Instrument 43-101 reporting. The effective date of the Mineral Reserve estimate is March 25th, 2015.

The Mineral Reserves were derived from the Mineral Resource Block Model that was presented in Section 14. The Mineral Reserves are the Measured and Indicated Mineral Resources that have been identified as being economically extractable and which incorporate mining loses and the addition of waste dilution. The Mineral Reserves form the basis for the mine plan presented in Section 16.

15.1 Geological Information

The following Section discusses the geological information that was used for the mine plan and Mineral Reserve estimate. This information includes the geological block model, the topographic and lithological surfaces and the material properties for ore, waste and overburden.

The mine planning work carried out for the Feasibility Study was done using MineSight® Version 8.5. MineSight® is a commercially available mine planning software that has been used by Met-Chem for over 30 years.

15.1.1 Topographic Surface

The mine design for the Feasibility Study was carried out using a topographic surface based on two (2) m contour intervals that were supplied to Met-Chem by SLI.

15.1.2 Resource Block Model

The mine design is based on the 3-dimensional geological block model that was prepared by WGM and presented in Section 14. Each block in the model is 50 m wide, 50 m long and 5 m high. The model is rotated 324.5°. Each block in the model has been assigned a value for the following items:

- Rock Type.
- Davis Tube Weight Recovery (DTWR).
- Percent Head Fe.
- Percent Magnetic Fe.
- Density.
- Resource classification of the block (Measured, Indicated and Inferred).

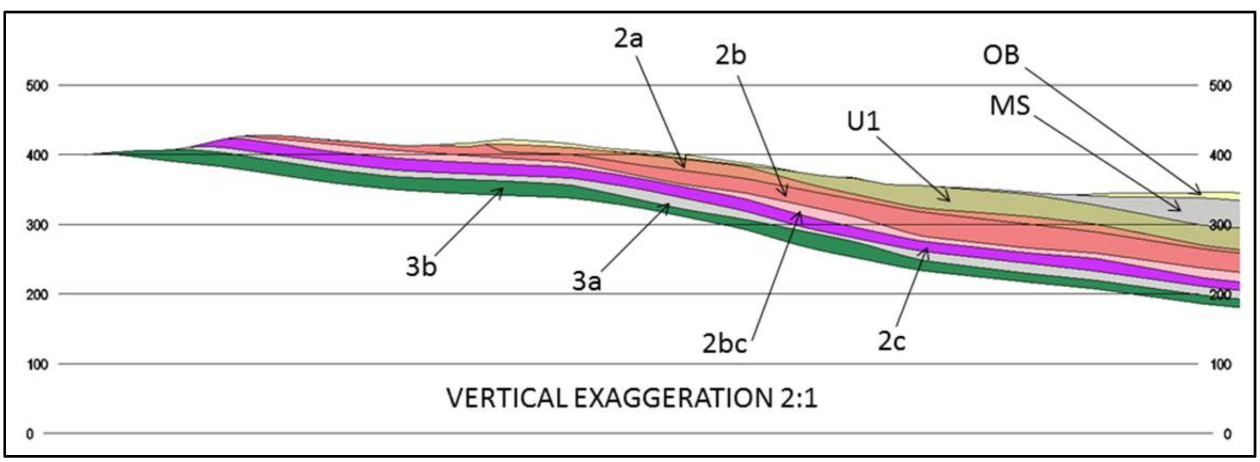
15.1.3 Lithological Surfaces

The iron formation in the Lac Otelnuk deposit is composed of a series of strata or seams, as presented in Figure 15.1. Each seam was modelled as a wireframe solid by WGM from the exploration drill holes by mapping changes in the lithology. These solids were supplied to Met-Chem, as were the solids representing the following waste formations:

- Overburden (OB) – The iron formation is covered by a layer of loose sand and gravel that is referred to as overburden. The overburden reaches a maximum thickness of 10 m and averages 4 m over the deposit area.
- Menihek shale (MS) – Towards the east end of the deposit, a rock formation known as the Menihek shale begins to appear. This formation lies between the overburden and the iron formation. The Menihek shale is considered a waste rock since it does not contain any iron mineralization. Since Menihek shale has the potential to produce acid rock drainage (ARD), the open pit design for the Feasibility Study does not include this type of waste rock.
- Upper Cherty carbonate (U1) – The U1 layer also appears on the east side of the deposit and is present between the Menihek shale and the top of the iron formation. The U1 formation has a very low concentration of magnetite and was therefore modelled as a waste rock.
- Shale waste unit – Towards the north end of the deposit, a shale unit begins to appear between Unit 2 and Unit 3. The shale unit which is barren of mineralized material was not included in the open pit design for the Feasibility Study.

Due to inconsistencies noted with the core logging of the 2012 drilling campaign, the 2bc layer has been grouped with the 2b layer for the purposes of mine planning for the Feasibility Study.

Figure 15.1 – Lithological Surfaces (Typical Section)



15.1.4 Material Properties

The material properties for the different rock types are outlined below. These properties are important in estimating the Mineral Reserves, the equipment fleet requirements as well as the rock pile and stockpile design capacities.

a) Rock Density

In order to estimate the density, WGM used the gas comparison pycnometer method as well as bulk density measurements on half split core by the weighing in air and weighing in water method. These results were modelled to obtain a correlation between the specific gravity and total head Fe grade. Using this correlation, WGM assigned a density to each block in the model, which is a function of the head Fe grade for that block.

Table 15.1 presents the average density for the Measured and Indicated Mineral Resources for each lithology. The densities presented are based on the in-situ dry value. The average density of the Measured and Indicated Mineral Resources is 3.36 t/m^3. The average bulk density (blasted rock) for the run of mine ore is 2.36 t/m^3. This value has been calculated using a swell factor of (45 %) and a moisture content of (2 %) which is presented in the following sections of this document.

Table 15.1 – Density by Rock Type

Rock Type	Description	Density (t/m^3)
2a	Cherty magnetite	3.39
2b	Jasper magnetite-hematite	3.46
2c	Banded Cherty magnetite	3.30
3a	Cherty carbonate	3.24
3b	Cherty magnetite	3.29

For the density of the overburden, an in-situ value of 2.1 t/m^3, a typical value for similar projects in the region was used.

As previously mentioned, the Menihek shale and the shale waste unit at the north end of the deposit were not included in the current design of the open pit. For this reason, their densities are not a concern for this Study and have therefore not been estimated.

For the density of the U1 (Upper Cherty carbonate) waste formation, an in-situ density of 3.19 t/m^3 was used in the Study. This is based on 33 samples that were measured for density using the pycnometer method.

b) Swell Factor

The swell factor reflects the increase in volume of material from its in-situ state to after it is blasted and loaded into a haul truck. A swell factor of 45 %, a typical value for open pit hard rock mines, was used for the Feasibility Study. The swell factor is reduced to 30 % when the material is placed in piles due to compaction.

c) Angle of Repose

An angle of repose of 38° was used for blasted rock. This value is typical for open pit hard rock mines and is used for the design of the rock piles and stockpiles.

d) Moisture Content

The moisture content reflects the amount of water that is present within the rock formation. The moisture content is important when estimating the truck requirements since this quantity of water must be considered during truck payload calculations. The moisture content is also an important factor for the process water balance.

A moisture content of 2 % was used for the Feasibility Study. This value is typical for similar projects in the Labrador Trough.

15.2 Pit Optimization

The first step in the Mineral Reserve estimate is to carry out a pit optimization analysis to determine the parts of the Mineral Resources that are economical to mine. Pit optimization takes place at the start of the study and uses initial assumed operational costs and product selling prices to estimate the economics of mining and processing each block in the model. The pit optimization identifies the limits of the pit and the depth at which the mining costs outweigh the benefits of processing and selling the concentrate.

Since the Lac Otelnuk deposit is close to the surface and contains very little waste rock, the pit optimization analysis showed that the entire resource is economical to mine and to process. It should be noted that pit optimization analyses are strictly based on operating costs and do not consider the capital cost component of a project.

15.3 Open Pit Design

The following section presents the parameters that were used for the design of the open pit. It is important to note that since the Project is at a feasibility study level, only the Measured and Indicated Mineral Resources can be considered. In order to comply with NI 43-101 guidelines regarding the Standards of Disclosure for Mineral Projects, the Inferred Mineral Resources are considered as waste rock for the Feasibility Study.

15.3.1 Mine Life

Although the 20,640 Mt of Measured and Indicated Mineral Resources are sufficient for a 105 year mine life at an annual production rate of 50 Mt of concentrate, it was decided early in the Project that the Feasibility Study would be limited to a 30 year mine life. The mine life was limited since a market study cannot be reliably make prediction for the period of 105 years and also because the cash flows generated beyond 30-years have little impact on the internal rate of return ("IRR"), and payback period of a Project.

The open pit for the Feasibility Study must therefore contain 1,325 Mt of concentrate. This is based on the 225 Mt of concentrate that will be produced during the first eight (8) years, followed by 22 years at 50 Mt/y of concentrate. The phased production ramp-up is presented in more detail in Section 16 of this Report.

15.3.2 Open Pit Location

The location for the 30-year open pit was determined to achieve the following objectives:

- Mine the Mineral Resources that are closest to the concentrator
- Mine the Mineral Resources that have a low waste-to-ore stripping ratio
- Mine the Mineral Resources that have a high weight recovery
- Mine the Mineral Resources that were estimated based on a high density of exploration drilling
- Avoid disturbing as many water bodies as possible to limit the hydrological footprint of the open pit.

Figure 15.2 presents a plan view showing the variability of the DTWR across the deposit and Figure 15.3 presents the variability of the strip ratio. These figures were used to help best locate the limits of the 30 year open pit.

The determined limits of the 30 year open pit incorporate the following additional constraints:

- Since the Menihek Shale has the potential to produce ARD, it was ensured that the current pit design did not include any of this rock type. This constraint ultimately defined the easternmost limit of the pit. The presence of the Menihek Shale was not the only reason for avoiding the eastern part of the deposit. Coincidentally, the areas that contain the Menihek Shale also happen to have a higher strip ratio
- Since no geochemical test work has been done on the Shale Waste Unit at the north end of the deposit, which was only identified in the most recent resource estimate, it was also decided to avoid areas that contain this rock type in the pit design. If geochemical test work identifies that the shale waste unit is not a producer of ARD, the pit should be redesigned to include this area in the next phase of the Project.
- The open pit must avoid the claim that falls within the Mineral Resource area and is held by a party other than LOM. A 15-m wide corridor has been designed to provide access to this claim from the east
- In order to maximize the storage capacity of the TMF and to minimize dyke construction requirements, a small part of the TMF will cover a portion of the Mineral Resource area. In this area, the TMF will reach an elevation of 354 m. In order to ensure that the tailings do not infiltrate into the open pit, the limit of the pit in this area was be designed to be a minimum of 200 m away from the 355 m contour. This distance accounts for a 1-m freeboard plus a corridor that is wide enough to place infrastructure such as a mine-haul road, mine electrical-power lines and water return pipelines from TMF. Approximately 60 Mt of ore at an average weight recovery of 28 % have been excluded from the current pit design as a result of this constraint. The exclusion of these Resources from the pit design should be further evaluated in the next phase of the Project.

Figure 15.2 – Davis Tube Weight Recovery Variability

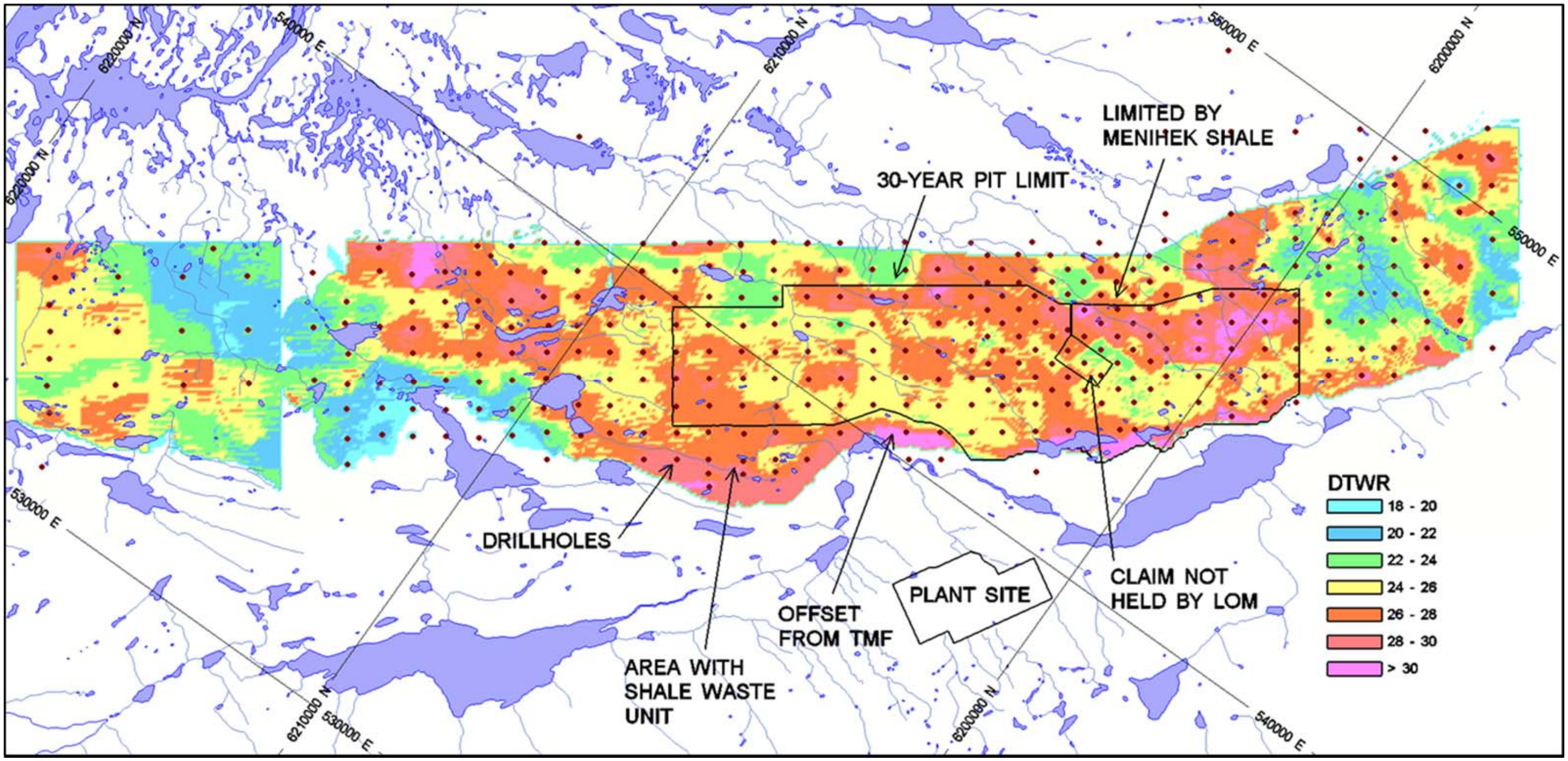
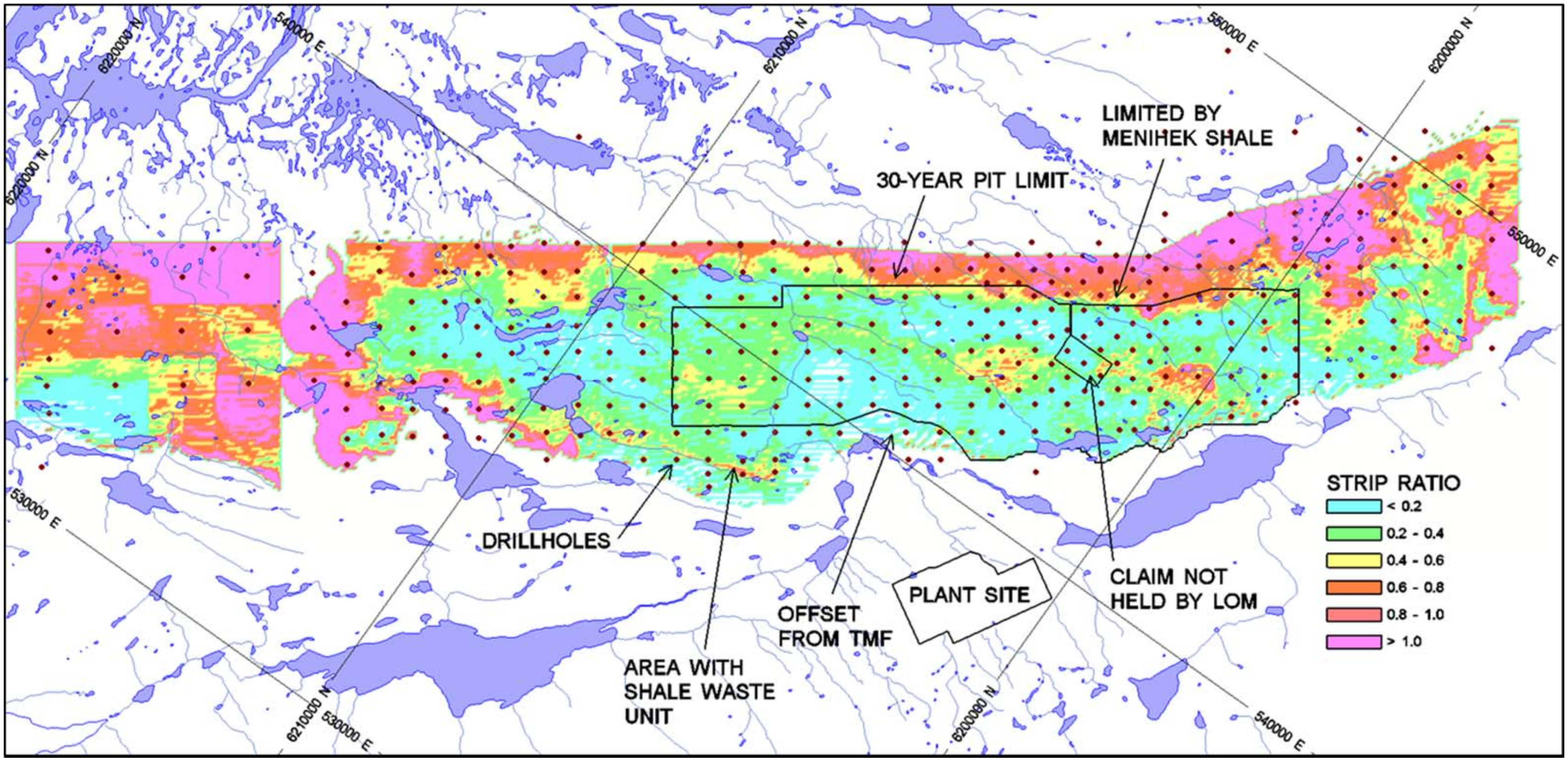


Figure 15.3 – Strip Ratio Variability



15.3.3 Cut-off Grade

An analysis was carried out to determine the appropriate cut-off grade for the Feasibility Study. A high cut-off grade means that a considerable amount of low grade material must be mined and stockpiled, resulting in a high operating cost for the mine. However, a high cut-off grade increases the average grade of the ore, which reduces the infrastructure required in the concentrator and reduces the operating cost of processing. A low cut-off grade has the opposite effect for both the mine and concentrator.

The analysis showed that the cut-off grade for the Feasibility Study should be at a DTWR ≥ 18 %. This implies that any block in the resource model that falls within the 30 year pit area and has a DTWR value < 18 % is considered as low grade material and is stockpiled rather than processed.

The cut-off grade was adjusted to 20.65 % following the inclusion of mining dilution.

15.3.4 Geotechnical Pit Slope Parameters

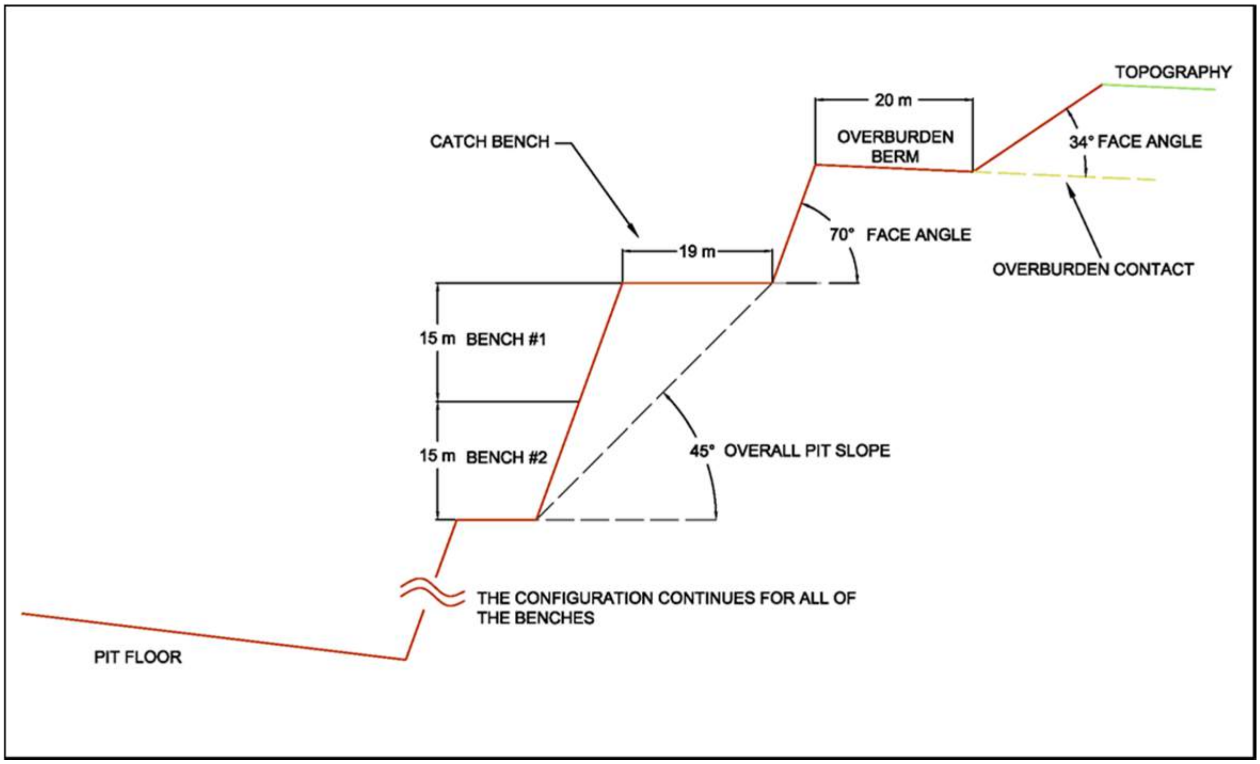
During the summer of 2013, a geotechnical investigation which included seven (7) inclined drill holes was carried out by Golder Associates Ltd. ("Golder") to characterize the rock mass of the Lac Otelnuk deposit. The results of this investigation were presented in Golder's report titled "Complementary Hydrogeological and Geotechnical Fieldwork Summer 2013". The conclusion of the report included the following statement: "the assessment of the rock masses at the Lac Otelnuk Project open pit has indicated Good to Very Good quality rock masses in terms of degree of fracturing and intact rock strength".

Since the configuration of the final pit wall has very little effect on the amount of waste rock that will be mined to access the ore body, a conservative overall slope of 45° was used for the final pit wall configuration for the Feasibility Study. This is a very stable slope for a competent rock mass. The pit will be mined with 15 m high benches, as shown in Figure 15.4 which presents the pit wall configuration. This bench height is well suited for the size of equipment (shovels and drills) that are planned for the mine.

The overall slope of 45° will be achieved by incorporating a 19 m wide catch bench for each 30 m in height (two,15 m benches), with a face angle of 70°. A width of 19 m for the catch benches is sufficient to maintain access to these benches with track dozers and pickup trucks at all times. A shallower pit slope of 34° will be used through the overburden formation since this material is less stable than the bedrock. A 20 m wide berm will be left at the contact between the overburden and bedrock.

The east wall is planned to be the highest wall of the pit, reaching a maximum height of approximately 130 m. For the most part, the pit will not have a west wall, as the iron formation is exposed directly at the surface on this side of the pit. The heights of the northern and southern walls increase gradually from null on the west side of the pit to the maximum height of approximately 130 m at the east wall. There will be no access ramp along the final pit wall since trucks will access the pit from the west side.

Figure 15.4 – Pit Wall Configuration



15.3.5 Haul Road Design

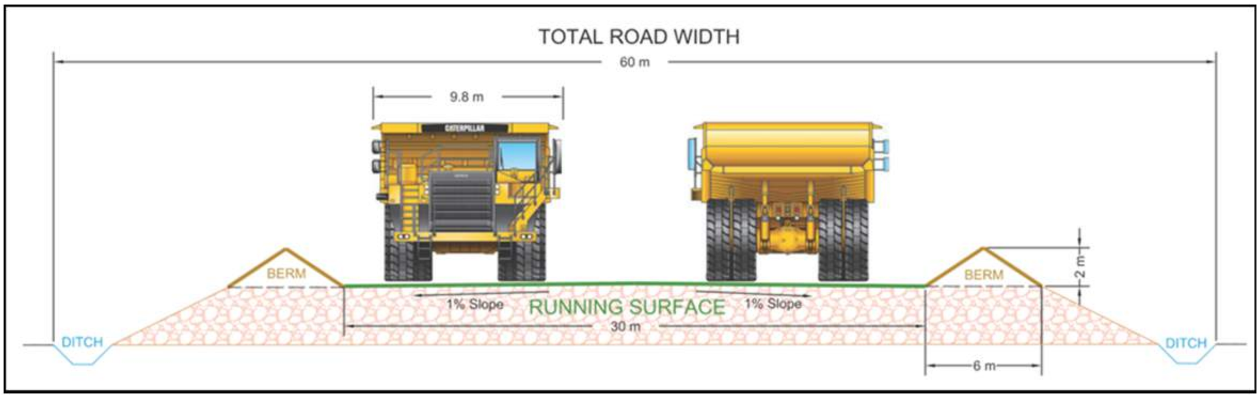
Since the iron formation is exposed directly on surface along the west side of the open pit, there is no need to include an access ramp in the design. The pit access will be developed along the bottom of the floor as the pit wall advances to the east. The pit floor will follow the bottom of the 3b lithology, which dips at approximately 5°. This translates in a grade of 8.7 %, which is well within the range for mining haul trucks.

Temporary ramps will be required in order to maintain access to the benches in the advancing wall. These ramps will either be cut with the shovels or backfilled with mined out waste rock or low grade material. The temporary ramps will be designed with a maximum grade of 8 %.

The mine haul roads have been designed for a 400-ton (short ton) haul truck. For 2-lane traffic, industry practice indicates the running surface width to be a minimum of three (3) times the width of the largest truck. The overall width of a 400-ton haul truck is 9.8 m, which results in a running surface of 30 m. The overall width of the haul road must account for safety berms. The following dimensions of the safety berms are based on industry standards: the safety berm height should be a minimum of half the height of the largest truck tire. The radius of a 400-ton haul truck's tire is 2 m. The safety berm slopes are 34° and will be built in a triangular shape. The width required for the bottom of each safety berm is 6 m. The overall width of the road including safety berms is 42 m.

For the haul roads outside of the pit, a corridor of 60 m should be used in order to ensure that there is enough room for the ditches and the road base side slopes if fill material is required for construction. The maximum road grade should be 8 % and the road design should incorporate a crown of at least 1 %. The berms should be interrupted every 25 m in length to allow for water to run-off into the ditches. Figure 15.5 presents a typical section of the haul road.

Figure 15.5 – Haul Road Configuration



15.3.6 Mine Dilution and Ore Loss

In a productive mining operation, it is typical for contamination to occur between the ore and waste at the contact boundaries. This is due to the nature of the large size of shovels and the fact that the rock requires blasting. Mine dilution occurs when waste rock is mixed with the ore and sent to the processing plant. Mine dilution has the effect of increasing the ore tonnage while reducing the ore grade. Ore losses occur when ore is mixed with the waste rock and sent to the rock pile. This ore is never processed, thus reducing the mineral reserves.

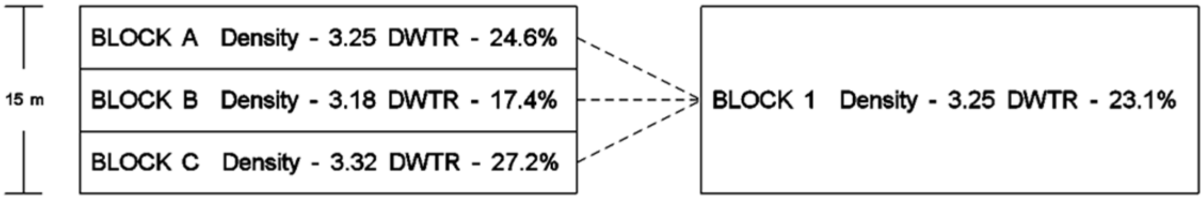
Two (2) areas have been identified where mining dilution and ore losses are expected to occur during the mining of the Lac Otelnuk deposit:

- Impossible to separate waste bands that appear within the 15 m high benches
- Major contacts between the overburden, U1 waste unit and the pit floor.

a) Waste Bands within the 15 m Benches

The mineral resource estimate for the Lac Otelnuk deposit is based on 5 m high blocks. Since the mining operation will be carried out with 15 m high benches using drilling and blasting techniques, it will be impossible to separate specific blocks within the mining face. In order to account for this, Met-Chem has created a mine planning block model where the three (3) 5 m high blocks for each bench have been combined into one (1) 15 m high block. The density and grades of the new block are weighted averages of the densities and grades for the three (3) individual blocks. Figure 15.6 illustrates how the combination of resource blocks into a 15 m high mining block creates mining dilution. In this specific example, since Block B has a DTWR of less than 18 %, it is considered as low grade ore in the resource model. In the mine planning model, this block is mined as ore, thus increasing the tonnage but reducing the grade.

Figure 15.6 – Combination of Resource Blocks



b) Major Ore / Waste Contacts

Mining dilution may also occur at the major ore / waste contacts – i.e. where the iron formation is in contact with the overburden and the U1 Waste Unit:

- Overburden – The mining dilution effects from the overburden should be negligible because the overburden is a layer of sand and gravel that does not require blasting and is easily identifiable in the field
- U1 Waste Unit and 3c Unit on the pit floor – The U1 waste unit that occurs at the top of the iron formation and the 3c Unit that is below the pit floor are both lower grade iron formations. Dilution with these layers was not considered since even if they are mined, they do contain a certain amount of mineralization.

In order to maintain the 30-year life of the open pit with an average weight recovery of 26.5 %, the figure used for the design of the concentrator, the cut-off grade for mine planning was increased to DTWR ≥ 20.65 %. The result of accounting for mining dilution and applying this higher mine planning cut-off grade is an ore tonnage decrease of 5.0 %. Material below this cut-off will be directed to the low grade stockpiles.

15.3.7 Mineral Reserves

The 30-year pit design, which is presented in Figure 15.7, is 11.6 km long and 2.8 km wide and has an overall surface area of 2,900 ha. The pit contains 4,993 Mt of ore at an average DTWR of 26.5 %, 180 Mt of overburden, 142 Mt of waste rock and 1,052 Mt of low grade material which is below the cut-off of 20.65 % DTWR. The stripping ratio is 0.28 to 1. Table 15.2 presents a summary of the mineral reserves for the Lac Otelnuk deposit and Table 15.3 presents the mineral reserves by rock type. There are no inferred resources within the open pit.

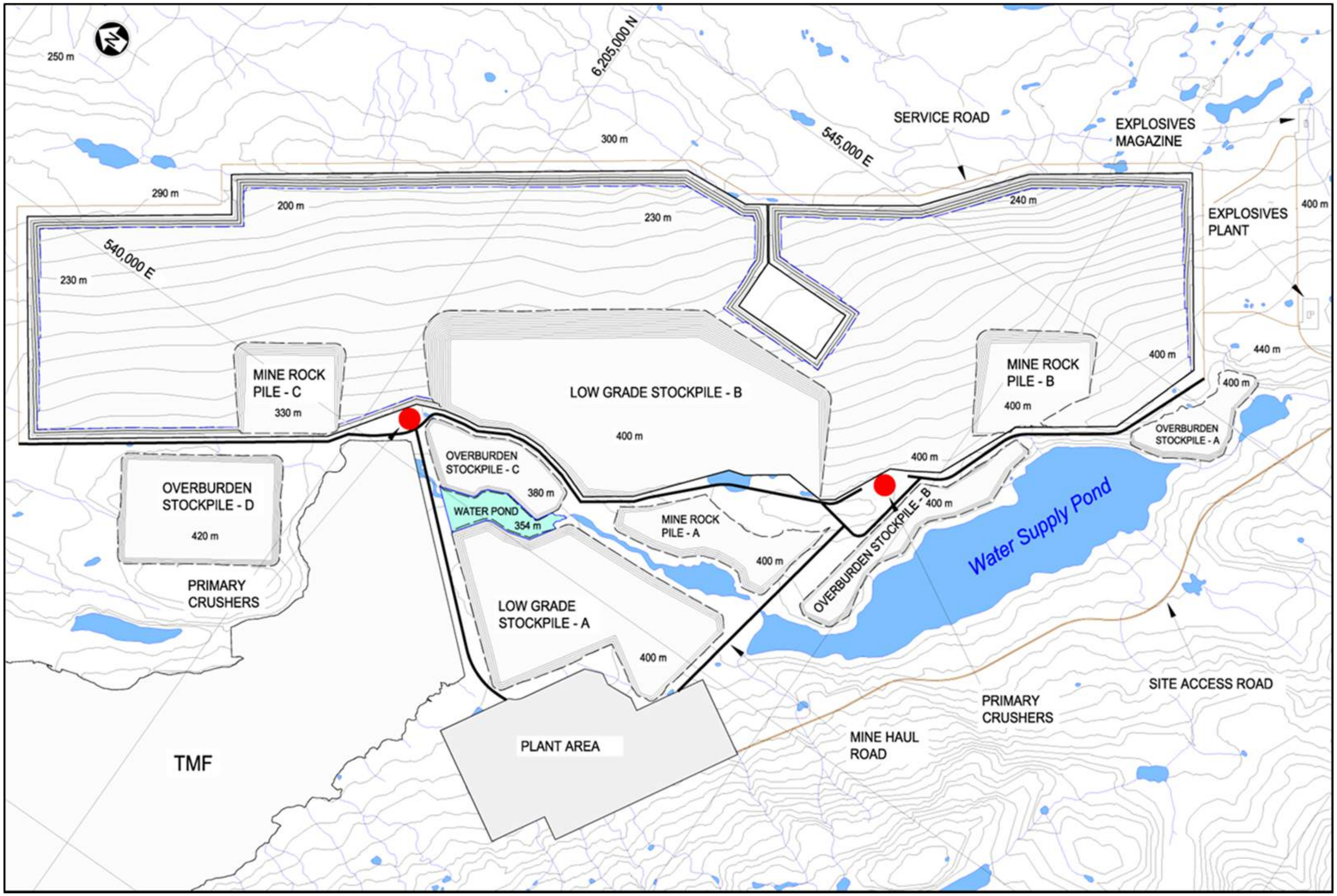
Table 15.2 – Mineral Reserves

Category	Tonnage (Mt)	Total Fe Head (%)	DTWR (%)	Magnetic Fe (%)
Proven	4,943	28.7	26.5	18.3
Probable	50	27.5	26.6	18.3
Total	**4,993**	**28.7**	**26.5**	**18.3**

Table 15.3 – Mineral Reserves by Rock Type

Crock Type	Tonnage (Mt)	Total Fe Head (%)	DTWR (%)	Magnetic Fe (%)
2a	369	33.1	33.6	23.1
2b	926	30.8	26.5	18.2
2c	1,594	27.7	24.3	16.8
3a	417	27.3	23.8	16.4
3b	1,686	27.9	27.8	19.1
Total	**4,993**	**28.7**	**26.5**	**18.3**

Figure 15.7 – Mine Layout



16.0 MINING METHODS

The mining method selected for the Project is a conventional open pit consisting of drilling and blasting and a truck and shovel operation. The designed bench height is 15 m. Due to the high tonnages expected to be mined and the relatively long haul distances, the trucks and shovels selected for the Project will be the largest available on the market.

Trees will be cleared prior to the start of mining. Next, a mining contractor will remove the topsoil and overburden using a fleet of dozers, excavators and haul trucks. The ore and waste rock will be drilled, blasted, and then loaded with large mining shovels into a fleet of rigid frame trucks which will haul the material either to the primary crushers, the low grade stockpiles, or the mine rock piles (for the waste rock).

To properly manage water infiltration into the pit, sumps will be established at the lowest point of the active pit floor. Water collected in these sumps will be pumped to the stream system that flows towards the tailings management facility (TMF). Water management is discussed in more detail in Section 18 of this report.

16.1 Rock Pile and Stockpile Design

This section of the report presents the design and layout for the rock piles and stockpiles which are illustrated in Figure 15.7.

The 30-year open pit for the Lac Otelnuk deposit includes the following five (5) different material types that will be dumped or stockpiled in separate appropriate locations:

- Topsoil and organic materials.
- Overburden.
- Waste rock (U1 unit).
- Low grade material (below the DTWR cut-off of 20.65 %).
- Run of mine ore stockpile next to the primary crushers.

In addition to the specific criteria presented below, the rock piles and stockpiles were designed within claims belonging to Lac Otelnuk Mining. Although there is very limited space on the west side of the deposit, it was ensured that none of the rock piles and stockpiles were designed on areas that have the potential for mineral resources, with the exception of Overburden Stockpile D which will need to be re-evaluated in the next phase of the Project. All of the piles will have a perimeter ditch around the toe to capture run-off, which will be discharged into the streams that flow towards the TMF.

The toes (footprints) of the piles were designed at a minimum distance of 50 m from all infrastructure such as roads, conveyors, power lines and pipelines.

Any waste rock and overburden that is deemed suitable for the construction of roads, tailings dykes and other infrastructure will be used when economical to do so. The storage capacity of each stockpile is presented in Table 16.1.

Table 16.1 – Rock Pile and Stockpile Capacities

Category	Capacity (Mm^3)
Overburden Stockpile A	9
Overburden Stockpile B	18
Overburden Stockpile C	20
Overburden Stockpile D	64
Mine Rock Pile A	14
Mine Rock Pile B	21
Mine Rock Pile C	23
Low Grade Stockpile A	106
Low Grade Stockpile B	309

a) Topsoil and Overburden Stockpiles

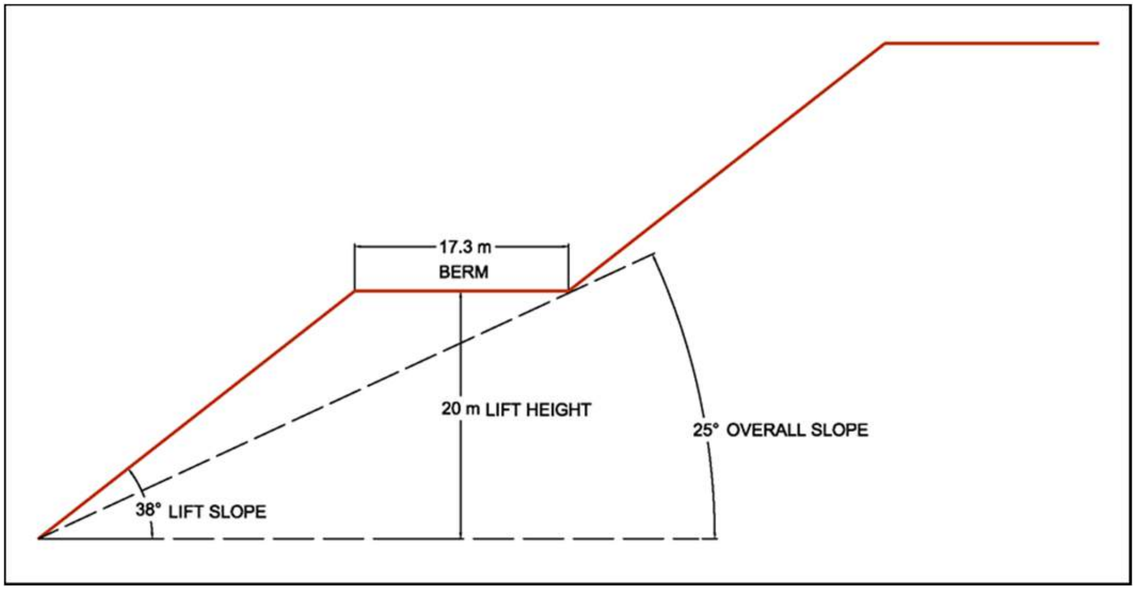
Since the topsoil and organic materials are very sporadic throughout the deposit and since neither is very thick where they do appear, they will be stripped and stockpiled together with the overburden. The topsoil and overburden will be stockpiled separately from the waste rock since they will be used for future reclamation use. The overburden stockpiles were designed with an overall slope of 34°.

Four (4) areas for topsoil and overburden stockpiles have been designated for the Feasibility Study in order to minimize haul distances. The stockpiles were also located close to the sites that will require this material for reclamation. Stockpiles A and B are at the south end of the open pit and Stockpiles C and D are at the north end.

b) Mine Rock Piles

The waste rock (U1 unit) will be hauled to rock piles where it will be permanently stockpiled. The rock piles were designed with an overall slope of 25°. Although the material placed in the rock piles has an angle of repose of 38°, a shallower slope is required to assist with the reclamation as proposed in the closure plan. In order to achieve the 25° slope, the rock pile design incorporates a 17.3 m wide berm for each 20 m lift, as illustrated in Figure 16.1. The Feasibility Study includes three (3) mine rock piles: one (1) is located on the west side of the deposit and two (2) are located inside the pit area, and will be built once the pit floor has been developed.

Figure 16.1 – Mine Rock Pile Configuration



c) Low Grade Stockpiles

The low grade material that will be mined will be stockpiled for potential future processing. The stockpiles were designed with the same 25° overall slope as the mine rock piles to ensure the rehandling of the low grade stockpiles can be done safely.

The Feasibility Study includes two (2) low grade stockpiles: one (1) low grade stockpile that was designed next to the processing plant and one (1) in-pit stockpile that will be built once the final pit floor is developed.

d) Run of Mine Ore Stockpiles

To ensure that the crushing operation can continue in the event that the mine is shut down, each primary crusher will have a run of mine ore stockpile next to it with 24 hours of feed. Each stockpile will have a design capacity of 40,000 m^3 (100,000 t), a maximum height of 10 m and an overall slope of 38° (angle of repose).

e) In-Pit Dumping

As was previously mentioned, a low grade stockpile and two (2) mine rock piles have been designed inside the open pit. These piles will be built once the pit wall has advanced to the east and the pit floor has been developed. In-pit dumping is required since there is very limited space on the west side of the deposit where there are no mineral resources. In-pit dumping is also advantageous since it reduces haulage distances, resulting in a lower operating cost for the mine.

16.2 Mine Planning

The following section discusses the 30-year mine plan for the Feasibility Study. The mine plan was established on an annual basis for the first ten (10) years of production followed by four 5-year periods for the remaining 20 years.

A pre-production period of six (6) months has been included before the start of the operation. This period includes tree clearing, topsoil and overburden removal, mine haul road construction, and the development of the pit for ore production.

The mine plan follows the phased approach of the Feasibility Study, producing 30 Mt/yof concentrate in Phase 1 and 50 Mt/yof concentrate in Phase 2. Each phase includes a production ramp up as presented in Table 16.2.

Table 16.2 – Production Ramp-up

Year	Concentrate (Mtpy)
1	10
2	25
3	30
4	30
5	30
6	30
7	30
8	40
9	50

The amount of concentrate that will be produced was calculated by multiplying the ore tonnage for a given year with the average Davis Tube weight recovery.

One of the goals of the mine plan is to ensure that several different ore types are always in production in order to avoid fluctuations in grade and hardness. Blending of the different ores will occur at the primary crushers as trucks arrive from different shovels that are mining the different ore types. Blending will also happen directly at the shovel face since several ore types may be present within a given 15 m high bench.

In order to improve the economics of the Project, mining will begin in the southeastern corner of the open pit where there is a considerable area of high grade ore which is easily identified on Figure 15.2. This high grade zone is limited to the 2a and 2b ore types at the top of the iron formation. In order to ensure the previously discussed blending of ore types, a second area will be developed at the start of the operation on the western side of the deposit, adjacent to the primary crushers. This area will provide the 2c, 3a and 3b ore types. Both of these mining areas will be developed at an even pace for the first seven (7) years of the operation.

Mining will begin in the northern part of the open pit in year 7 to prepare for Phase 2, when the additional two (2) primary crushers will be installed. During Phase 2, 1/3 of the production will come from the northern part of the open pit and 2/3 from the southern part. The purpose of this split in production is to optimize the use of the five (5) primary crushers.

The high grade area in the southeastern corner of the open pit will be abandoned in year 10 when the 2a and 2b ore types are depleted. The equipment in this pit will be relocated to the main pit adjacent to the Phase 1 primary crushers.

The mine production schedule is presented in Table 16.3. The table provides the tonnage to be mined in each period of the mine plan as well as the DTWR and the magnetic Fe. Figure 16.2 presents a plan view showing the sequence of the pit development during the mine plan. Figure 16.3 presents the mine layout at the end of year 10, including the status of the open pit, rock piles, and stockpiles at that point in time.

During the pre-production phase, a total of 2.1 Mt of overburden will be stripped and 4 Mt of ore will be mined. This ore will be stockpiled and rehandled in the first year of the operation. The DTWR throughout the mine plan averages 26.5 % and varies from a high of 33.0 % during year 1 to a low of 25.7 % in year 10. The total material mined ranges from 34.8 Mt in year 1 to a peak of 260 Mt/y for the years 11 to 15. Figure 16.4 shows the tonnage of each material to be mined each year. The tonnages have been annualized for the 5-year periods. Figure 16.5 presents the average DTWR for each year of the mine plan.

In the early stages of the Feasibility Study, an evaluation was carried out to determine the appropriate locations for the primary crushers and whether they should be in fixed locations or relocated throughout the 30 years of the mine plan. It was decided that the crushers should not be relocated for the current design concept. The locations of the crushers were also identified based on the notion of minimizing the haulage distance. Certain other aspects such as conveyor routing were also taken into consideration for the crusher locations.

Table 16.3 – Mine Production Schedule

Description	Units	Pre-Prod	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09	Year 10	Year 11 - 15	Year 16 - 20	Year 21 - 25	Year 26 - 30	Total
Concentrate	**Mt**	**0.0**	**10.0**	**25.0**	**30.0**	**30.0**	**30.0**	**30.0**	**30.0**	**40.0**	**50.0**	**50.0**	**250.0**	**250.0**	**250.0**	**250.0**	**1,325**
Ore to Plant	**Mt**	**0.0**	**30.3**	**85.0**	**103.8**	**104.6**	**104.2**	**106.7**	**109.5**	**152.9**	**191.1**	**194.7**	**975.1**	**950.8**	**927.9**	**956.6**	**4,993**
Rock Type 2a	Mt	0.0	8.1	20.0	21.2	14.5	13.7	11.7	13.6	14.0	4.9	0.0	0.1	75.4	81.4	90.3	369
Rock Type 2b	Mt	0.0	7.1	25.3	39.1	43.1	38.3	42.8	26.1	42.0	37.5	35.1	65.1	195.9	175.5	153.2	926
Rock Type 2c	Mt	0.0	0.0	12.0	22.5	18.7	16.5	16.7	29.9	71.0	52.7	59.5	371.7	365.4	246.2	311.5	1,594
Rock Type 3a	Mt	0.0	0.0	3.7	5.6	5.9	5.7	7.3	8.7	8.5	23.3	15.8	121.0	60.5	82.5	68.8	417
Rock Type 3b	Mt	0.0	15.1	24.0	15.5	22.3	30.0	28.1	31.2	17.4	72.8	84.3	417.2	253.6	342.3	332.7	1,686
DT Weight Recovery	**%**	**0.0**	**33.0**	**29.4**	**28.9**	**28.7**	**28.8**	**28.1**	**27.4**	**26.1**	**26.1**	**25.7**	**25.6**	**26.3**	**26.9**	**26.1**	**26.5**
Fe Total	**%**	**0.0**	**31.3**	**30.0**	**29.9**	**29.6**	**29.4**	**29.2**	**29.1**	**29.1**	**28.4**	**28.1**	**27.5**	**28.9**	**28.9**	**29.1**	**28.7**
Mag Fe	**%**	**0.0**	**22.7**	**20.2**	**19.8**	**19.7**	**19.8**	**19.4**	**18.9**	**18.2**	**18.3**	**17.7**	**17.6**	**18.1**	**18.6**	**18.0**	**18.3**
Ore to Stockpile	**Mt**	**4.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**4.0**
Total Waste	**Mt**	**2.1**	**4.5**	**26.3**	**34.1**	**30.9**	**24.2**	**19.4**	**23.6**	**48.8**	**31.0**	**54.1**	**318.2**	**297.8**	**193.0**	**266.4**	**1,374**
Overburden	Mt	2.1	3.6	9.5	9.5	7.4	7.4	9.7	9.7	9.7	9.7	9.7	25.5	25.5	25.5	15.3	180
Unit 1 Waste Rock	Mt	0.0	0.3	6.0	12.1	14.2	8.9	2.9	4.0	7.2	0.6	0.5	0.9	2.4	37.9	44.2	142
Low Grade	Mt	0.0	0.6	10.8	12.6	9.3	7.9	6.8	9.9	31.9	20.7	43.9	291.9	269.8	129.5	206.9	1,052
Total Material Moved	**Mt**	**6.1**	**34.8**	**111.3**	**137.9**	**135.5**	**128.3**	**126.1**	**133.1**	**201.7**	**222.1**	**248.8**	**1,293.3**	**1,248.6**	**1,120.9**	**1,223.0**	**6,371**
Stripping Ratio		**n/a**	**0.15**	**0.31**	**0.33**	**0.30**	**0.23**	**0.18**	**0.22**	**0.32**	**0.16**	**0.28**	**0.33**	**0.31**	**0.21**	**0.28**	**0.28**

Notes:
- Tonnages are shown on a dry basis
- The Davis Tube (DT) weight recovery accounts for mining dilution
- A cut-off of DT weight recovery ≥ 20.65 % was used to split the ore and low grade.

Figure 16.2 – Mining Sequence

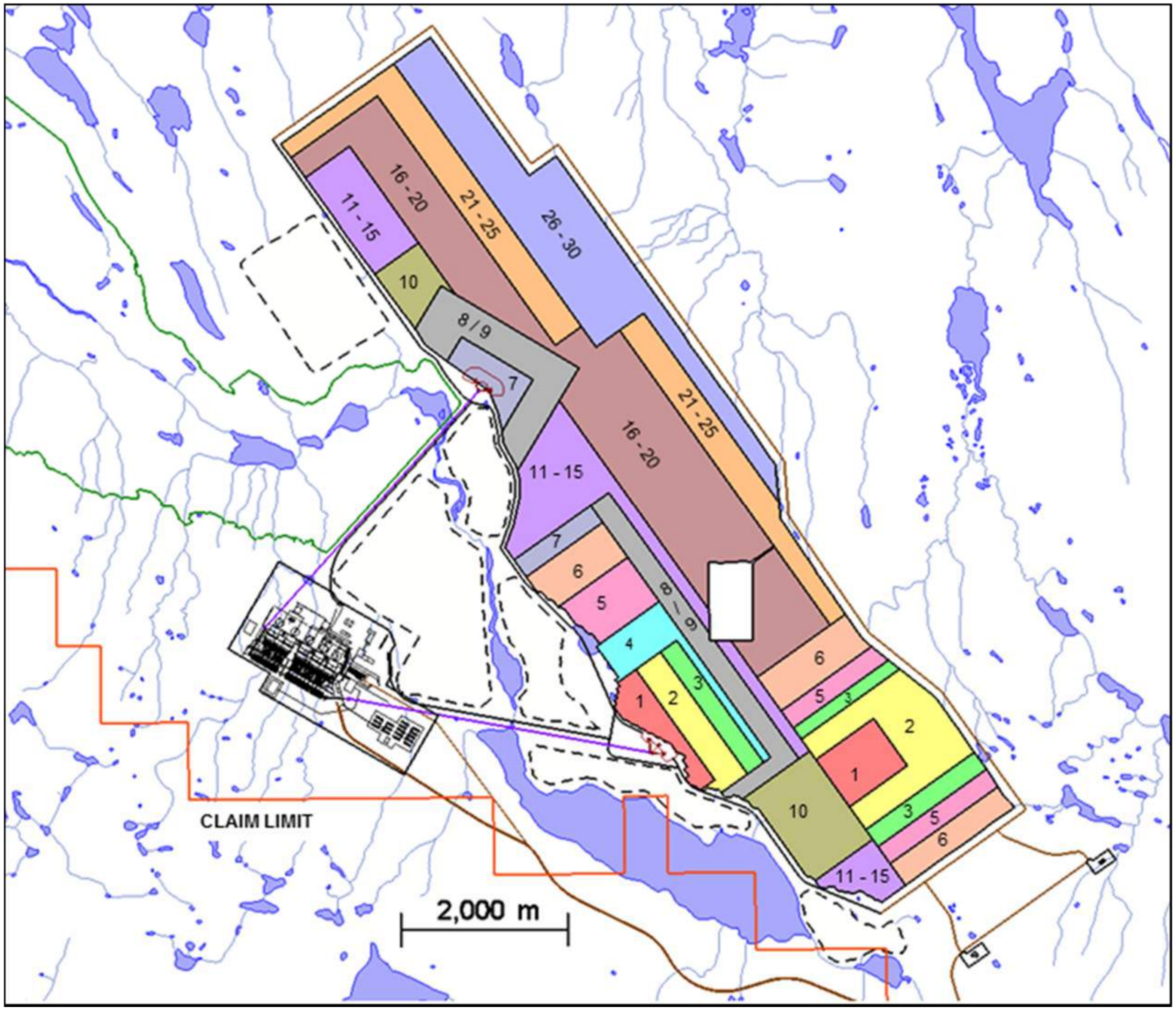


Figure 16.3 – Mine Layout (End of Year 10)

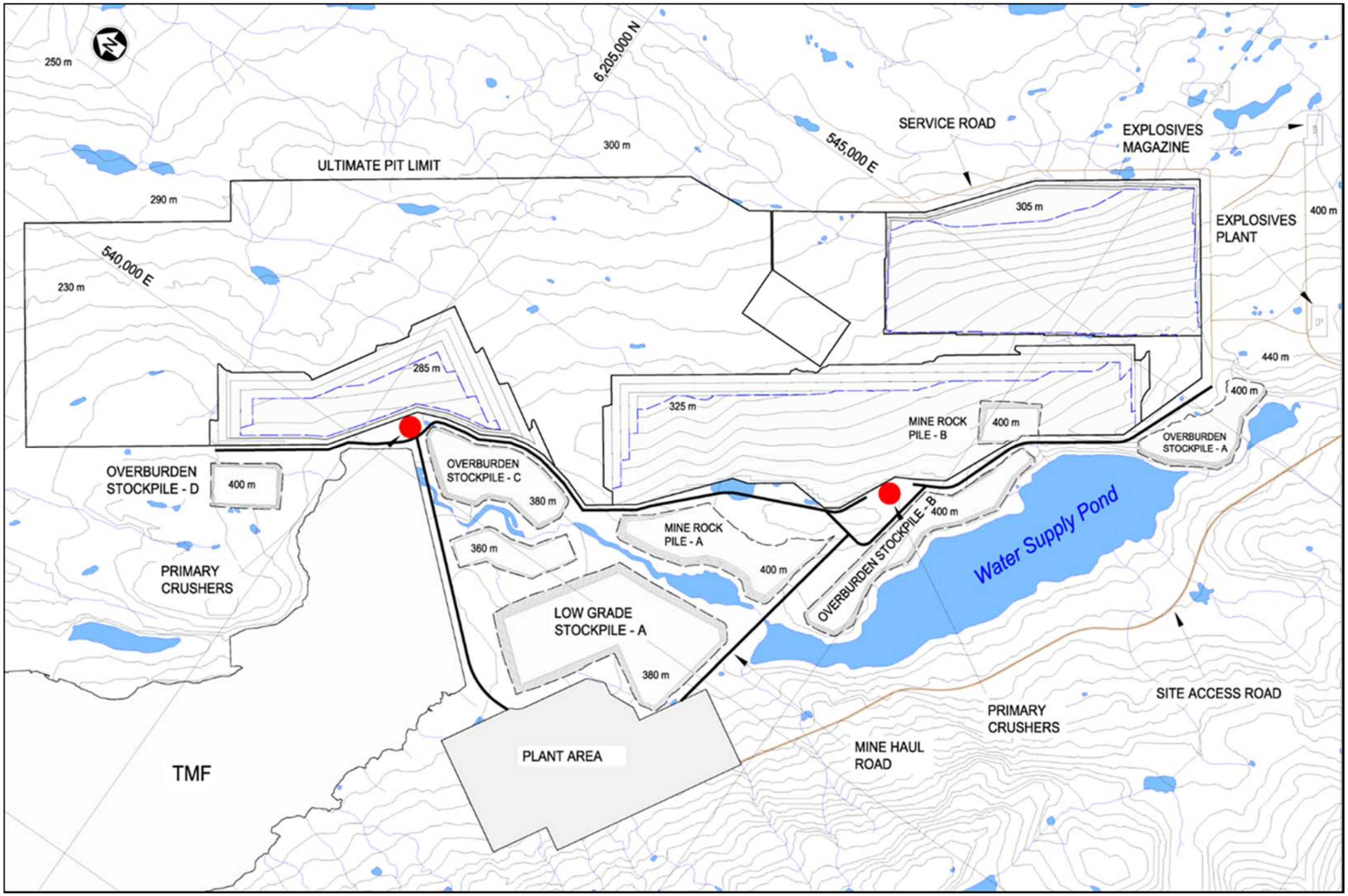


Figure 16.4 – Mine Production Schedule (Production)

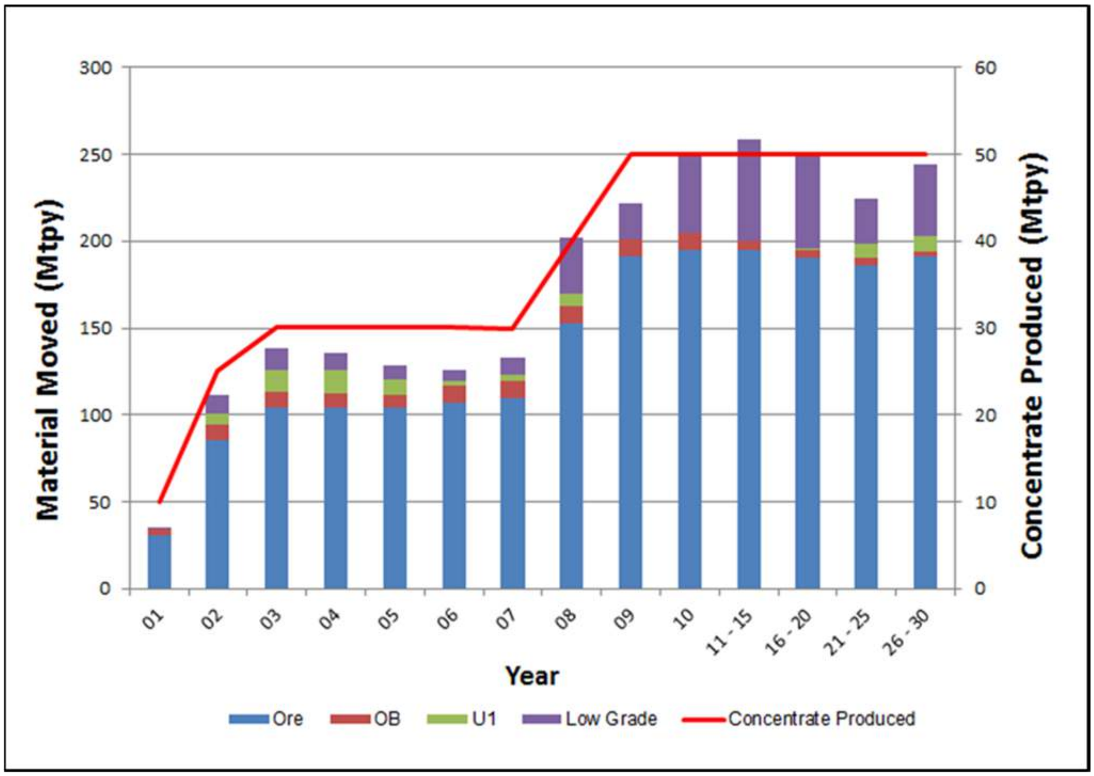
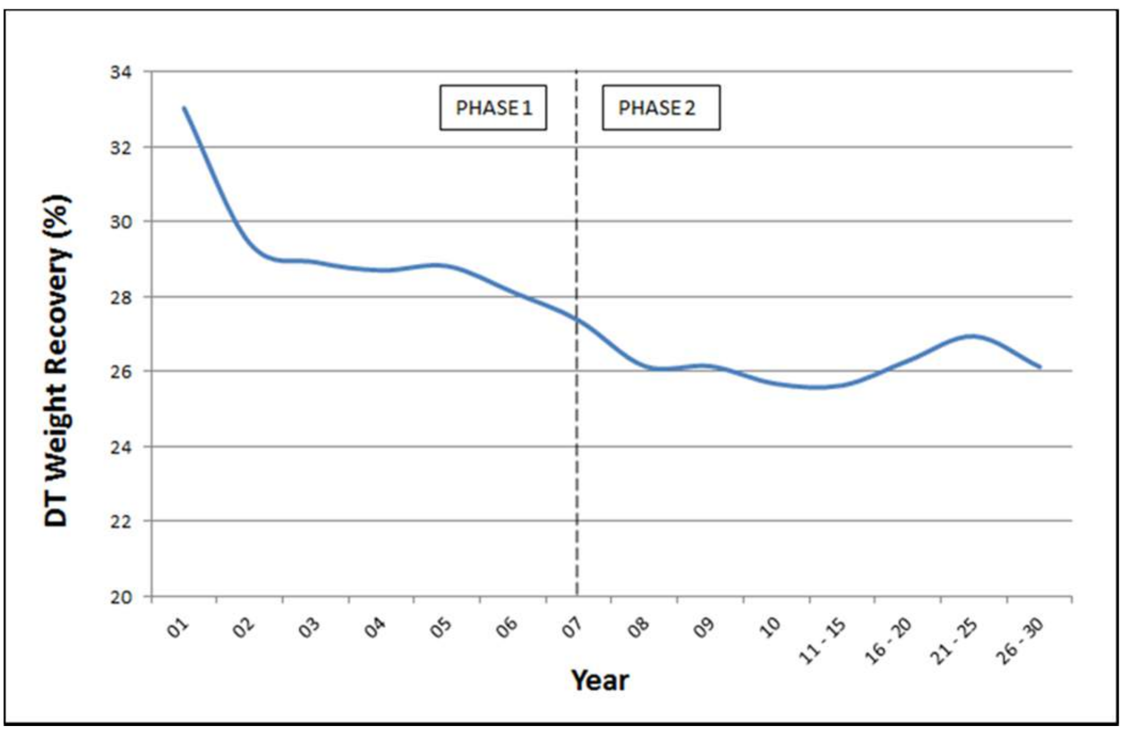


Figure 16.5 – Mine Production Schedule (Grades)



16.3 Mine Equipment Fleet

The following presents the equipment selection and methodology that was used to estimate the fleet requirements for the Feasibility Study. Table 16.4 presents the fleet of major equipment that will be required for each phase. An example model is given for each piece of equipment to give the reader an appreciation for the size, although the specific equipment will be selected during the procurement phase of the Project.

Table 16.4 – Major Mine Equipment Fleet

Description	Phase 1	Phase 2	Total	Major Specification	Example Model
Haul Truck	23	27	**50**	Payload: 363 tonnes	CAT 797F
Cable Shovel	5	5	**10**	Bucket payload: 100 – 110 t	P&H 4100
Hydraulic Shovel	2	0	**2**	Power: 2,900 – 3,360 kW	CAT 6090 FS
Front-End Loader	3	1	**4**	Power: 1,491 – 1,715 kW	Letourneau 2350
Production Drill	8	8	**16**	Bit load: 57,000 – 68,000 kg	P&H 320

Note: The equipment requirements for Phase 2 are in addition to those from Phase 1.

16.3.1 Work Cycle

The mine will operate 365 days per year and around the clock on two 12-hour shifts. Given the northern location, it is assumed that the mine will be shut down during five (5) days every year due to extreme winter conditions. During these periods, the primary crushers will be fed from the run-of-mine ore stockpiles using front end wheel loaders.

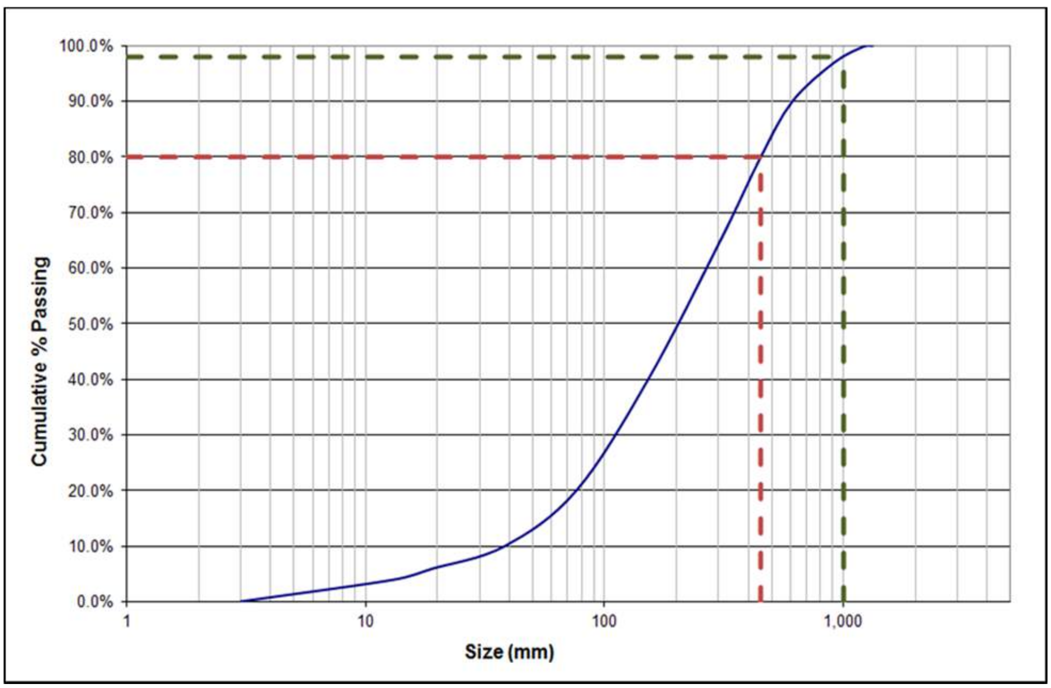
16.3.2 Drilling and Blasting

Production drilling will be carried out using a fleet of electric powered rotary drills following the blast pattern presented in Table 16.5. The blast pattern was determined based on the hardness of the rock and was benchmarked with similar operations. Figure 16.6 presents a curve showing the expected distribution for the rock fragmentation. The curve shows that 98 % of the material fed to the primary crushers is expected to be smaller than 1,000 mm and 80 % is expected to be smaller than 450 mm.

Table 16.5 – Blast Pattern Design

Parameter	Units	Value
Bench height	m	15
Blasthole diameter	mm	381
Burden	m	7.9
Spacing	m	7.9
Sub drilling	m	1.8
Stemming	m	5.0
Explosives density	g/cm3	1.28
Powder factor	kg/t	0.55

Figure 16.6 – Fragmentation Curve



The number of drills required was calculated to be eight (8) during Phase 1 plus an additional eight (8) during Phase 2 for a total of 16. Using the assumptions presented below, each production drill will drill off approximately 4,800 blast holes per year, which equates to 15 Mt of rock. The drill calculations are based on the following assumptions:

- Mechanical availability – A value of 85 % was used which represents the time when the drill is not down for unscheduled breakdowns or preventative maintenance.
- Utilization – A value of 75 % was used. Non-utilized time is accrued when the drill is relocating during blasts and when the preparation of the drill pad is not complete.
- Operating delays – A total of 90 minutes per shift has been allocated to operating delays such as shift change, equipment inspection and lunch / breaks.
- Drilling efficiency – A drilling efficiency of 50 min/h was used which accounts for the time when the drill is repositioning and setting up between drill holes as well as bit and rod changes.
- Penetration rate – A penetration rate of 20 m/h was used for the Feasibility Study. This represents the speed of drilling, which is relatively low due to the hardness of the deposit.

The production drills will be replaced after 50,000 hours of operation (approximately 10 years).

The blasting will be carried out with bulk emulsion which will be manufactured in a facility that will be built and operated on site by a properly licensed explosives supplier. The explosives supplier will be responsible for transporting the raw materials to site, manufacturing the bulk emulsion and loading the blast holes using his own fleet of pumper trucks. The explosives supplier will also be responsible for supplying the blasting accessories such as detonators, boosters and priming cord, although it will be the mine's responsibility to transport these accessories from the magazines to the drill patterns and to tie-in the blast holes. The blasts will be triggered using electronic detonators.

The explosives plant and the magazines to store the accessories will be located at the south end of the open pit and are shown on Figure 16.3. The locations of these sites account for the minimum distance requirements that are specified by the Canadian Explosives Regulations. Approvals and permits will be required from government regulating bodies prior to the construction of these facilities.

The explosives plant will be composed of predesigned / prebuilt modules that are easily transported and assembled. The facilities include storage silos for raw materials, the offices and garages, as well as the emulsion plant and pumper truck loading area.

In order to support the explosives supplier, the mine operator is required to build and maintain the access road to the two (2) sites and to supply electric power, communications and diesel fuel for the manufacturing of the emulsion as well as the operation of mobile equipment. The mine operator is also required to mobilize and house the contractor's workforce.

When the mine is operating at full production during Phase 2, there will be roughly four (4) to five (5) blasts per week, each producing approximately 1 Mt of material. The amount of explosives required per year is approximately 130,000 tonnes.

16.3.3 Shovels and Loaders

The main loading machines selected for the Project are electric cable shovels with 100 to 110-tonne capacity buckets. These shovels, which are the largest on the market, offer high productivity, which is required to excavate the tonnages expected. These large-size cable shovels can load the 344 tonne haul trucks, presented in the next section, in four (4) passes.

Electric powered shovels were selected over diesel powered shovels since the cost of electricity for the Project is far less than the cost of diesel fuel. The disadvantage of using electric powered equipment is the extra manipulation of the trailing cable that results in more time required to relocate the equipment. Additional electrical infrastructure such as cable stands and cable reelers are also required.

Since hydraulic shovels, which were historically only offered as diesel powered machines, are now available as electric powered machines, a trade-off study was done to determine whether the Project should use cable shovels or hydraulic shovels as the main loading machines. The three (3) main factors that were considered in the evaluation were: selective face digging, productivity and cost.

a) Selective Face Digging

Unlike a cable shovel, a hydraulic shovel can mine the face selectively due to the configuration of the boom and bucket. Nevertheless, selectivity is not of huge importance for the Lac Otelnuk deposit since the iron formation is very homogeneous and there are not many small waste pockets that must be separated from the ore.

b) Productivity

The design of the cable shovel allows for larger buckets than a hydraulic shovel. The large cable shovels can be equipped with 110-tonne buckets while the large hydraulic shovels are typically maxed out at a 93-tonne bucket. Even though the hydraulic shovel has roughly a 30 % faster cycle time, the cable shovel will outperform the hydraulic shovel in terms of material mined. The estimated productivity of a cable shovel is 7,432 t/h and the estimated productivity of a hydraulic shovel is 7,091 t/h.

c) Cost

The purchase price including freight and assembly of a large cable shovel is roughly 37 $ M and the purchase price of a large hydraulic shovel is roughly $ 23 M. The expected life of a cable shovel is 120,000 hours although this can be extended with some component rebuilds. The expected life of a hydraulic shovel is 40,000 hours. Therefore, during the 30-year span of the Feasibility Study, the cable shovels would not require replacement, while the hydraulic shovels would be replaced every 10 years.

The estimated operating cost of a large cable shovel has been estimated at $ 1,121/h. This accounts for power, repair parts and GET items (Ground Engaging Tools). The estimated operating cost of a large hydraulic shovel is $ 1,628/h.

An analysis was carried out to determine the life cycle cost (purchase price, machine replacement and operating cost for each type of shovel over a 30-year span. The analysis indicated that the cable shovel would be about 25 % cheaper, accounting for the time value of money.

The results of the trade-off show that, over the life of the Project, cable shovels are preferable as the main loading machines. Nevertheless, in order to provide the flexibility of faster relocations and face selectivity, the fleet of shovels will also include two (2) diesel powered hydraulic shovels and four (4) front end wheel loaders.

The number of cable shovels required was calculated to be five (5) during Phase 1 plus an additional five (5) during Phase 2 for a total of ten (10). Each cable shovel will mine approximately 22 Mt of rock per year, as estimated using the following assumptions:

- Mechanical availability – 85 %;
- Utilization – 80 % (Non-utilized time is accrued when the shovel is relocating during blasts and when the shovel is parked for an extended period of time due to not enough trucks available to load);
- Operating delays – A total of 90 minutes per shift has been allocated to operating delays such as shift change, equipment inspection and lunch / breaks;
- Bucket fill factor – 90 %;
- Efficiency – An efficiency of 37 min/h was used. This number may seem low, but it reflects the fact that there are times when the shovel is sitting idle waiting for a truck and also takes into account the fact that productivities will be reduced to due blending requirements.

With a good maintenance program and regular parts replacement, a cable shovel can typically operate for at least 25 years. No replacement cable shovels have been included in the Feasibility Study.

As was previously mentioned, the fleet includes two (2) diesel powered hydraulic shovels which will be used as secondary loading machines and will be added to the fleet at the start of the operation. Hydraulic shovels add flexibility to the operations since they are more mobile than the electric cable shovels and can be moved around to assist with ore blending. The hydraulic shovels are also more productive than the cable shovels for development work and when establishing ramps. If there are electrical distribution problems, the hydraulic shovels will continue to operate since they are powered by diesel fuel.

The hydraulic shovels typically last for 40,000 hours (10 years) and will therefore be replaced twice during the 30-year span of the Feasibility Study.

The fleet includes three (3) front end wheel loaders in Phase 1 and an additional one (1) in Phase 2. These loaders will be used for ore rehandling, as additional loading units in the pit, and to clean up the low faces after the blasts.

The front end loaders will operate roughly 3,000 hours per year and will be replaced after 35,000 hours of operation (approximately 12 years).

16.3.4 Haul Trucks

The haul truck selected for the Project has a nominal payload of 400 short tons (363 metric tonnes). This size of truck was selected since it is the largest available on the market and is required in order to keep the number of trucks in the fleet to a manageable size. Reducing the number of trucks minimizes the number of operators, which is a high-cost item due to the remoteness of the Project, and maximizes the hauling efficiency by minimizing interferences on the roads, in the shovel pits, and on the crusher pads.

Since the haul trucks will be used in an iron ore environment, the truck manufacturers have recommended a special body liner which minimizes wear and tear and extends the life of the box. Due to the weight of this liner, the truck's payload is reduced to 344 metric tonnes.

The truck requirements were calculated assuming that each truck can deliver 5,443 hours of productive work per year which is based on the following assumptions:

- Mechanical availability – 87.5 %.
- Utilization – 90 %.
- Operational delays – 80 min/shift (this includes 15 minutes for shift change, 40 minutes for lunch and coffee breaks, 15 minutes for equipment inspection and 10 minutes for re-fueling which will be carried out every second shift and last for 20 minutes).
- Job efficiency – 90 % (54 min/h; this represents lost time due to queuing at the shovel and dumping locations as well as interference on the haul road).

The number of trucks required was calculated to be 23 during Phase 1 plus an additional 27 during Phase 2 for a total of 50. In order to calculate the truck requirements, haul routes were generated for each period of the mine plan for ore, waste and low grade. These haul routes were imported in Talpac©, a commercially available truck simulation software package that Met-Chem has validated with other mining operations. Talpac© calculated the travel time required for a 344-tonne haul truck to complete each route. A rolling resistance of 3 % and a maximum truck speed of 50 km/h were used in the calculation. The travel time for each haul was then added to the spotting, loading and dumping time to arrive at the total cycle time which was used to determine the productivity for each haul. Table 16.6 below shows the various components of a truck's cycle time.

Total truck hours for each period of the mine plan were calculated by multiplying the tonnage with the haul productivities (t/h). The trucks requirements were then calculated by dividing the total hours required with the 5,443 hours that each truck can deliver per year. The haul trucks will be replaced after 85,000 hours of operation (approximately 15 years).

Table 16.6 – Truck Cycle Time

Activity	Duration (sec)
Spot time at shovel	45
Load time	180
Travel time	Calculated by Talpac©
Spot time at dump	30
Dump time	30

Note: The Load Time of 180 seconds assumes four (4) passes @ 45 sec/pass.

16.3.5 Auxiliary Equipment

The following fleet of support and service equipment will be required to carry out the mine plan when the mine reaches full production in Phase 2:

- Ten (10) track dozers (634 - 655 kW) for the construction of the mine rock piles and low grade stockpiles as well as drill pad preparation and pit and road maintenance.
- Four (4) wheel dozers (637 - 674 kW) for the cleanup of the shovel pits. In order to estimate the number of wheel dozers required, it was assumed that one (1) wheel dozer is required for every three (3) cable and hydraulic shovels.
- Five (5) road graders (400 kW) for mine haul road maintenance and snow removal.
- Three (3) water trucks equipped with 200,000 litre tanks for dust suppression. The water trucks will be converted to spread crushed stone for the roads during the winter.
- Two (2) secondary track drills for development work, establishing pre shear holes when wall stability control is required, blasting of oversized boulders and additional drilling that the main units will not be able to achieve.
- Two (2) small excavators (110 - 130 kW) and two (2) large excavators (350 – 400 kW) for establishing dewatering ditches and sumps as well as performing many other miscellaneous jobs.
- Eight (8) small utility haul trucks in the 60 – 65 tonne payload class for miscellaneous jobs which include road maintenance.
- Three (3) utility front end wheel loaders (250 - 300 kW).
- Four (4) front end wheel loaders (250 - 300 kW) equipped with a cable reel attachment to manipulate the trailing cable during shovel and drill relocations.

The support and service equipment fleet is presented in Table 16.7. The remaining support and service equipment includes fuel/lube trucks, mechanic trucks, tire handlers, tow trucks, boom trucks, mobile cranes, lowboys, transport busses, pickup trucks and light towers.

Table 16.7 – Support and Service Equipment

Description	Phase 1	Phase 2	Total	Major Specification	Example Model
Support Equipment					
Track dozer	6	4	**10**	Power: 634 – 655 kW	CAT D11T
Wheel dozer	3	1	**4**	Power: 630 – 675 kW	CAT 854K
Road grader	3	2	**5**	Power: 400 kW	Caterpillar 24M
Water / sand truck	2	1	**3**	Capacity: 200,000 litres	CAT 793 Chassis
Secondary drill	2	0	**2**	Hole diameter: 102 - 203 mm	Atlas Copco ROC L9
Utility excavator (a)	1	1	**2**	Power: 110 - 130 kW	CAT 320D
Utility excavator (b)	1	1	**2**	Power: 350 - 400 kW	CAT 390D
Utility haul truck	5	3	**8**	Payload: 60 - 65 t	CAT 775G
Utility front end loader	2	1	**3**	Power: 250 - 300 kW	CAT 980K
Lighting plant	10	5	**15**	Power: 5 - 15 kW	Magnum MLT3060
Cable reeler	2	2	**4**	Power: 250 – 300 kW	CAT 980
Powder truck	3	6	**9**		Ford F250
Service Equipment					
Fuel / Lube Truck	3	1	**4**	Capacity: 11,000 litres	Peterbuilt 365
Mechanic truck	7	3	**10**		Peterbuilt 348
Tire handler	1	1	**2**		Kalmar DCF280-12LB
Boom truck	2	2	**4**	21 tonne capacity	Terex BT4792
Mobile crane (a)	2	1	**3**	75 tonne capacity	Terex RTC8080
Mobile crane (b)	2	1	**3**	250 tonne capacity	Terex ATC3275
Tow truck / lowboy	2	0	**2**	227 tonne capacity	CAT 793 Chassis
Transport bus	3	2	**5**	20 person capacity	Ford E350

16.3.6 Mine Dispatch

A mine dispatch system will be installed since the fleet size will be very large and ore-blending requirements are important. The costs to install and maintain the mine dispatch system are included in the cost estimate for the Feasibility Study.

16.3.7 Equipment Simulation System

Equipment simulators will be used to train the operators of trucks, shovels, dozers and graders so that the operation can run safely and efficiently. The simulators will be located in the mine administration building. The simulators are included in the cost estimate for the Feasibility Study.

16.3.8 Contract Mining for Overburden Removal

As mentioned previously, the overburden will be removed by a mining contractor. This is typical in large open-pit operations since the overburden stripping is done seasonally, usually in the winter, and the mine will not have the correct size of equipment for the job. Several mining contractors were solicited for budgetary pricing and were provided with the overburden removal requirements as well as the haul distances to the stockpiles for each period of the mine plan. The overburden stripping will be carried out each year ahead of the mining operation and the cost for the contract has been included in the cost estimate for the Feasibility Study.

16.4 Mine Dewatering

The term "dewatering" applies to the management of water which, if not diverted from the pit or pumped out from it, would impede mining operations or add to operating costs in other ways. Typical related issues include access to benches, trafficability, blasting costs and wear and tear on machinery. Table 16.8 presents the average daily mine dewatering flow rates for each period of the mine plan for the wettest month, June. The table also presents the total annual volume of water that will collect in the open pit and which must be removed by mine dewatering system. The table accounts for water that originates from the following sources that are discussed below; surface runoff, rainfall, snowfall and snowmelt and groundwater.

Table 16.8 – Average Daily Mine Dewatering Flow Rates (June) and Yearly Total

Year	Precipitation (m^3/d)	Groundwater (m^3/d)	Total (m^3/d)	Yearly Total (Mm^3/y)
1	4,072	1,850	**5,922**	**0.8**
2	10,180	2,704	**12,884**	**1.6**
3	18,324	3,558	**21,882**	**2.6**
4	26,468	4,199	**30,667**	**3.5**
5	35,291	4,839	**40,130**	**4.5**
6	42,756	6,832	**49,588**	**5.7**
7	47,507	8,825	**56,332**	**6.6**
8	63,116	9,394	**72,510**	**8.2**
9	65,152	9,964	**75,116**	**8.6**
10	88,227	10,248	**98,475**	**10.8**
11 – 15	123,517	10,533	**134,050**	**14.1**
16 – 20	149,307	10,960	**160,267**	**16.6**
21 – 25	176,453	13,807	**190,260**	**19.8**
26 – 30	192,741	18,931	**211,672**	**22.6**

a) Surface Runoff

In order to minimize the amount of water collecting in the pit, a system of ditches will be established around the perimeter of the open pit to divert surface runoff water. This system of ditches will evolve as the pit area increases over time. The ditches will mostly be dug in the overburden but they may also penetrate into the bedrock due to topographical constraints. In order to avoid blasting the bedrock, short berms may also be used to establish parts of the ditch diversion system. The ditches will be constructed and maintained by the mine operations group who will have a dewatering crew and a small fleet of utility excavators and haul trucks. Water that is collected in this perimeter ditch network will not come into contact with the mining operation and should therefore be suitable for discharge into to the environment. When required, monitoring systems will be installed at the discharge points.

b) Rainfall and Snowmelt (Precipitation)

Table 16.9 presents the estimated average monthly runoff within the open pit and accounts for rainfall, snowfall, snowmelt and an evaporation rate of 10 %. In order to estimate the volumes of precipitation that will accumulate in the open pit for each period of the mine plan as presented in Table 16.8, the surface area of the open pit was multiplied by the runoff.

The annual runoff from rainfall and snowmelt was estimated to represent approximately 76 % of the total water balance in the open pit.

Table 16.9 – Estimated Average Monthly Runoff within the Open Pit

Month	Runoff (mm)
January	1.4
February	0.3
March	0.1
April	0.1
May	30.9
June	203.6
July	151.9
August	82.0
September	74.6
October	58.5
November	17.5
December	4.9
Total	**625.7**

c) Groundwater

The amount of groundwater that is estimated to enter into the open pit was provided by Golder Associates Ltd. ("Golder") in the memorandum titled "Preliminary Hydrogeological Numerical Modelling Results - March 25th 2014". Golder created a 3D numerical groundwater flow model using data gathered during the 2011, 2012 and 2013 hydrogeological field investigations as well as geological information. Based on the mine plan for the Feasibility Study, Golder estimated the annual amount of groundwater that can be expected to infiltrate into the open pit each year.

The annual groundwater flow rates were partitioned into monthly rates and the flows were assumed to be limited to 40 % from December to April due to the cold temperatures. The average daily groundwater flow rates for June are presented in Table 16.8.

The amount of groundwater that will infiltrate into the open pit was estimated to represent approximately 24 % of the total water balance in the open pit.

To capture the water infiltrating the open pit, sumps will be established at the lowest part of the pit floor for each active area of operation. During the 30 year mine life, between two (2) and four (4) sumps will be active at any time. These sumps will be re-established roughly every six (6) months to a year, as the pit wall advances and the floor deepens. The sumps will be excavated in the pit floor to a maximum depth of 8 m so that the pumps will have enough suction power to completely evacuate the water. The excavation of the sumps will require drilling and blasting and will be carried out by the mine's dewatering crew. The volume of the sumps will be determined at the time of excavation and will depend on the expected flow rates. Due to the sloping topography of the pit floor, the water will naturally drain towards the sumps, however temporary ditching may be required in certain situations.

Each sump will be equipped with either one (1) or two (2) centrifugal pumps which will be connected to the electrical grid of the mine. The pumps will be mounted on skids and placed on the ground next to the sump. Since the mine's electrical grid is at 7,200 volts and the pumps are rated at 4,160 volts, the trailing cable that connects to the mobile substation will terminate at a mobile transformer on a skid that will be located next to the pump. The dewatering pipes will be equipped with valves that automatically open during a loss of power to ensure that the lines are drained so that they do not freeze. The pumps will be operated by a motor that will be started automatically when the water level in the sump reaches a certain level.

The pump requirements were estimated for each period of the mine plan to ensure that the water can be evacuated during the wettest month of the year, June. Since the pumps are only designed for the average, there may be situations such as extreme rainfall events where the pumps will not be able to completely evacuate the water collected in the sumps. This is a risk that must be taken since it would be too costly to design the dewatering system for extreme events. To minimize this risk, the water collected in the sumps should be evacuated as soon as possible to ensure that there is sufficient sump capacity to contain large quantities of water.

Several different sized pumps have been specified for the Project to support the different periods of the mine plan: as the open pit progresses, the water quantities increase, as do the head and friction losses. The pump that has been specified for the majority of the 30-year mine life has a 447 kW motor, a flow rate between 95 and 335 L/s and a maximum dynamic head of 115 m. As an example, the Godwin HL260 is a centrifugal pump that falls into this category.

A total of five (5) pumps are required during Phase 1, followed by an additional five (5) in Phase 2. This includes a spare pump in each phase.

The pumps will be connected with 8, 12, 18 or 24-inch-diameter flexible high-density polyethylene (HDPE) pipes and will discharge into either the stream system that flows into the TMF or directly into the TMF, depending on which is closer to the discharge point at a given point in time.

16.5 Mine Workforce

The total mine workforce for the Project peaks at 662 employees in Phase 1 and 1,002 employees in Phase 2. The mine workforce has been grouped into the following categories; Mine Operations, Technical Services, Field Maintenance and Shop Maintenance.

The workforce is comprised of staff employees and shift employees. The staff employees include most of the supervisory roles as well as the majority of the technical services group. These employees will work on a 2 weeks-on, 2 weeks-off rotation and have a cross-shift to cover for them while they are off site. The shift employees will work on four (4) crews in order to provide 24 hours-per-day coverage, seven (7) days a week. At all times, one (1) crew will be on day shift, one (1) crew will be on night shift and two (2) crews will be off site. Table 16.10 presents the peak mine workforce requirements for Phase 1 and Phase 2.

Table 16.10 – Mine Workforce

Description	Phase 1	Phase 2 (additional)	Total
Mine Operations			
Mine Manager	2	0	**2**
Mine Superintendent	2	0	**2**
Clerk	2	0	**2**
Mine Supervisor	4	0	**4**
Pit Supervisor	8	4	**12**
Dispatch Supervisor	4	4	**8**
Blast Supervisor	2	2	**4**
Truck Operators	84	92	**176**
Shovel Operators	28	20	**48**
Loader Operators	12	4	**16**
Drill Operators	32	32	**64**
Track Dozer Operator	24	16	**40**
Wheel Dozer Operators	12	4	**16**
Grader Operators	12	8	**20**
Water / Sand Truck Operators	8	4	**12**

Description	Phase 1	Phase 2 (additional)	Total
Fuel & Lube Truck Operator	8	4	**12**
Labourer	16	8	**24**
Dewatering Crew	8	4	**12**
Power Distribution Crew	8	0	**8**
Aggregate Plant Operator	8	0	**8**
Blasting Crew	24	24	**48**
Janitor	8	4	**12**
Trainers	8	8	**16**
Technical Services			
Engineering Superintendent	2	0	**2**
Clerk	2	0	**2**
Senior Mining Engineer	2	0	**2**
Mining Engineer	4	2	**6**
Geologist	4	2	**6**
Grade Control Technician	8	0	**8**
Mine Technician	2	2	**4**
Surveyor	4	4	**8**
Field Maintenance			
Field Superintendent	2	0	**2**
Clerk	2	0	**2**
Maintenance Supervisor	8	0	**8**
Electrical Supervisor	2	0	**2**
Maintenance Planner	2	2	**4**
Mechanic	32	28	**60**
Equipment Electrical	8	8	**16**
Lineman Electrical Technician	4	0	**4**
Electrical shop	8	8	**16**
Main Power Line Electrician	8	0	**8**
Welder	8	8	**16**
Instrumentation	8	0	**8**

Description	Phase 1	Phase 2 (additional)	Total
Shop Maintenance			
Shop Superintendent	2	0	**2**
Clerk	2	0	**2**
Shop Supervisor	2	0	**2**
Truck Supervisor	4	0	**4**
Auxiliary Supervisor	4	0	**4**
Machine Shop Supervisor	4	0	**4**
Small Vehicle Supervisor	2	0	**2**
Electrical/Instr. Supervisor	4	0	**4**
Maintenance Planner	8	4	**12**
Electrical Engineer	4	0	**4**
Mechanical Engineer	4	0	**4**
Mechanic Truck	32	8	**40**
Mechanic Shovel/Drill	8	0	**8**
Mechanic Auxiliary	16	0	**16**
Mechanic Service Equipment	16	0	**16**
Electrical Technician	12	0	**12**
Instrumentation Technician	8	0	**8**
Mechanic Technician	8	0	**8**
Lab Technician	4	0	**4**
Machinist	8	4	**12**
Component shop	12	0	**12**
Carpenter	8	0	**8**
Plumber	8	0	**8**
Apprentice	6	6	**12**
Welder	8	0	**8**
Janitor	8	0	**8**
Wash Bay Attendant	4	4	**8**
Tire Bay Attendant	8	4	**12**
Tool Crib Attendant	4	4	**8**
Total Mine Workforce	**662**	**340**	**1,002**

17.0 RECOVERY METHODS

The process plant will treat magnetite ore extracted from the Lac Otelnuk open pit mine located in the Nunavik region of the province of Quebec, about midway north in the Labrador Trough iron range, to produce a concentrate. The operation will mill approximately 188.7 Mt/y of iron bearing mineralization to produce 50 Mt/y of magnetite concentrate.

The Project will be developed in two phases: Phase 1 will produce 30 Mt/y of iron ore concentrate and Phase 2 will produce 20 Mt/y to bring the plant to 50 Mt/y capacity. The process plant involves multiple grinding and milling stages followed by a conventional magnetite recovery circuit with desliming thickeners and magnetic separators to produce a pellet feed iron concentrate.

Unless otherwise noted, all weight and throughput are expressed in dry metric tonnes.

17.1 Process Design Criteria

The process plant is designed to treat approximately 188.7 Mt/y of taconite ore with an average magnetite content of 18.2 % and Fe grade of approximately 28.7 % that will permit production of 50 Mt/y concentrate with a Fe content of about 68.5 % and less than 4 % silica. The magnetite weight recovery is critical since any hematite present will not be recovered in the concentration process. The ROM necessary to produce the required tonnage of concentrate is calculated based on the weight recovery and the design factor, part of the design criteria, to ensure that the process equipment has sufficient capacity to take care of the expected feed variation.

Operating 365 days per year, the process plant will recover a nominal 50 Mt/y of pellet feed iron ore concentrate. The plant design is based on a 30-year mine life.

17.1.1 Ore Characteristics

The ore composition and head grade composition used for the process plant design purposes is specified in Table 17.1 and is based on the understanding of the mineral resource and mining plan.

Table 17.1 – Ore Composition

	ROM Mining Plan First 10 Years of operation	ROM Mining Plan Total 30 Years	Head Characterization from Laboratory Testing SGS 11727-012
Element	**Percentage**	**Percentage**	**Percentage**
Fe (total)	28.8	28.7	29.6
Magnetite	18.8	18.2	19.5
SiO_2			45
Al_2O_3			0.05
MgO			2.64
CaO			3.09
Na_2O			0.04
K_2O			0.01
TiO_2			<0.01
P_2O_5			0.04
MnO			0.83
V_2O_3			<0.01

The concentrate quality is shown in Table 17.2. This concentrate quality was achieved during laboratory tests at SGS Lakefield.

Table 17.2 – Concentrate Quality Based on Test Work Results

Element	Percentage
Fe	69.0
SiO_2	2.95
Al_2O_3	0.02
S	0.01
MgO	0.15
CaO	0.24
Na_2O	<0.01
K_2O	<0.01
TiO_2	<0.01
P_2O_5	0.02
MnO	0.16
Cr_2O_3	0.04
V_2O_5	<0.01
Total Carbon	0.23

17.1.2 Metallurgical Mass Balance

Table 17.3 presented below shows the overall metallurgical mass balance.

Table 17.3 – Metallurgical Mass Balance

STREAM	Flow (solid) tph	%	% Fe	% MagFe	% Recovery	
					Fe	MagFe
Feed to 3 SAG	4786.38	100.00	28.7	18.2	100.00	100.00
Recalculated Head	4786.38		28.7	18.2		
Cobber Concentrate	3407.91	71.20	34.3	25.1	85.1	98.2
Cobber Tails	1378.48	28.80	14.9	1.1	14.91	1.8
Feed	4786.38	100.00	28.70	18.2	100.00	100.00
Rougher Concentrate	2191.28	45.78	46.7	38.6	74.49	97.1
Rougher Tails	1216.62	25.42	12.0	0.8	10.60	1.1
Cobber Concentrate	3407.91	71.20	34.3	25.1	85.1	98.2
Slimes	755.99	15.79	12.7	1.8	7.00	1.5
Feed to Cleaner Separators	1435.29	29.99	64.6	58.0	67.50	95.6
Rougher Concentrate	2191.28	45.78	46.7	38.6		
Concentrate	1268.80	26.5	69.10	65.60	63.82	95.5
Cleaner Tails	166.49	3.48	30.00	0.1	3.64	0.02
Feed to CS	1435.29	29.99	64.6	58.0	67.46	95.56
Total Tailings	3517.59	73.5	14.11	1.10	36.14	4.5

The ore grade used in the final mass balance calculations is 28.7 % total Fe, based on the 30 year mine plan average value.

17.1.3 Process Plant General Design Basis

The process plant is designed to operate for 24 hours per day, seven (7) days per week and 52 weeks per year. Primary crushing will have an availability of approximately 70 %. The Process plant will operate at an availability of approximately 90 % as per the Process Design parameters shown in Table 17.4 below.

Table 17.4 – Process Plant General Design Basis

Description	Unit	Value
Availability Primary Crusher	%	70
Availability Process Plant	%	90
Annual Concentrator Throughput	Mt/y	188.7
Average Davis Tube Weight Recovery (30 Years)	%	26.5
Hourly Processing Plant Throughput (Dry Basis, Both Phases)	t/h	23,932
Primary Crusher Feed Rate (Dry Basis Per Crusher)	t/h	6,154
Number Of Primary Crushers (Both Phases)		5
Work Index For Crusher Design	kWh/t	19.9
Pocket Design Capacity (Live, For 2 Trucks)	t	688
Emergency Stockpile Design Capacity	h	24
Maximum Crusher Feed Size	mm	1,321
SAG Specific Power Consumption	kWh/t	12.9
SAG Recirculating Load	%	30
Cobber Magnetic Field Strength At 50 Mm	Gauss	1,150
First Stage Ball Mill Bond Work Index	kWh/t	14.8
First Stage Ball Mill Recirculation Rate	%	250
Rougher Magnetic Field Strength At 50 Mm	Gauss	1,000
Second Stage Ball Mill Bond Work Index	kWh/t	17.5
Second Stage Ball Mill Recirculation Rate	%	250
Cleaner Magnetic Field Strength At 50 Mm	Gauss	900
Concentrate Thickener Underflow Density	wt%	67
Concentrate Thickeners Flow Rate (Dry Basis, 1 Train)	t/h	1,269
Concentrator Annual Flow Rate (Dry Basis, Both Phases)	Mt/y	50
Tailing Thickener Underflow Slurry Concentration By Weight	wt%	60
Tailing Thickener Flow Rate (Dry Basis, 1 Train)	t/h	2,829

17.2 Process Facilities Location Criteria

The following key design considerations were used to establish the location and layout of the process plant complex:

- Minimize environmental impact;
- Provide a safe working facility to suit subarctic conditions;
- Ensure that no permanent infrastructure interferes with potential mineral resources;
- Ensure that the equipment and process used are of known and proven technology;

- Design primary crusher area with 70 % availability and process plant with 90 % availability;
- Consider the phasing of the project in the design: Phase 1 at 30 Mt/y and Phase 2 for an additional 20 Mt/y to be constructed a few years later;
- Integrate operability, maintainability, and constructability elements;
- Carefully consider natural topography to minimize surface preparations (earthworks) and facilitate building and system erection and construction during harsh weather conditions;
- Consider prevailing wind directions to limit dust carry-over from tailings and mine site, snow accumulation and excessive cold air infiltrations;
- Minimize the distance between the living camp and work areas by locating buildings, services, and annexes as close together as possible to facilitate transfer between them, while ensuring adequate safety and providing adequate nuisance (noise, dust, etc.) separation;
- Optimize the material handling of ore to maximize energy conservation, minimize length of conveyors and number of transfer towers;
- Optimize the location of main equipment in the plant to minimize pumping requirement and make maximum use of gravity for slurry flows;
- Optimize the design of the plant complex to avoid unnecessary circulation and minimize ground use;
- Provide adequate and effective living accommodation and conditions for employees.

17.3 Description and Geometry

The main process plant complex includes:

- Primary crushing including ore conveyor;
- Ore stockpile and reclaim systems;
- Process plant with SAG mills, ball mills, magnetic separators, and deslimers;
- Tailings thickeners;
- Concentrate thickeners;
- Slurry storage tanks and pump stations for PDS (product delivery system).

The following three figures show progressively closer views of the plant complex:

- Figure 17.1 – Aerial View of Process Plant, Mine, TMF, WSP, Camp and Stockpiles:

  The location of the process plant complex relative to the mine, Tailings Management Facility (TMF), Water Supply Pond (WSP) and stockpiles.
- Figure 17.2 – Closer Aerial View of Process Plant and Primary Crushers:

  A view of process plant complex and primary crushers.

- Figure 17.3 – Phase 1 Process Plant Facilities:

  Isometric view of the Phase 1 process plant facilities.

The plant is located within the LOM claim boundaries in the proximity but not on top of the potential mineral resources. It is positioned on solid bedrock approximately 4 km west of the mine pit.

The benefits of the selected location are as follows:

- The plant makes effective use of topographic conditions in order to limit cut-and-fill quantities and to reduce energy consumption during operation since a big portion of the slurry flow will be by gravity
- The location provides space potential for future expansion in the area
- The relative proximity of the process plant and the mine allows for most buildings to be shared, thereby providing a more compact and efficient facility
- The distance for the transfer of ore between the facilities is as short as practically possible. The proximity to the tailing management facility on the downstream side will help minimize the power required for the transport of the tailings slurry via piping systems
- The proximity to a significant water pond (WSP) about 2 km upstream provides an adequate and reliable source of raw water supply to the plant.
- The airstrip is located about 12 km from process plant, far enough away for the mine pit, on the main access road on the direction to Schefferville.

Figure 17.1 – Aerial View of Process Plant, Mine, TMF, WSP, Camp and Stockpiles

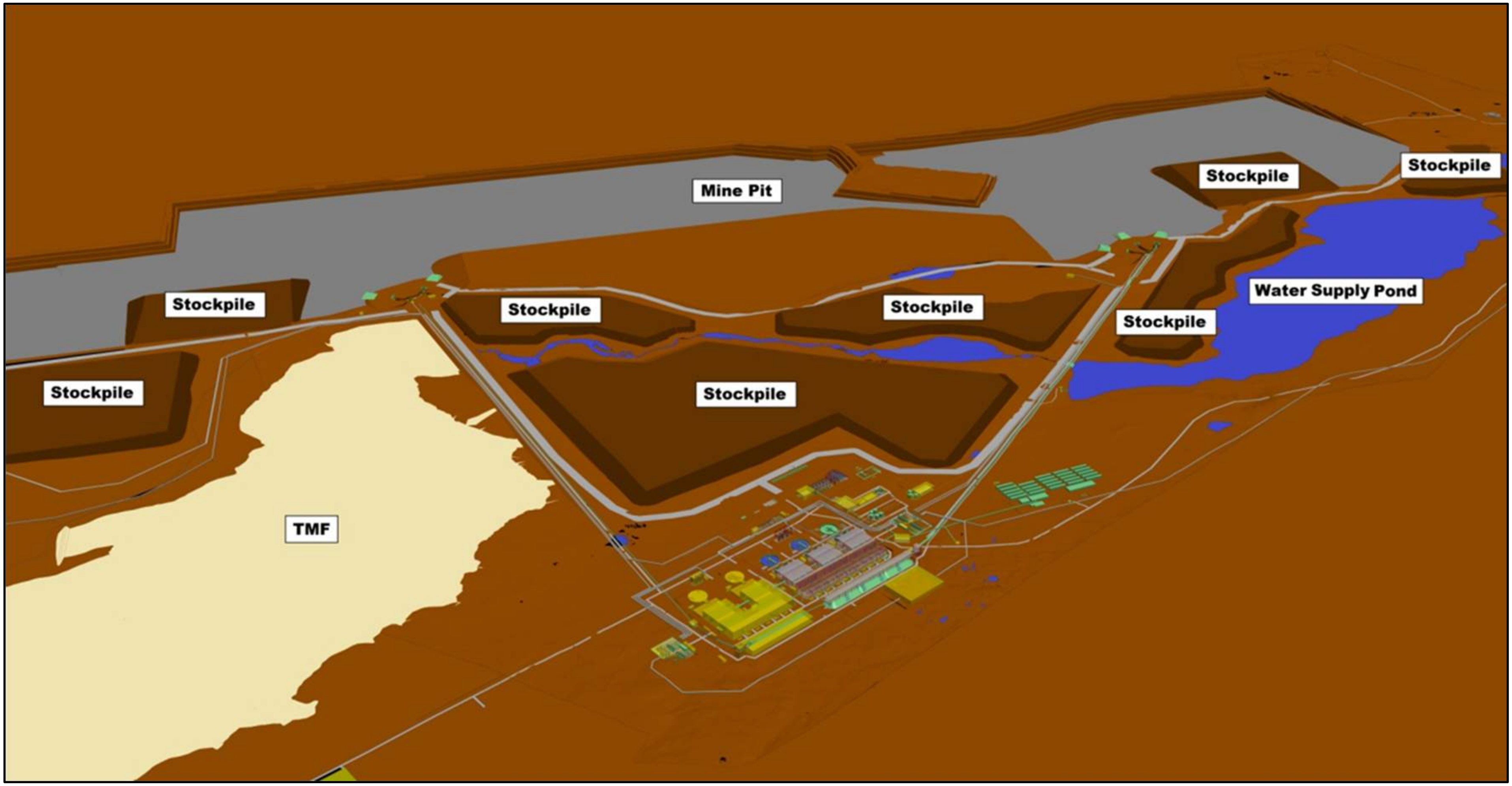
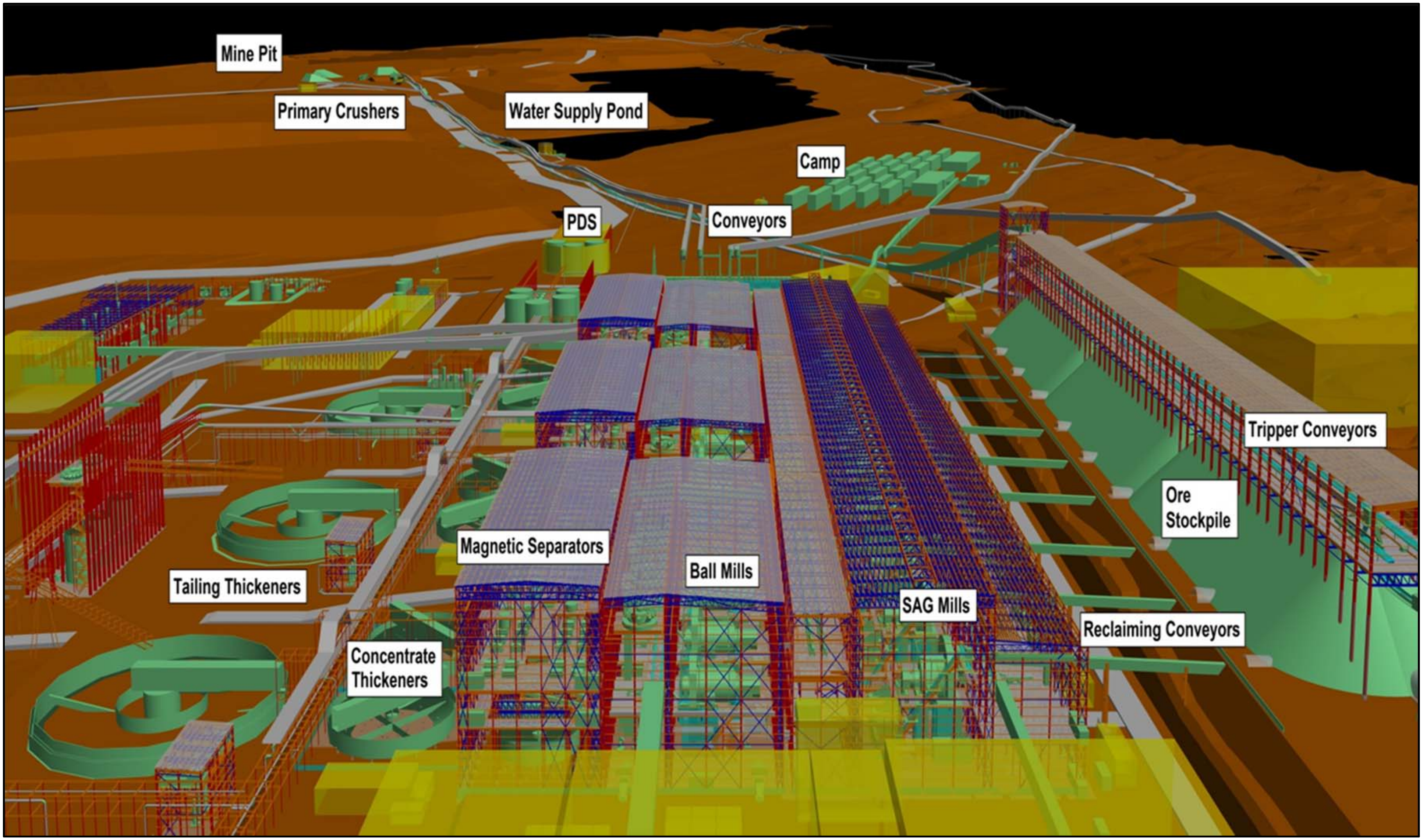


Figure 17.2 – Closer Aerial View of Process Plant and Primary Crushers

Figure 17.3 – Phase 1 Process Plant Facilities



17.4 Process Flow Sheet

17.4.1 Feasibility Study Flow Sheet Development

The processing plant flow sheet and design criteria are based on the results from the metallurgical test work, program discussed in the Section 13.0 of this report.

The process flow sheet, refer to Figure 17.4 and Figure 17.5 below, was developed on the bench-scale test work results and complemented by supplier tests for equipment sizing. The bench-scale test work was performed on the 30 Y composite samples since these samples are representative of the average feed from the mine plan. The SAG process design has been defined with the pilot plant test results.

- The final product is a pellet feed concentrate with a minimum Fe content of 68.5 % and with less than 4.0 % SiO_2 + Al_2O_3, with an average Fe total weight recovery estimated at 26.5 % during the 30 years of mine life.
- The process design is based on conventional equipment and processes that are currently used in the industry and have a proven record.
- Primary crushing is ensured by five large-capacity gyratory crushers. The Project is planned to be implemented in two phases: Phase 1 (30 Mt/y of concentrate) with three crushers and Phase 2 (additional 20 Mt/y of concentrate) with two additional crushers to reach a total capacity of 50 Mt/y of concentrate.
- The capacity of the ROM stockpile located between the mine and the beneficiation plant has been designed for 16 hours.
- From that stockpile, five parallel production trains are fed, each using three parallel 40’ semi-autogenous grinding mills (SAGs) equipped with gearless drives. These represent the largest currently proven size of SAG mills.
- The SAG mills are sized to accept a maximum of 15 % ball load and they are in a closed circuit with a 3.35 mm screen to keep the circulating load below 30 %. A 10 % safety factor on the throughput is applied on the SAG calculation.
- The cobber magnetic separation is performed at a transfer size in the range of 700-800 microns to balance the load between the SAG mill and the first stage ball mill. Even though slightly better metallurgical results can be achieved with 1.00 mm size of screens, this screen size was not selected because smaller transfer size would require more power on the SAG stage.
- Single-drum counter-rotation LIMS units are used, in line with conventional practice for cobber separation.
- Fine grinding is achieved by two-stage ball milling separated by a roughing LIMS section. This is in accordance with laboratory and pilot test work results.
- The cobber-concentrate weight recovery from the bench-scale test work, 71.2 %, appears very conservative in comparison to the pilot-plant results, 47 %, so no safety factor is added to the design of the first-stage ball-mill regarding throughput.
- Two (2) first stage ball mills with gear drives are necessary for each production train.

- Roughing-concentrate weight-recovery of bench-scale test work, 64.3 %, is used in basic design since it is similar to the pilot plant average in operation, 62.6 %.
- A 10 % safety factor on throughput is applied to the second-stage ball-mill calculation.
- Single drum counter-current LIMS units are used, in line with conventional practice for rougher separation.
- The final grind size of D_{80} = 48 microns has produced good quality concentrate in the pilot plant.
- A typical value of 250 % for ball-mill circulation load has been used for both ball milling stages since the pilot plant operation was not representative of the industrial standard.
- For each production train, one large second-stage ball mill with wraparound motor is preferred to two smaller second-stage ball mills with gear drives.
- Cleaner magnetic separators are used, in accordance with the flow sheet testing. Since initial testing showed a high loading of slimes in the product after the three-stage cleaner LIMS, it was decided to include a de-sliming thickener before the cleaner LIMS.
- Triple-drum counter-current LIMS units are used, in line with conventional practice for cleaner separation.
- The process mass balance has been developed using metallurgical results from bench-scale test work on the 30 Y composite. A weight recovery correction was made to account for the average magnetite grade of the mine plan, which is slightly lower than that of the 30 Y composite.
- The process parameters mentioned above have been used for the metallurgical balance. The mill power consumptions, flows, and product particle-size distributions have been determined using the pilot plant results and in-house database calculation.
- JKSimMet® software simulation has been performed to confirm the data obtained in the pilot plant. Some adjustments were made on the SAG power consumption to take into account a coarser industrial feed.

Figure 17.4 – Simplified Process Block Flow Diagram − Phase 1

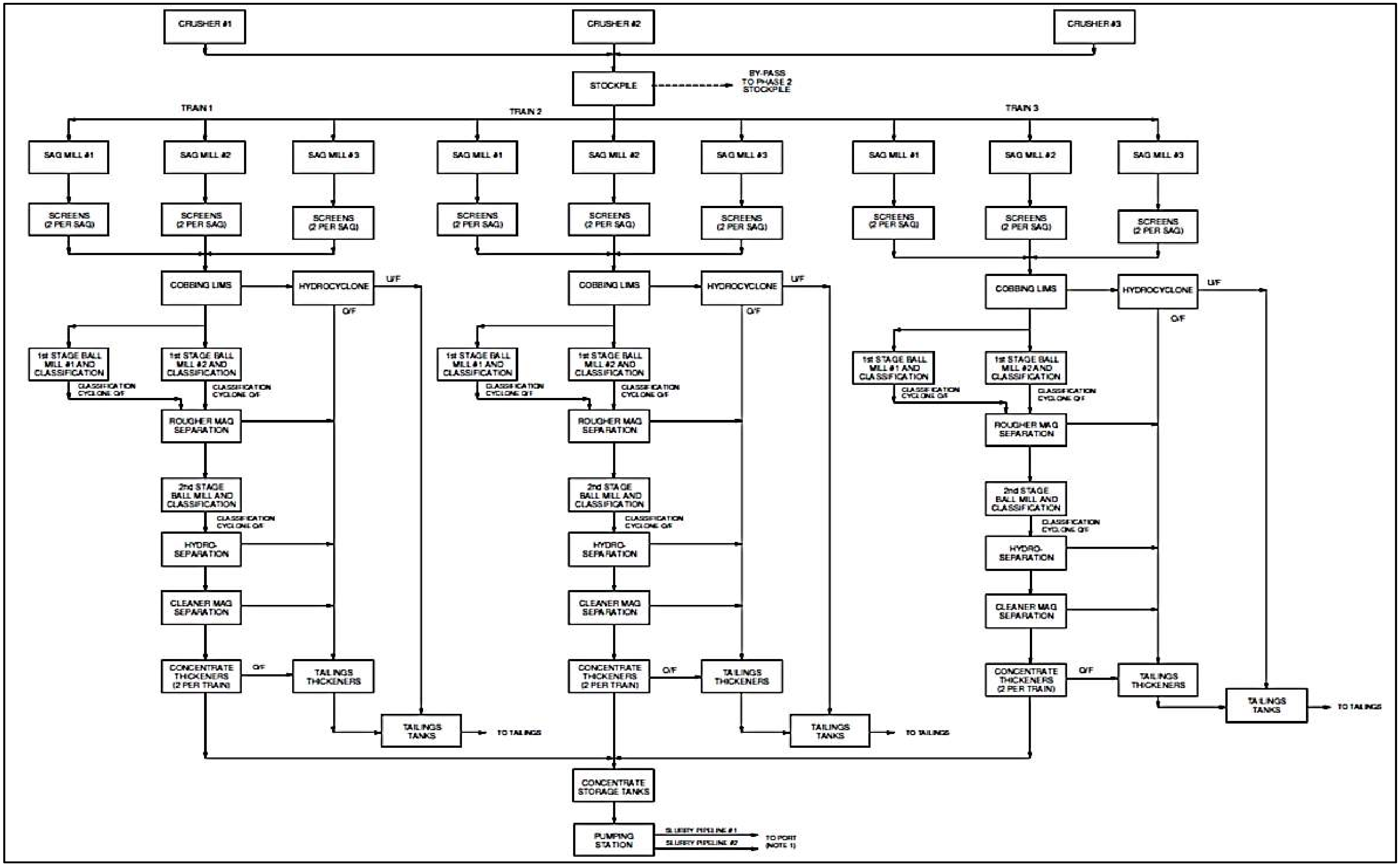
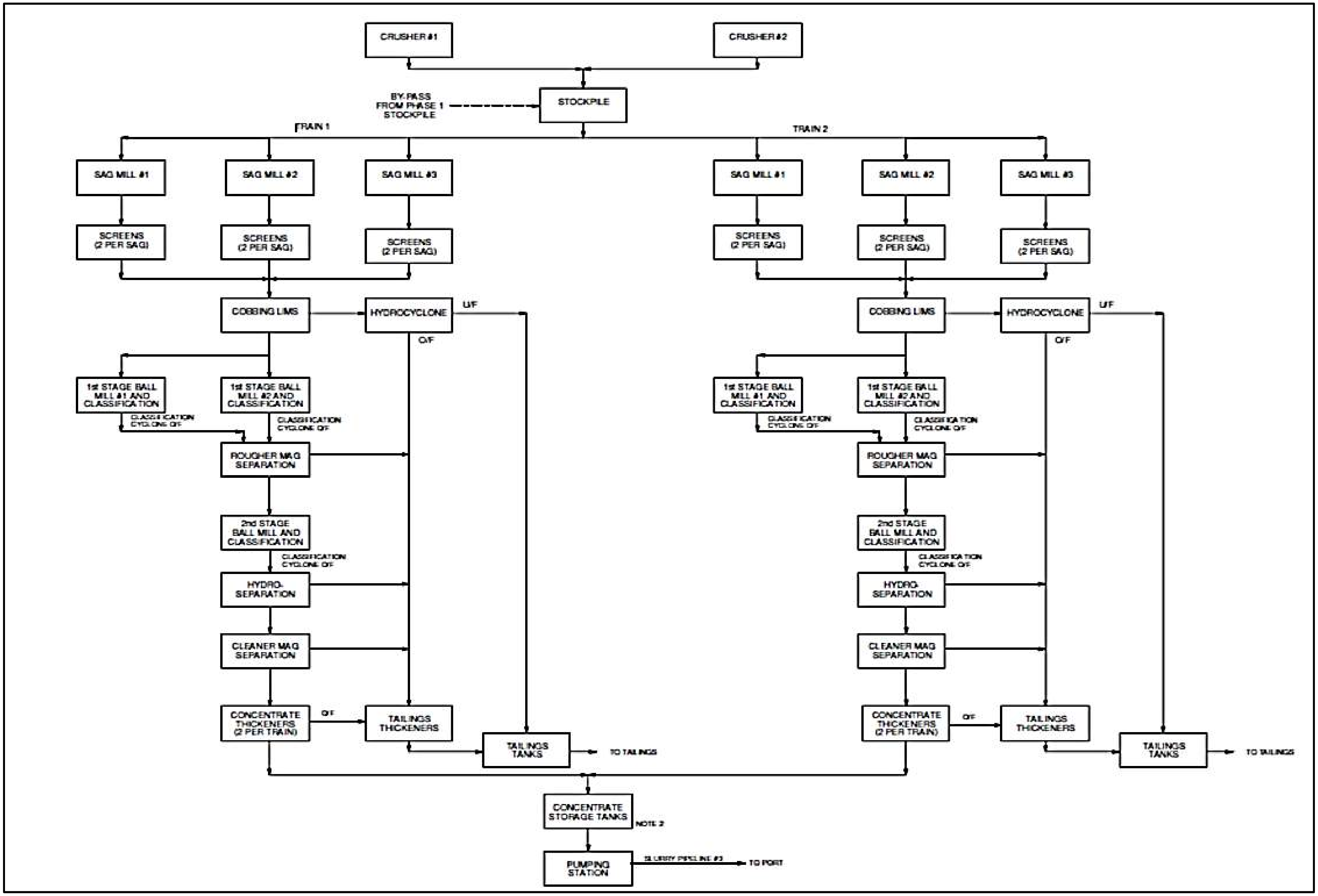


Figure 17.5 – Simplified Process Block Flow Diagram − Phase 2



17.4.2 Primary Crushers:

Primary crushing is performed by five large-capacity gyratory crushers:

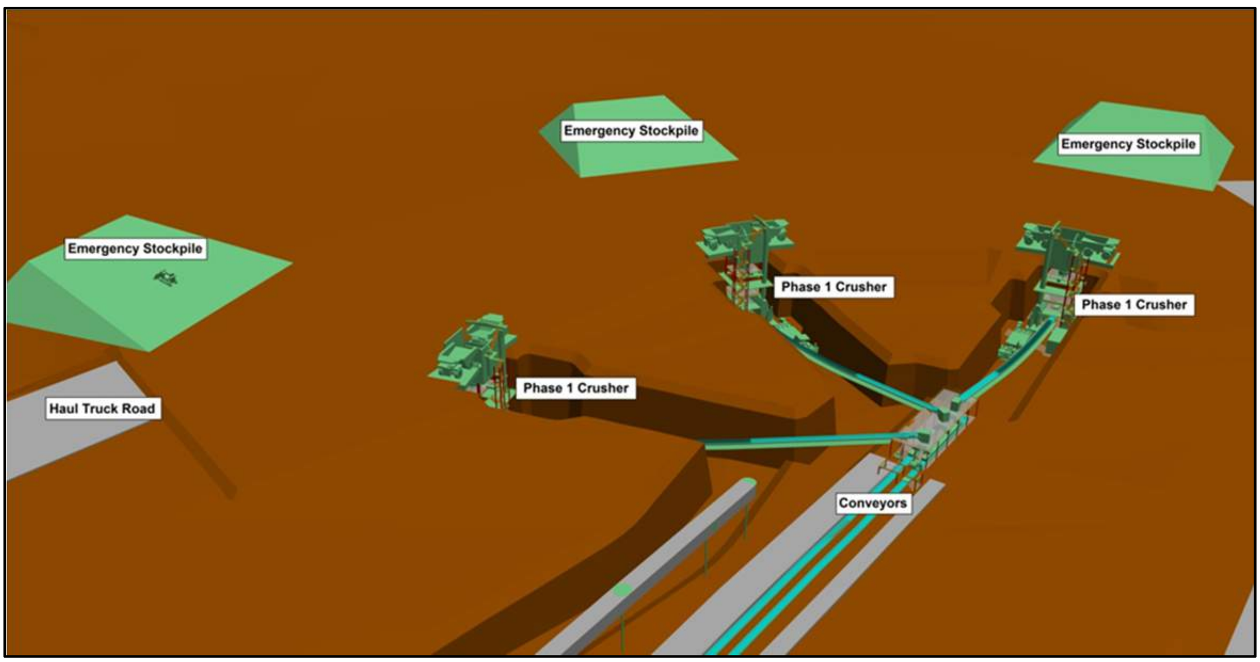
- Phase 1 comprises three crushers
- Phase 2 comprises two additional crushers at a different location.

The arrangement of Phase 1 primary crushers is shown in Figure 17.6. The crushers are all located at a minimum of approximately 0.5 km from the mine pit for safety reasons during blasting. The ore is transported from the mine to the primary crushers by 400 ton (short ton) haul trucks.

A one-day emergency ore storage capacity area is provided close to each crusher for unexpected situations. This ore can be charged directly to the crushers by using front end loaders.

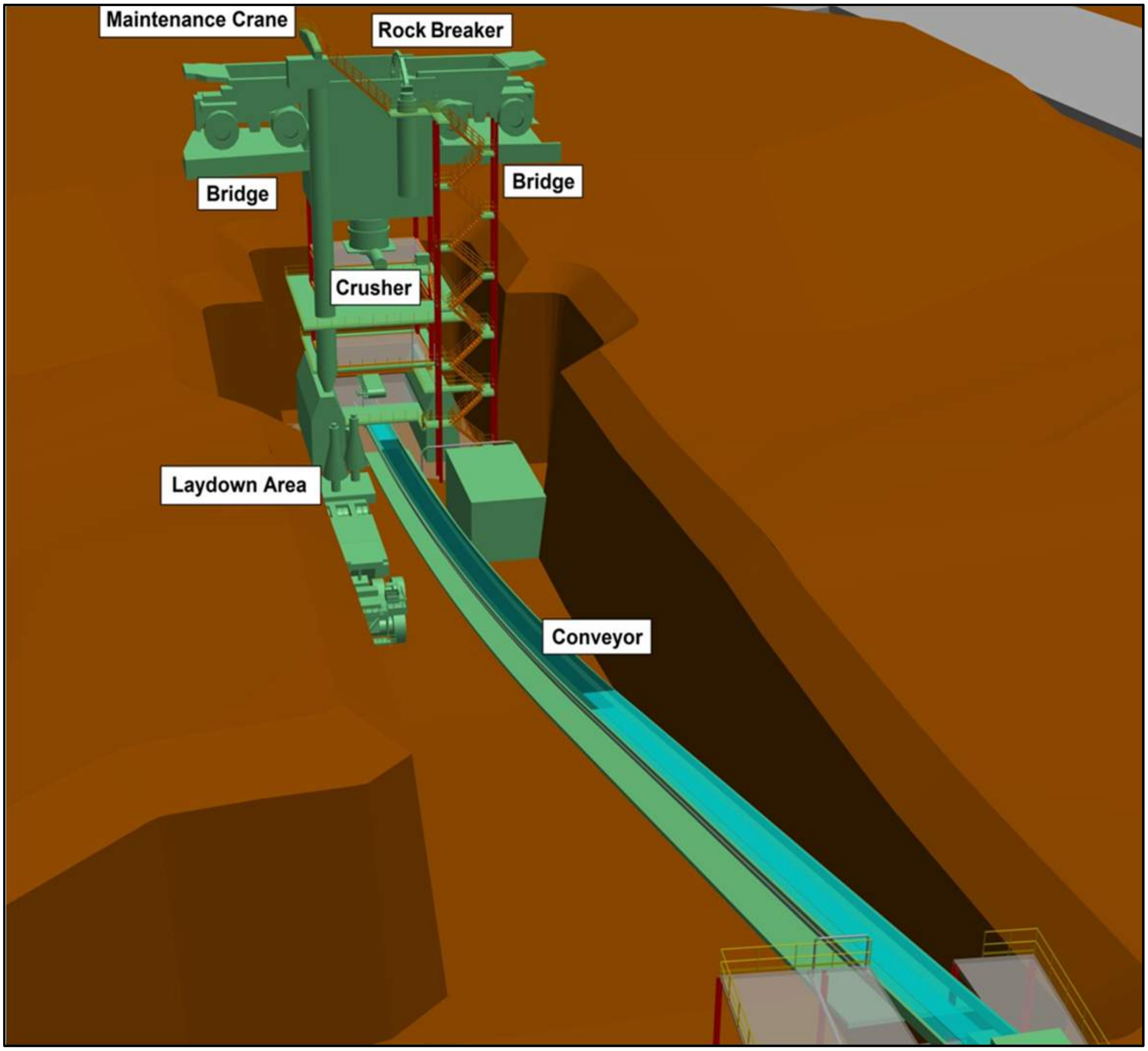
In the foreground of Figure 17.6, the main conveyor for the crushed ore leads to the process plant.

Figure 17.6 – Phase 1 Arrangement of Primary Crushers



A close-up view of the typical primary crusher is presented in Figure 17.7 where haul trucks can be seen at the receiving end of the crusher. At each primary crusher, two (2) ore unloading points are located at 180° to each other. Since two trucks can approach the ore discharge chute simultaneously, this arrangement is safer and facilitates truck approach while it eliminates waiting time.

Figure 17.7 – Close-up of Typical Primary Crusher



The process plant utilization factor is 90 %, while the primary crushers are at 70 % utilization factor. The capacity of each crusher at 70 % utilization factor is therefore 6,154 t/h. The supply rate of the ore conveyed to the stockpile at the process plant is greater than the capacity being processed. When all crushers are operating, the overcapacity is used to replenish the plant stockpile. This ensures sufficient supply whenever one of the primary crushers is not operating.

The semi-mobile primary crushers and all ancillary equipment are installed outdoors. A small heated operation building is provided. Water-spray dust suppression is provided at the truck-unloading pit.

The following equipment and facilities are installed with each primary crusher:

- Main steel structure
- Jib service crane
- Rock breaker
- Dust suppression system (water pipe, spray nozzles, air blower)
- Crusher lubrication systems
- Hydraulic units
- Crusher service platform
- All required chutes and bins
- Crusher apron discharge conveyor
- Electrical room
- Control room.

Water required for the crusher operation is provided by the plant water supply system.

It is noted that the current design concept for the crushers is based on "fixed locations". Moving of the crushers has not been considered. However, the concept of moving the crushers into the open pit to reduce the hauling costs during the operation may need further assessment if such repositioning of the crushers is deemed feasible.

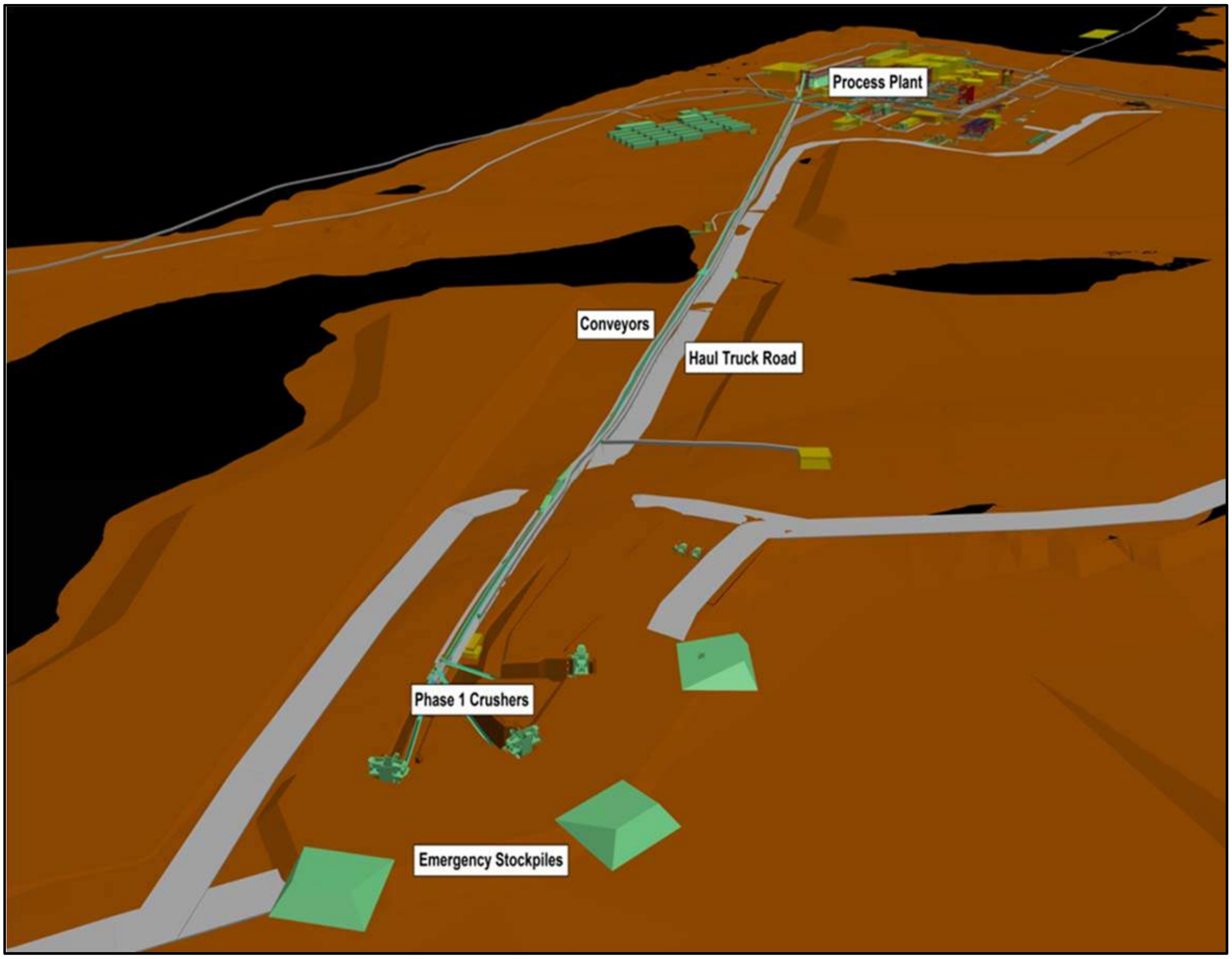
17.4.3 Ore Conveyors

The discharge apron feeder of each primary crusher, whether Phase 1 or Phase 2, moves the crushed ore to a transfer belt conveyor. This belt conveyor discharges onto a main overland conveyor through an open transfer tower.

In Phase 1, two main overland conveyors link the three primary crushers to the process plant (Figure 17.8). Each of the three primary crusher conveyors can deliver material to either one of the two main overland conveyors.

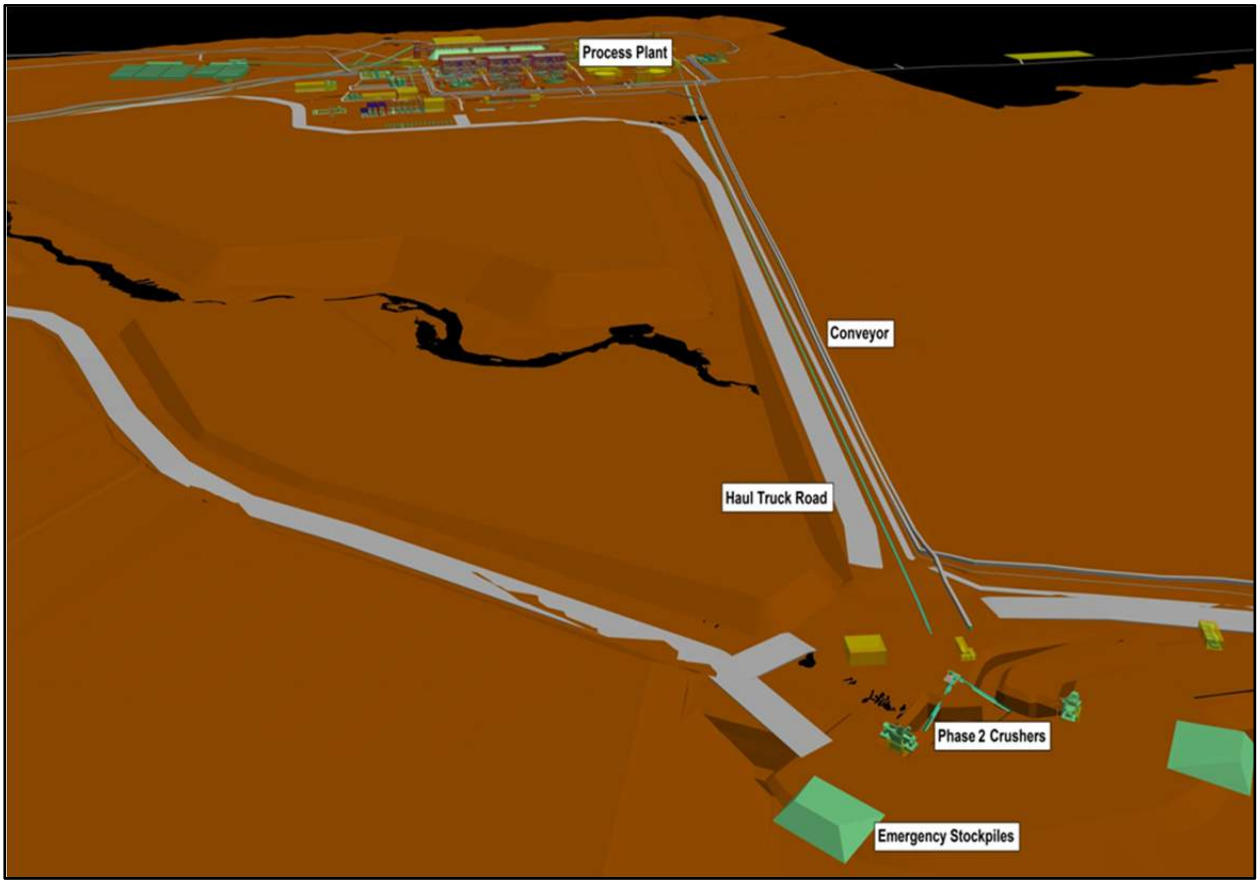
The two parallel main overland conveyors in Phase 1 are approximately 4 km long. Each conveyor's nominal capacity is 9,231 t/h (dry basis), while the design capacity is 12,308 t/h.

Figure 17.8 – Phase 1 Main Overland Conveyors



In Phase 2, the two additional primary crushers are serviced by a new main overland conveyor to the plant (Figure 17.9). Each Phase 2 primary crusher conveyor delivers material to the new main overland conveyor through an open transfer tower.

Figure 17.9 – Phase 2 Main Overland Conveyor



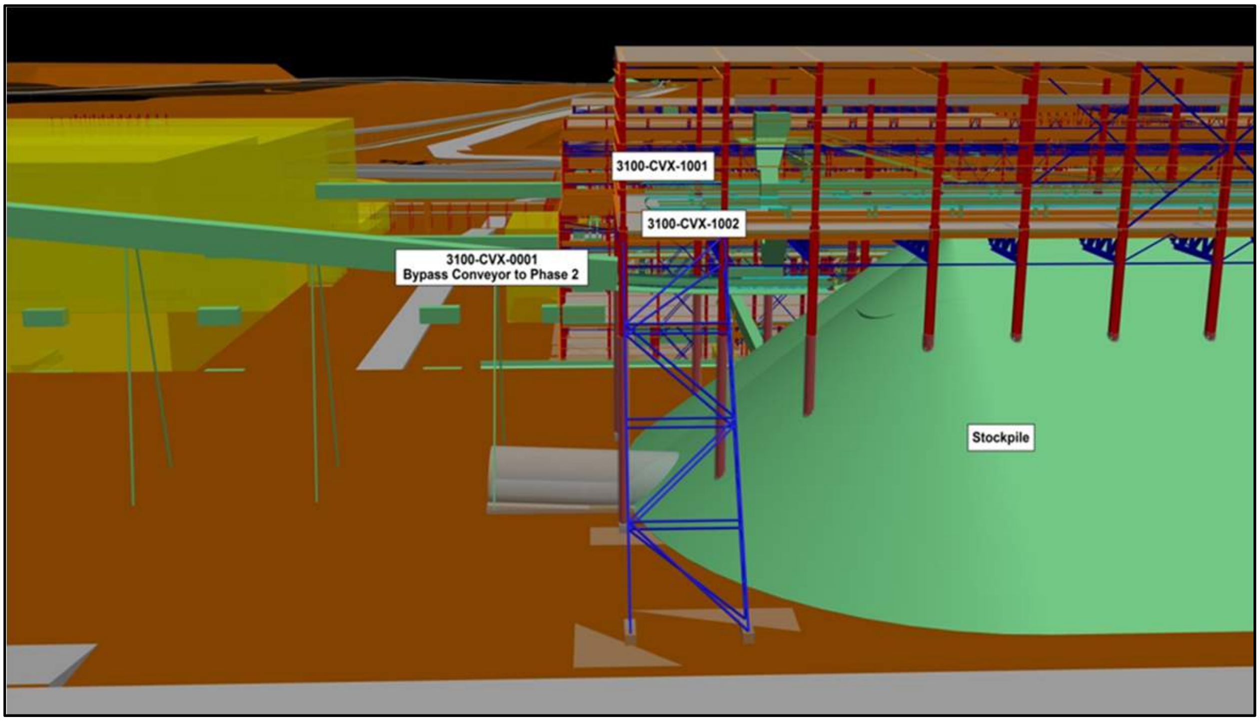
For Phase 2, one main overland conveyor, approximately 4 km long, is installed to feed the plant stockpile. The conveyor's nominal capacity is 12,308 t/h and design capacity is 12,923 t/h.

The overland conveyors are not covered. Each conveyor's drive is located in the open transfer tower near the feed end. A small covered area is provided for belt repairs.

At the process plant, the overland conveyors discharge the ore onto belt tripping conveyors installed on the top of the plant stockpile. The tripping conveyors are inside a gallery on a partially covered structure.

To ensure ore supply redundancy for Phase 2, a transfer conveyor enables the bypassing of ore, when required, from Phase 1 to Phase 2 belt tripping conveyor. The two tripping conveyors and the bypass transfer conveyor (tagged 3100-CVX-0001) are shown in Figure 17.10

Figure 17.10 – Plant Stockpile with Two Tripping Conveyors and a Bypass Transfer Conveyor



Water mist generators are installed at each conveyor transfer point to control dust generated.

Magnetized coils located at the first conveyor leaving each primary crusher will be activated when tramp metal is detected on the belt and will transfer the tramp metal to a bin next to the conveyor.

Cameras installed at critical locations along the conveying system will provide a remote visual overview of the conveyor operation.

17.4.4 Ore Stockpile and Reclaiming System

The ore stockpile located on the north end of the process plant is designed for 16 hours (equivalent to approximately 380,000 t for both phases combined) of live ore storage at nominal capacity. The volume capacity of dead material storage is about five times greater than the live ore capacity, for a total capacity (dead and live) of approximately 2.3 Mt for both phases.

Each reclaiming line consists of three (3) reclaim apron feeders, one belt conveyor, a main ore belt conveyor, dust suppression system, a rod gate, hydraulic safety gate, and sump pump.

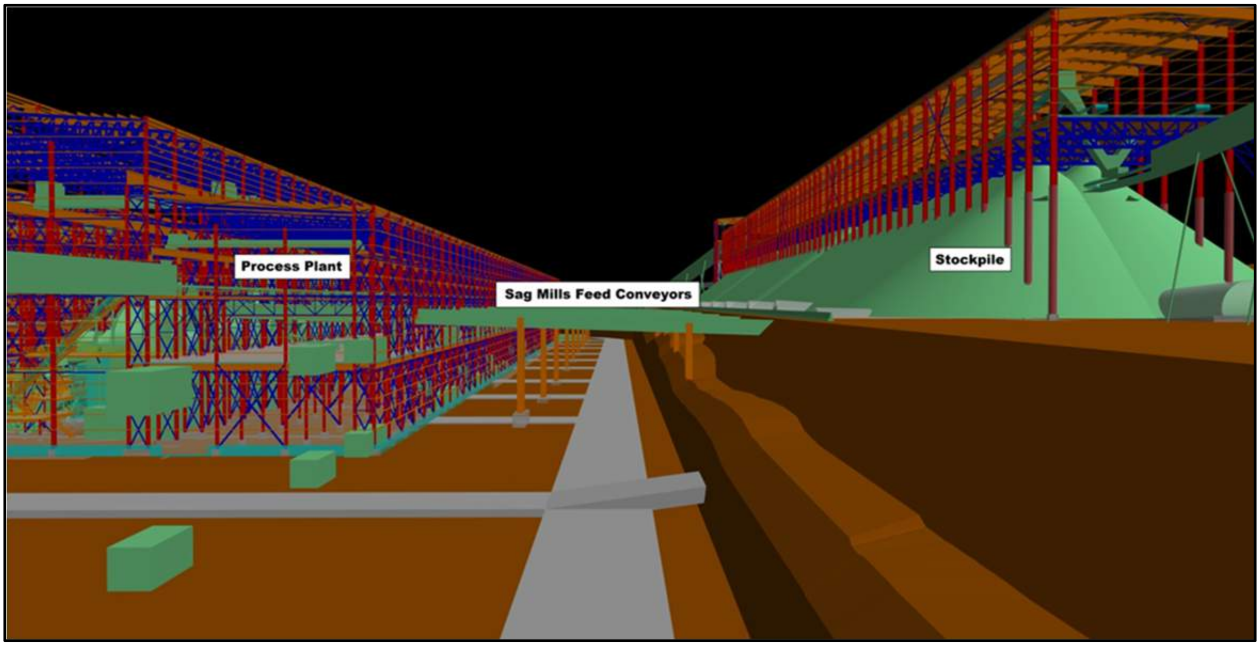
For each ore-reclaiming line, three openings, complete with chutes with rod gates, are provided underneath the stockpile and above the apron feeders.

The design capacity of the apron feeders allows for redundancy to achieve 100 % of each SAG feed line rate with only two feeders in operation while the third is out of service.

In Phase 1, an enclosed steel structure at the bottom of the stockpile houses 27 apron feeders for 9 ore-reclaiming lines installed in individual tunnels. In Phase 2, there are 18 apron feeders for 6 ore-reclaiming lines installed in individual tunnels.

As shown in Figure 17.11, the stockpile is situated higher than the process plant, such as to reduce the amount of earthworks cut-and-fill material and to provide a shorter conveyor distance between load and discharge facilities.

Figure 17.11 – Stockpile Reclaiming Arrangement



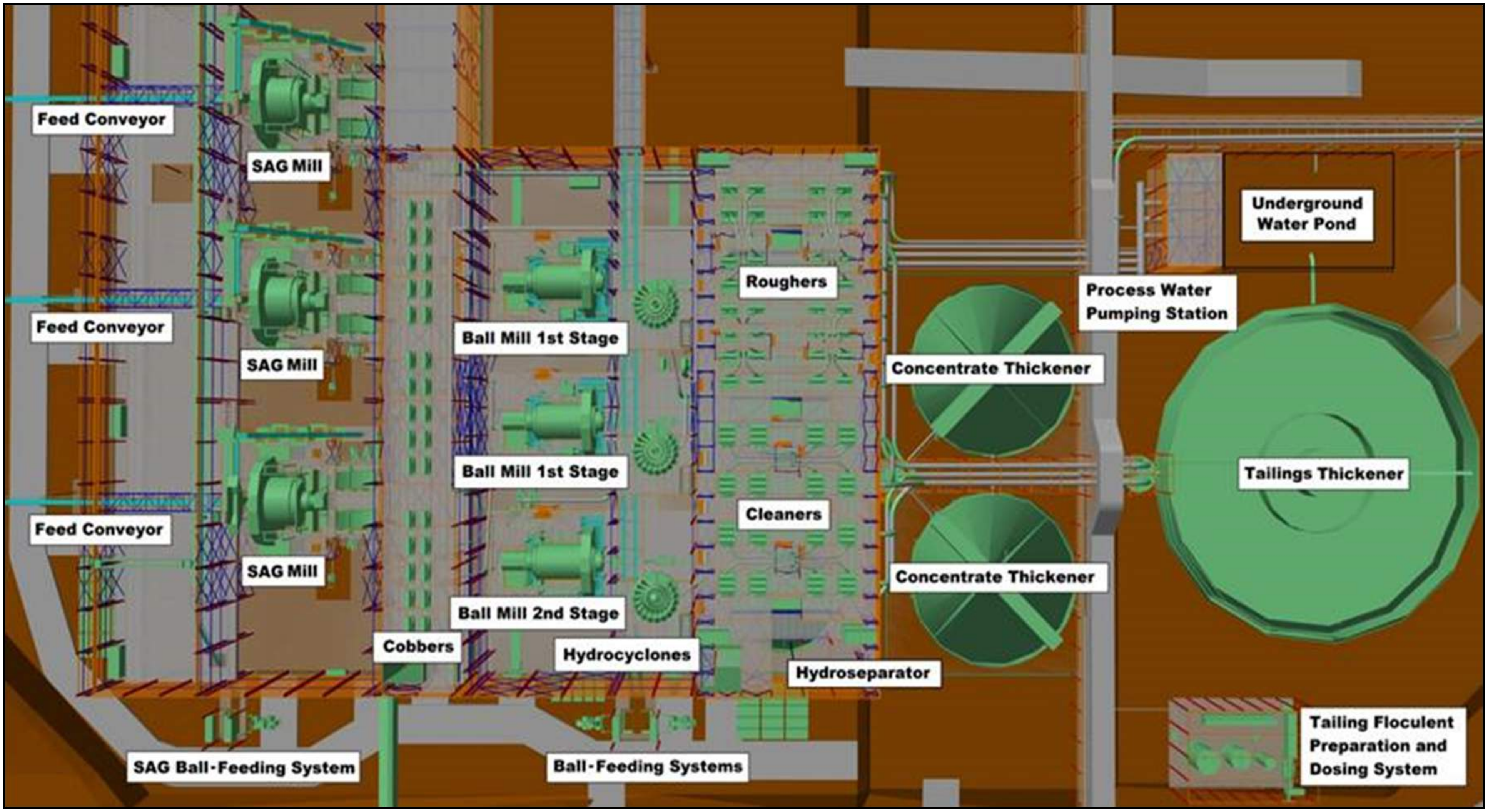
Steel tunnel construction is the safest and preferred solution for the design of ore reclaiming from stockpile. At the exit of each ore reclaiming tunnel, a main ore belt conveyor is installed in a gallery that extends into the process plant and discharges ore into the SAG mill feed hopper.

17.4.5 Process Plant General Description

In Phase 1, three independent ore processing trains are installed. Each train produces nominally 10 Mt/y of iron ore concentrate (dry basis).

Figure 17.12 – Typical Processing Train Arrangement shows a typical overall train arrangement from the feed conveyor to the concentrate and tailing thickeners.

Figure 17.12 – Typical Processing Train Arrangement



In Phase 2, two independent ore processing trains are installed. Each train also produces nominally 10 Mt/y of iron ore concentrate.

Each train consists of the following major equipment or systems:

- Three (3) SAG mills;
- One common steel ball feeding system for all SAG mills;
- Vibrating screens;
- Oversized recirculation conveyors;
- Three (3) hydrocyclones;
- Two (2) first-stage ball mills;
- Steel-ball feeding system for all first-stage ball mills;
- One (1) second-stage ball mill;
- Steel ball feeding system for all second-stage ball mills;
- Three stages of magnetic separators (cobber, rougher, and cleaner LIMS);
- Three (3) hydro-separators (desliming thickeners);
- Two (2) concentrate thickeners;
- One (1) tailings thickener;
- Coarse tailings hydrocyclone;
- Process water pond and pumping station.

Sump pumps are provided in all buildings where water or slurry is handled. Any slurry or water is pumped to the tailings thickener.

All centrifugal slurry pumps are provided with a pump box. The underflow of thickeners and agitated tanks do not need pump boxes.

Overhead cranes are provided inside the plant building to handle pieces of equipment and for removal of liners and for handling tools.

The tailings and concentrate thickeners are not enclosed in buildings, so lifting of equipment during maintenance is performed using mobile cranes.

The description in Sections 17.4.6 to 17.4.14 here below applies to each train.

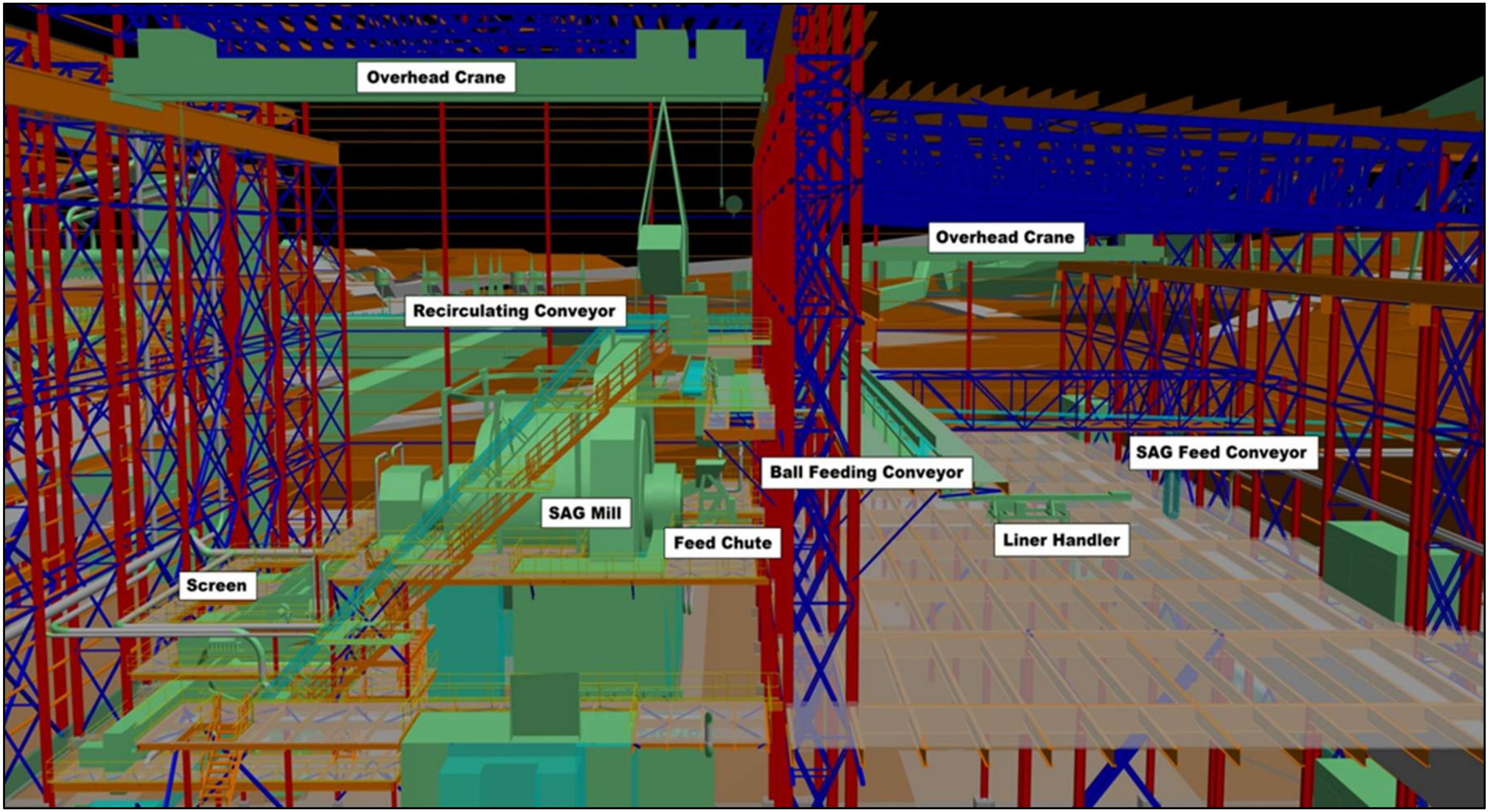
17.4.6 Semi-Autogenous Grinding (SAG) Mills and Classification

Three (3) SAG mills per train, installed in parallel, will reduce ore size to a P_{80} of about 800 microns.

Each SAG mill is designed for a fresh iron ore feed of 1,755 t/h. The calculated required power for each SAG mill is 23.5 MW. The SAG mill dimensions are 12.2 m (40 feet) diameter and 7.6 m equivalent grinding length (EGL), which is the biggest size of proven technology operating today. The SAG mill is driven by gearless drive motor (also called wrap-around motor).

The SAG mills are provided with all auxiliaries for lubrication and tools for liner replacement. An automatic ball-feeding system shared by all the SAG mills consists of conveyor and diverter gates. The ball-feeding system will supply 125 mm diameter balls to the SAG feed chutes at a regular and controllable feed rate. Figure 17.13 presents the typical arrangement of a SAG mill and its auxiliaries.

Figure 17.13 – SAG Mill Arrangement



The product from the SAG mill is discharged into a diversion box that splits the flow into two parallel 3.35 mm cut size screens.

Oversize material, assumed at 30 %, is recirculated via conveyors to the feed chute of the SAG.

Product that passes the screens flows to a pump box and is pumped to the next separation stage.

17.4.7 Cobber Low Intensity Magnetic Separators (LIMS)

The underflow of the screen is pumped to distribution boxes that equally feed cobber LIMS which are installed in parallel. Each production train features thirty (30) cobber LIMS, six (6) of which are on stand-by. These single-stage counter-flow LIMS separate magnetic material from non-magnetic material. The underflow of the screen is pumped to distribution boxes that equally feed cobber LIMS which are installed in parallel. Each production train features thirty (30) cobber LIMS, six (6) of which are on stand-by. These single-stage counter-flow LIMS separate magnetic material from non-magnetic material.

Magnetic material will flow by gravity to the next stage of grinding (first-stage ball mill), as shown in Figure 17.14.

17.4.8 First-Stage Ball Mills and Classification

On each train, two (2) ball mill circuits will operate in parallel in a closed circuit with a hydrocyclone cluster, operating with a recirculation rate of 250 % of the underflow.

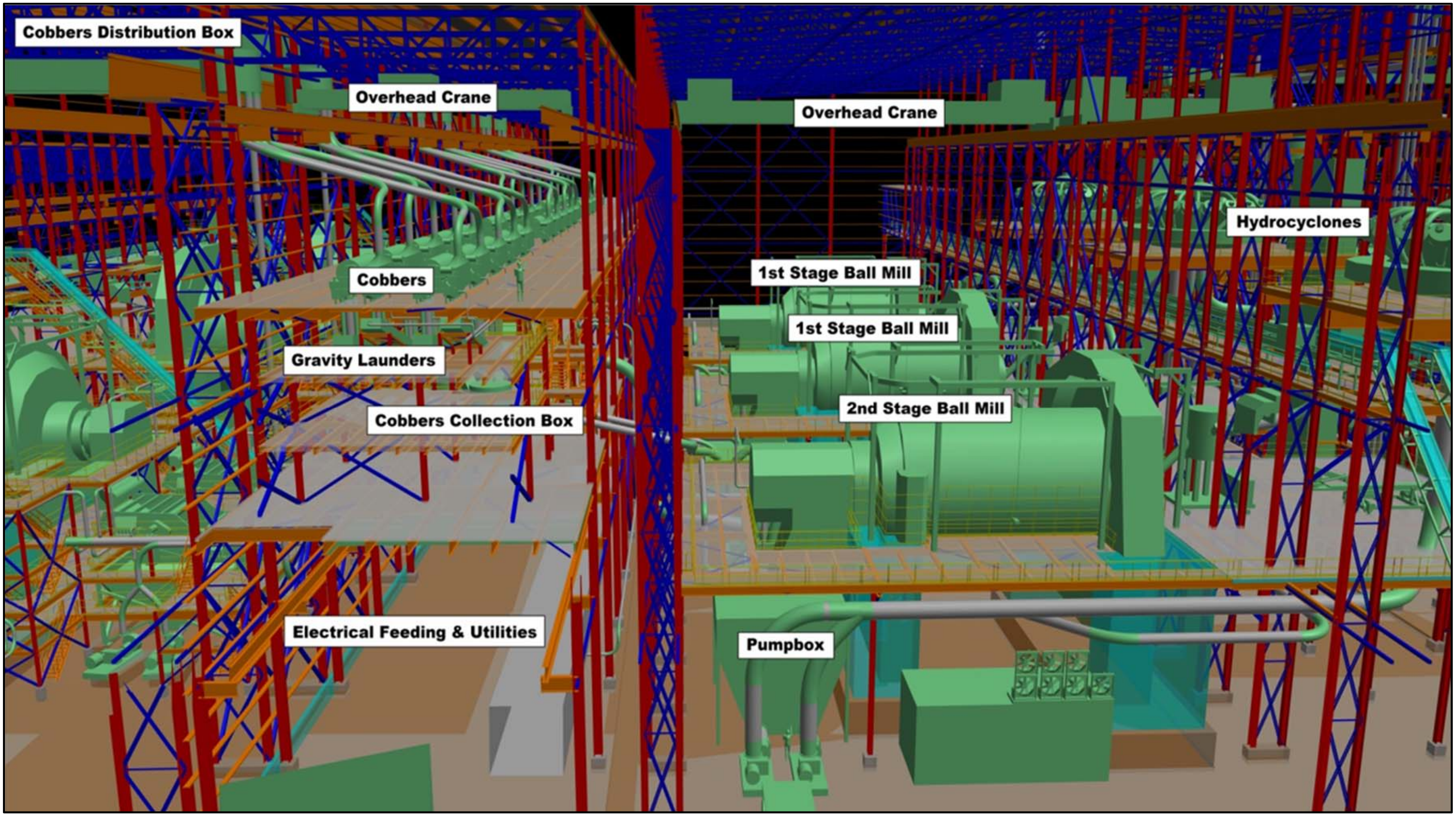
The overflow of the hydrocyclones will be transferred to the next stage of magnetic separation (rougher LIMS).

Each first-stage ball mill is designed for a feed rate at 4,260 t/h. The ball mills are gear driven with synchronized dual pinion. The calculated required power for each of the two pinion-drive motors is 7,650 kW. The ball mill dimensions are 7.9 m (26 feet) diameter and 12.66 m EGL. The ball mills are provided with all auxiliaries and tools for liners replacement.

An automatic ball-feeding system (similar to that of the SAG mill and shared by all first-stage ball mills) will convey 50 mm diameter balls to each feed chute at a regular and controllable feed rate.

Similar to the SAG building, the area is serviced with an overhead crane and liner-handling tools. Figure 17.14 below presents the arrangement of the cobber LIMS and the ball mills.

Figure 17.14 – Cobber LIMS and First-Stage and Second-Stage Ball Mills



17.4.9 Rougher LIMS

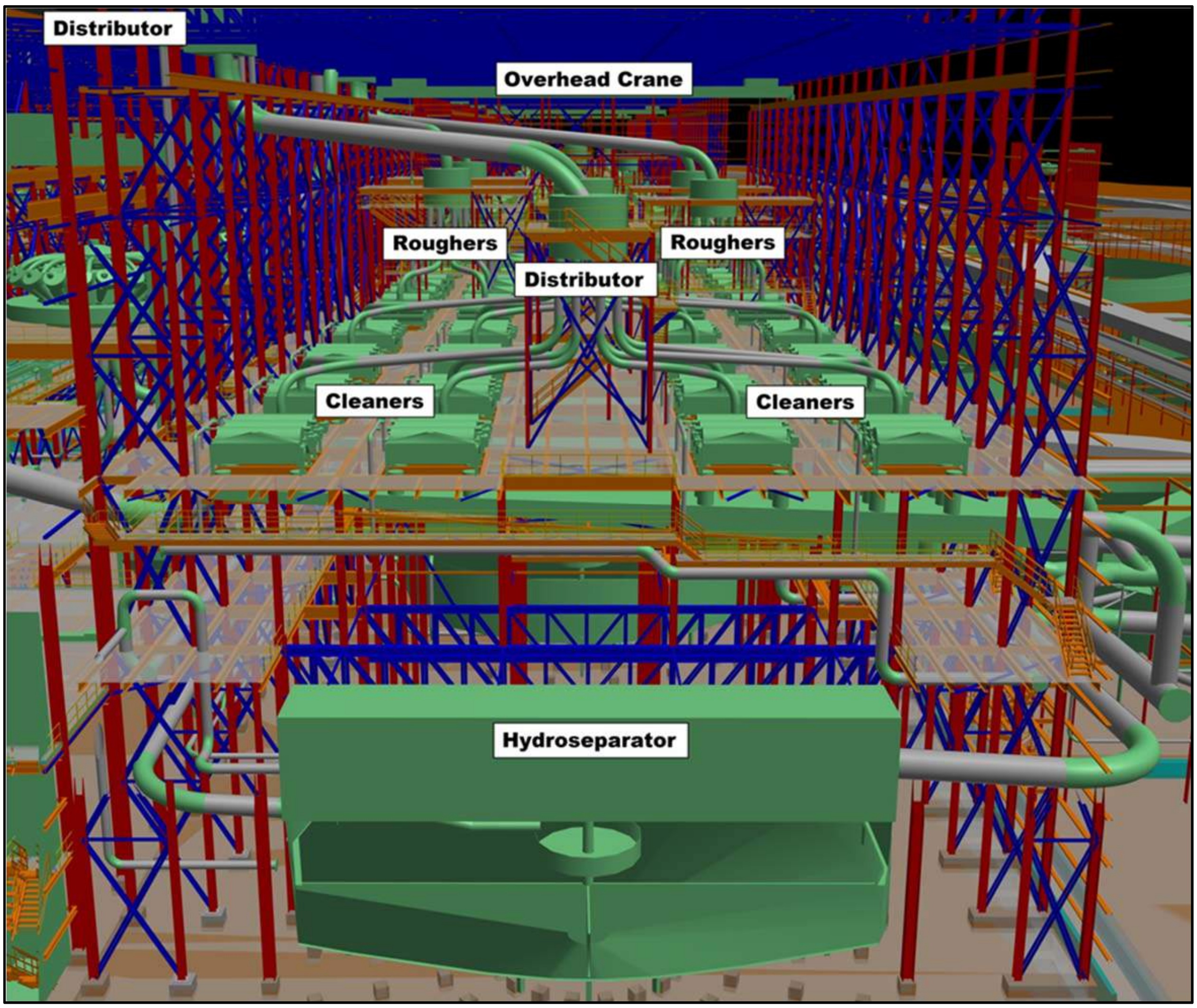
Similar to the cobber LIMS, the rougher LIMS recovering magnetic concentrate are single-stage, counter-flow type but they operate at a lower intensity. Two (2) levels of distribution boxes direct the slurry to the LIMS. Each production train features 32 rougher separators, four (4) of which are on stand-by.

Non-magnetic material (rougher tailings) will flow by gravity to the tailings thickener.

Magnetic product will be re-slurried and pumped back to the second-stage ball mill.

Figure 17.15 below shows the location of the rougher LIMS.

Figure 17.15 – Rougher and Cleaner LIMS and Hydro-Separators



17.4.10 Second-Stage Ball Mill and Classification

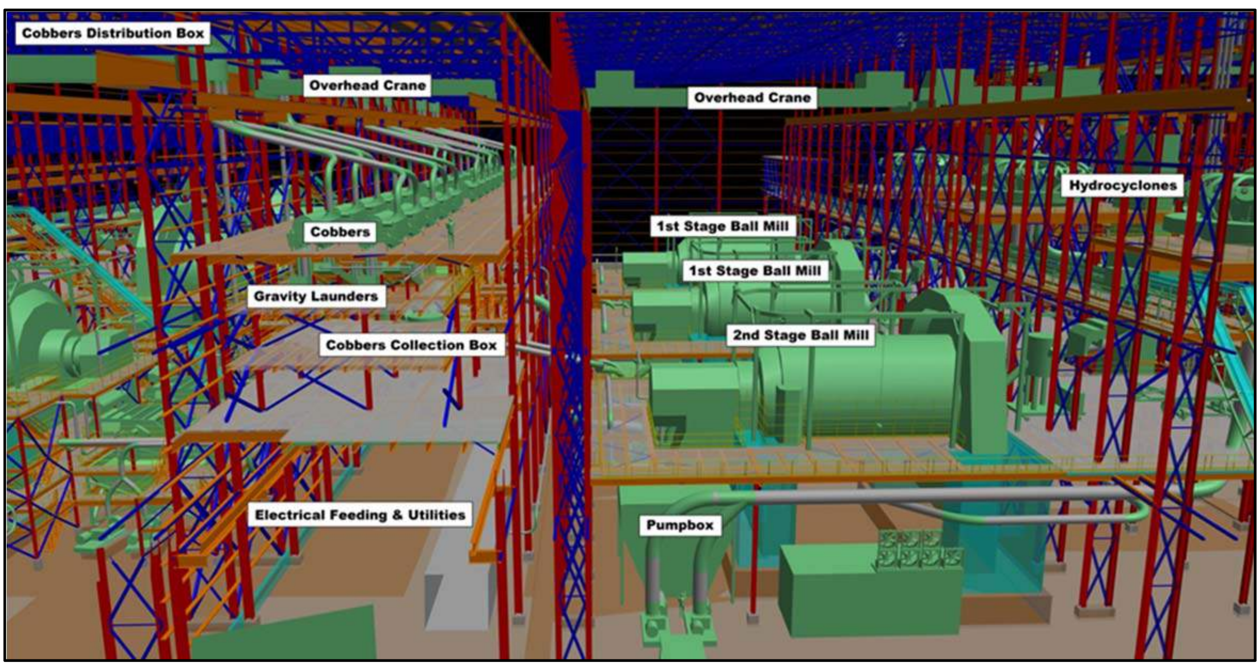
On each train, a single second-stage ball mill will operate in closed circuit with a cyclone cluster from where the overflow product, D_{80} = about 48 microns, will flow by gravity to the hydro-separators (also called de-sliming thickeners).

The second-stage ball mill is designed for a feed rate of 6,026 t/h. It is driven by a 25 MW gearless motor wrapped around the rotating shell. The ball mill dimensions are 8.5 m (28 feet) diameter and 15.44 m EGL. The second-stage ball mill is provided with all auxiliaries and tools for liner replacement.

Since the first-stage and second-stage ball mills are located in the same area, they share the same overhead cranes and liner handler. This facilitates maintenance, operation, and access.

An automatic ball-feeding system shared by the second stage ball mills of every train, will convey 28 mm diameter balls to each feed chute at a regular and controllable feed rate. Figure 17.16 shows the location of the second stage ball mills.

Figure 17.16 – Cobber LIMS and First Stage and Second Stage Ball Mill



17.4.11 Hydro-Separators

The overflow from the second-stage grinding cyclone cluster will flow by gravity to a three-way distribution box, which will feed equally the three (3) hydro-separators (de-sliming thickeners). At the overflow, the slimes will discharge by gravity to the tailings thickener. The underflow will be pumped to the cleaner LIMS.

Hatches are provided on the top floor of the rougher and cleaner LIMS area for the purpose of hydro-separator maintenance and for handling access for major pieces of equipment.

Figure 17.15 above shows the location of the hydro-separators, there is only one appearing as the others are just hidden behind.

17.4.12 Cleaner LIMS

The cleaner LIMS recovering magnetic concentrate are triple-drum counter-flow type operating at a lower intensity than both upstream magnetic separators.

Each production train features sixteen (16) cleaner LIMS. Since this equipment is less critical, there are no stand-by units. At the output of the LIMS, the concentrate will flow by gravity to the concentrate thickening section.

The cleaner LIMS tailings will be collected and will flow by gravity to the tailings thickener.

The cleaner LIMS are strategically located in the same building as the rougher LIMS to facilitate the gravity flow of slurry to the downstream tailings thickener and either to the hydro-separator or concentrate thickener located at lower level. The gravity flow also allows simpler design to equalized slurry flow as they both simply use the principle of distribution boxes and gravity lines.

Since the cleaner LIMS and rougher LIMS are the same type of equipment, regrouping both circuits on the same floor and in the same area is also more convenient for operation and maintenance.

Figure 17.15 above shows the location of the cleaner LIMS within the production train.

17.4.13 Concentrate Thickeners

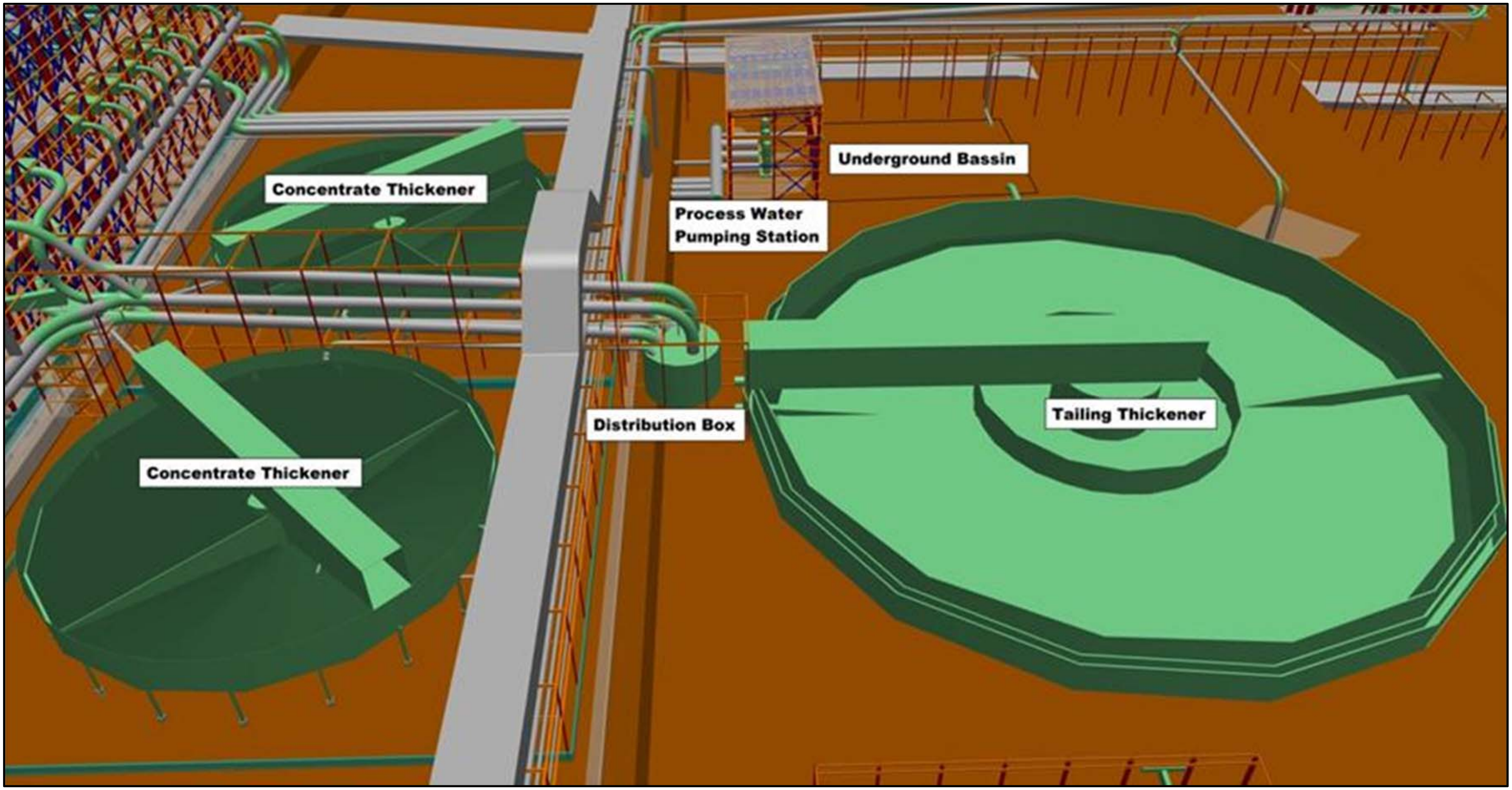
Concentrate from the cleaner LIMS will flow by gravity to a collection box that will feed an automatic sampler. Next, a two-way distributor feeds two (2) concentrate thickeners. Each concentrate thickener has a 45 m diameter. No flocculent is added to the concentrate thickeners.

The overflow will flow by gravity to the tailings thickener inlet box located nearby. The arrangement of concentrate thickeners on each train is shown in Figure 17.17 below.

The underflow at about 67 % solid content will be pumped to slurry concentrate storage tanks that are part of the product delivery system (PDS). One common (PDS) will service all three Phase 1 process trains, while another PDS will service the two additional Phase 2 trains.

Section 14 provides a detailed description of the PDS system that is required to bring the concentrate from the Lac Otelnuk Iron Ore Project mine site to the Port of Sept-Îles.

Figure 17.17 – Concentrate and Tailings Thickeners



17.4.14 Tailings Thickener and Slurry Tailings Pumping

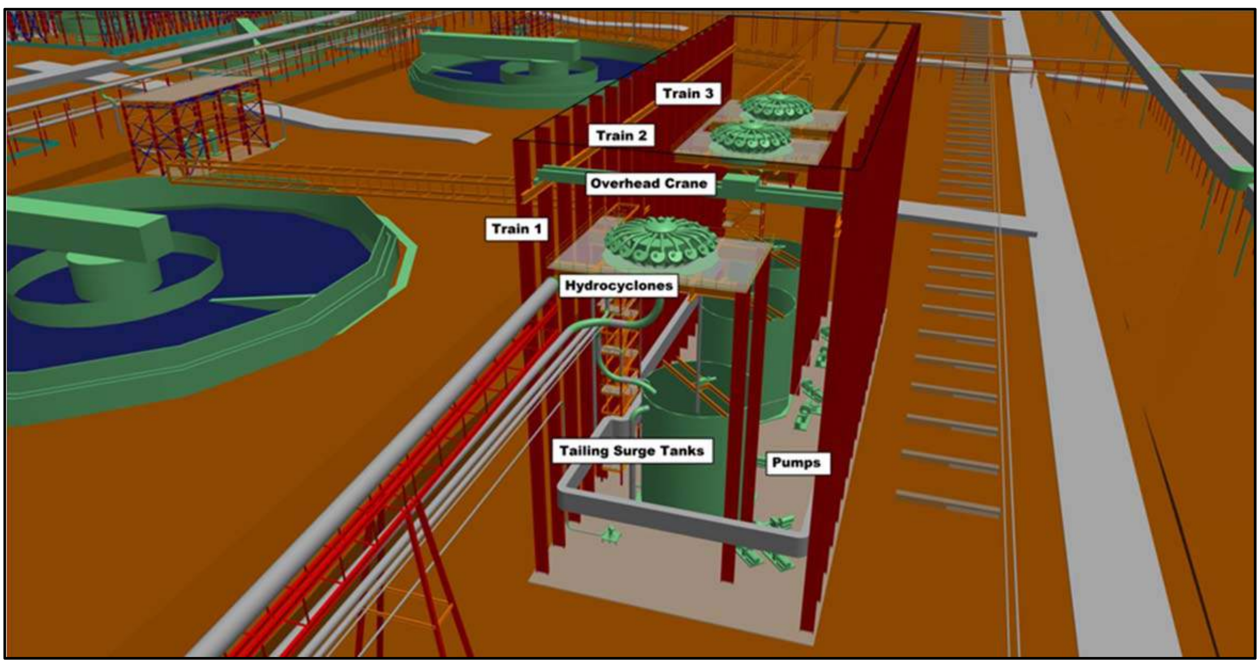
One tailings thickener is installed per train. The thickener diameter is 87 m.

The underflow of the thickener is pumped into a tailings surge tank located in the nearby tailings pumping station. In the tailings surge tank, the coarse cobber tailings fraction is mixed with the tailings thickener underflow. From this agitated surge tank, a pumping system discharges the tailings to the wet tailings deposition pond. The tailings slurry content is designed at 60 % solids.

For each phase, one tailings pumping station is installed, housing all associated train surge tanks, tailings hydrocyclones, and pumping and piping systems. An overhead crane is provided inside the building for maintenance purposes.

Figure 17.18 shows the detailed arrangement of cobber hydrocyclones, tailing surge tanks, and the pumping system for Phase 1.

Figure 17.18 – Phase 1 Tailings Surge Tank and Pumping System



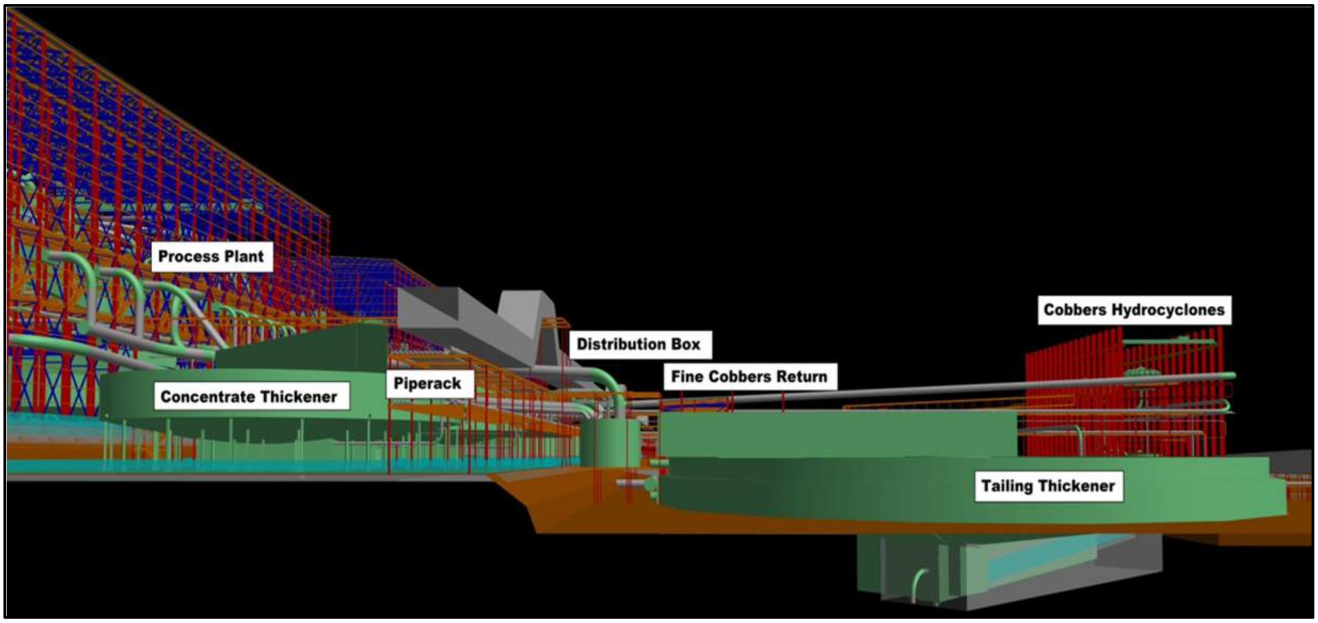
One tailings pipeline is provided per production train. An additional and shorter spare tailings pipeline (about 500 m long) will be available to be shared between all the production trains. It will be used as required in case of an emergency or in a transition period but not on a continuous basis.

The tailings pipeline discharge arrangement will periodically change based on tailings deposition plan and progress. Refer to Section 18 for more detail on the overall strategy of tailings deposition.

The clear water from the tailings thickener overflow will flow by gravity through a launder to the process water pond. The cross-section view in below shows the relative elevations of the process plant and the concentrate and tailings thickeners.

Figure 17.19 below shows the relative elevations of the process plant and the concentrate and tailings thickeners

Figure 17.19 – Elevation of Concentrate and Tailings Thickeners



17.4.15 Tailings Flocculent Preparation and Dosing System

Flocculents are assumed to be delivered to the plant in dry form in 750 kg bulk bags. The material will then be transferred to a storage silo. Conventional flocculent preparation packages will be used for mixing and dilution with water.

One common flocculent preparation and dosing system will be installed to fulfill the requirements of all Phase 1 and Phase 2 trains.

Dosed solution is delivered to each tailings thickener by metering pumps. An overhead crane is provided in the tailing flocculent preparation building for the material handling and for equipment maintenance. For the location of this building see the Figure 17.12.

17.4.16 Process Water Management

About 94 % of the normal process water flow rate need is provided by the tailings thickener overflow. For each production train, process water will overflow from the tailings thickener into the process water pond. This pond is located next to the tailing thickener and at slightly lower elevation.

Make-up water from the reclaimed water of the TMF (tailings management facility) and from WSP (water supply pond) will be pumped to each process water pond. The flow rate from each water supply will be adjusted to maintain a proper water level in each pond.

Four (4) vertical turbine pumps (3 operating and 1 on stand-by) pump water to the various process-water feed points, mainly for slurry dilution and flushing equipment needs of the SAG mill, ball mill, LIMS, thickeners.

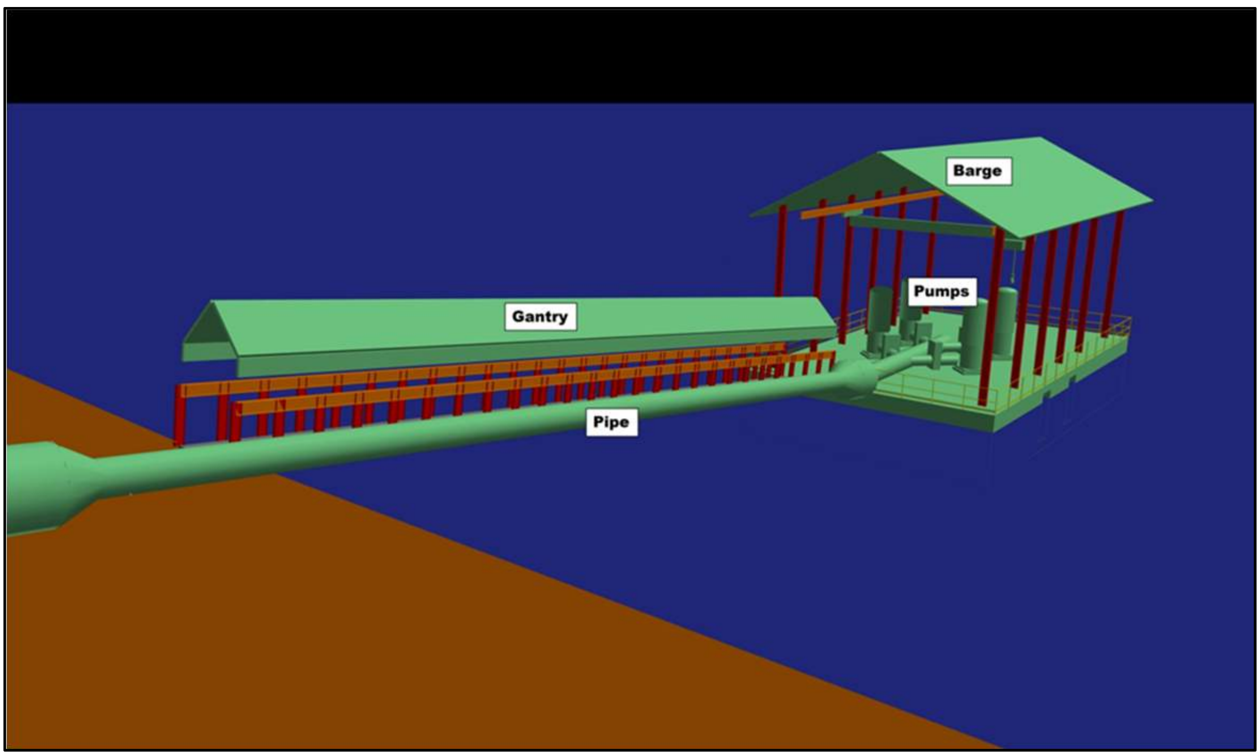
The location of the process water pumping station is shown in Figure 17.16 above.

In general, most of the water flow leaving the process plant area to the TMF and to the PDS is made up from the TMF pond water reclaim and from the water supply pond. The proportion of each make up depends on the phase water balance.

Water will be recycled from the TMF by the use of floating barges on the TMF water pond. Barges will have to be moved to follow the water level progression. There is one barge provided per phase.

The barge is illustrated on Figure 17.20.

Figure 17.20 – TMF Water Recycling Using a Floating Barge



17.5 Plant Control System

The plant control system (PCS) is based on programmable logic controller (PLC) and supervisory control and data acquisition (SCADA) technology on a redundant communication network. An uninterruptible power supply (UPS) system is provided to feed control system equipment. Motors are controlled and monitored via a digital bus network using smart relays provided in motor control centres (MCC).

The plant is operated and monitored from a central control room (CCR) located in the administration complex. Local control rooms (LCRs) are provided for crushers and each process plant train. The CCR houses a total of four operator interface stations (OISs): three to monitor concentrator operation and one for slurry pipeline equipment. Crushers and concentrator trains LCRs house one and two OISs respectively. Generally, all process systems and equipment are started from the CCR/LCRs. Except for crushers, facilities are provided for local operation via a Local-Remote soft select switch provided on the OIS.

Adjacent to the CCR is the technical room that houses the PCS peripheral equipment, primary servers/loggers, and an engineering work station (EWS). To ensure safety of the PCS data in the event of an accident, a set of secondary servers/loggers are provided in the data center. The PCS interface cabinets are installed in electrical rooms. Communication between the various process areas is established via redundant fibre optic cables.

Third party PLCs, supplied as a part of the major mechanical equipment packages, are connected to the PCS via a communication link. This link is used to transfer all process parameters, equipment status and alarms to the PCS.

Concentrator PCS, product delivery PCS and port PCS exchange data via the PDS backbone network.

17.6 Product Delivery System

The concentrate will be transported from the concentrator to the port facilities located in Port of Sept-Îles, where the slurry will be filtered and ship loaded. The description of the product delivery system is presented in Section 18.11.

17.7 Port Facilities

The port area which includes product dewatering, storage, reclaiming, and shipping of the concentrate. The concentrate slurry, at solids content in the design range of 65 % to 67 %, will be delivered by the PDS to terminal station in the Sept-Îles Port Area.

The slurry will be dewatered to achieve a dry concentrate at 8 % moisture content, then transferred to the concentrate storage building, and ultimately reclaimed and conveyed to the ship loader. Section 18.0 provides further physical description of the port facilities.

17.7.1 General Design Criteria for the Port Area

Target production rates are presented in Table 17.5.

Table 17.5 – Production Rates at the Port Facilities

Description	Unit	Value
Availability	%	90
Operating time per year	h	7,884
Hourly throughput at port facilities Phase 1, nominal	t/h	3,425
Hourly throughput at port facilities Phase 1, design	t/h	3,805
Hourly throughput at port facilities Phase 1+ Phase 2, nominal	t/h	5,708
Hourly throughput at port facilities Phase 1+ Phase 2, design	t/h	6,341
Ship Loading design capacity, dry basis, for each phase	t/h	8,280
Ship Loading design capacity, wet basis, for each phase	t/h	9,000

Head pressure dissipation and slurry storage capacity at Product Delivery System receiving terminal are presented in Table 17.6.

Table 17.6 – Slurry Reception and Storage

Description	Unit	Value
Head pressure dissipation – Phase 1	m	1,094
Head pressure dissipation – Phase 2	m	965
Total storage capacity – Phase 1	m^3	25,135
Total storage capacity – Phase 2	m^3	15,081

The dewatering facility design parameters are listed in Table 17.7.

Table 17.7 – Slurry Dewatering

Description	Unit	Value
Concentrate filter feed rate (per train, dry basis)	t/h	1,268
Number of trains – Phase 1	each	3
Number of trains – Phase 2	each	2
Slurry solids concentration from product delivery system (PDS)	w/w%	67
Filter feed solids concentration	w/w%	63
Filter feed, total flow rate	m^3/h	991
Slurry pH		9
Design minimum slurry temperature	°C	8
Filter unit capacity	$t/h/m^2$	0,89
Filter cake moisture	%	< 8
Filtrate, total suspended solids (TSS)	mg/L	<15

17.7.2 Slurry Terminal Station

Slurry arriving from pumping station 3 (PS3) is depressurized and flows to a group of five (5) tanks during Phase 1 of the project and three (3) additional tanks in Phase 2.

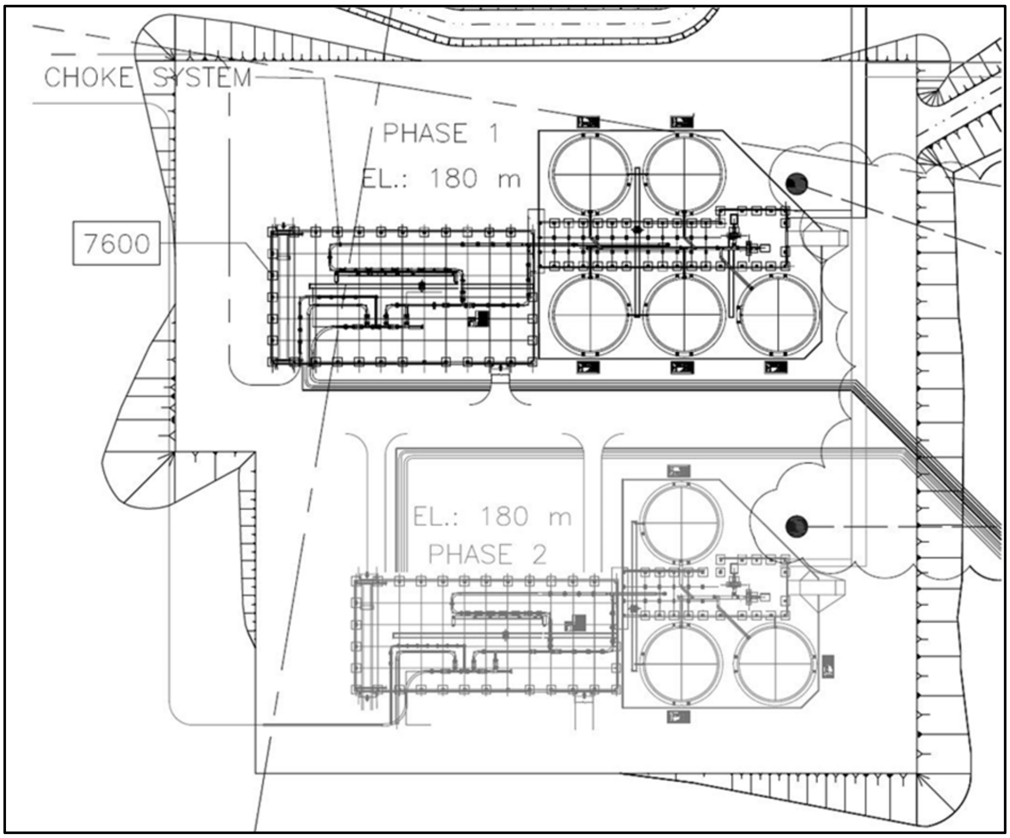
Station-isolation and pipeline-shutdown valves are installed at the entrance to the depressurizing station. Over-pressure protection is provided by a rupture disc that discharges to the emergency concentrate handling. Chokes are available for pressure dissipation. A pig receiver is provided.

Depressurization occurs at the choke station. Choke stations for slurry pipelines consist of ceramic chokes. Chokes force the slurry through a small wear-resistant orifice. The resultant turbulence and high flow velocities produce a high-pressure drop from one side of the choke to the other: up to 50 m of elevation drop per choke, or the equivalent of approximately 10 km of pipeline friction loss in the selected mainline pipe diameter. Loops containing 2 chokes each can be engaged or disengaged individually to provide variable amounts of pressure dissipation, depending on how much is required by the operating condition.

Local instrumentation measures the main process parameters, including incoming flow rate, pressure, and density.

Details of the slurry terminal station and slurry storage are provided in Figure 17.21 - Slurry Terminal Stations here below.

Figure 17.21 – Slurry Terminal Stations



17.7.3 Slurry Storage

Storage tanks are agitated continuously and provide about 9.5 hours of live storage capacity at nominal flow rates for Phase 1 and 8.6 hours for the Phase 2 PDS. Tanks used in Phase 1 are not interconnected with the second phase tanks.

17.7.4 Slurry Dewatering System

The slurry dewatering system is designed and sized per production train. There are five (5) trains in total (3 trains in Phase 1 and 2 trains in Phase 2). One set of filters is added for each train (n + 1) for increased availability. All systems are on emergency power in the case of a power outage.

The filters are ceramic capillary discs used for solid/liquid separation. The micro-porous structure of the ceramic filter media allows water to pass through, but not the solids. The disk pores do not allow the solid particles to enter. As a result, energy consumption is low and the filtrate is clear and particle-free.

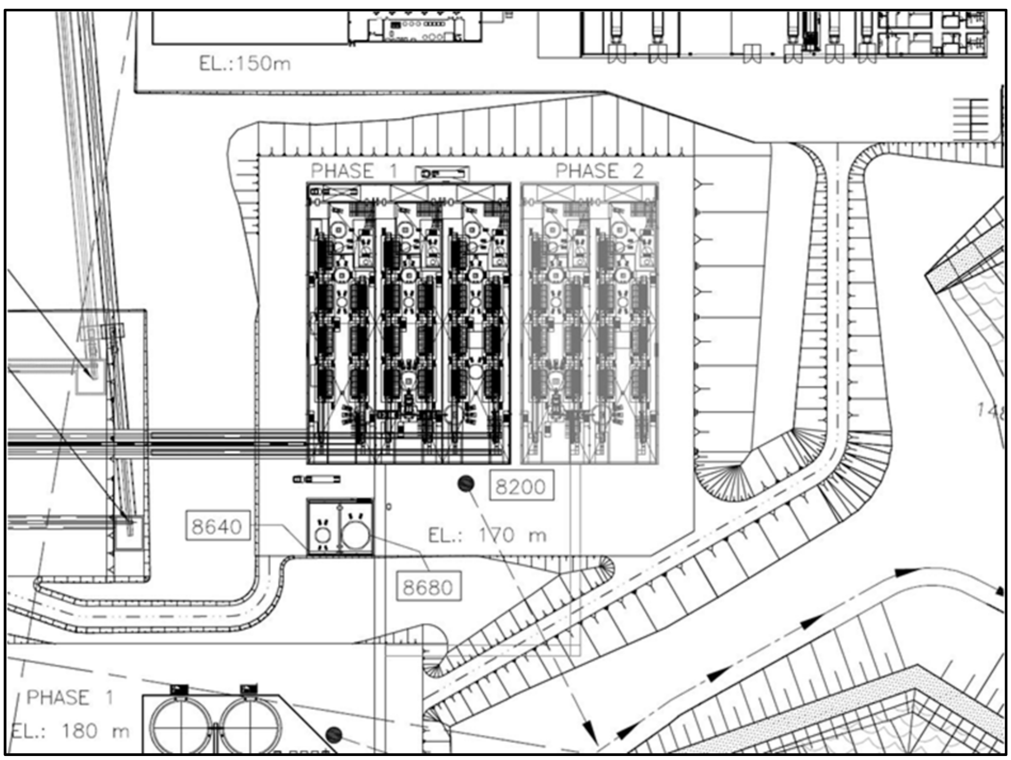
The filtration process proceeds as follows. The slurry at 63 % solid concentration (by weight) is fed into the slurry feed basin of the filter. At the beginning of the rotation cycle, the plates are submerged in the slurry. Vacuum inside the plates forces liquid through the microscopic pores of the ceramic filter media, and the solids agglomerate on the disk surface. As the disks rotate above the slurry, the vacuum continues to draw water and the cake dries up. The filtrate (water from the dewater process) is collected into a filtrate tank and then pumped to the water management. The filtrate contains only traces of solids that have a TSS of less than 12 mg/l.

The dewatered section of the cake is discharged from the rotating plates by scrapers. The cake at 8 % or less moisture content falls onto a conveyor belt and is carried to the concentrate stockpile.

Backwash cleans the plates once every rotation cycle. On a daily basis, the plate surface tends to accumulate deposits that reduce the capillarity of the filters and hinder production. It is therefore necessary to wash the plates periodically with a combination of ultrasonic washing and an acidic washing solution. The acid is re-circulated to minimize the quantity to be disposed of. When the combined washing is completed, the acid is recycled and normal filtering operation can resume.

The general layout of the dewatering facility is provided in Figure 17.22 below.

Figure 17.22 – Slurry Dewatering



17.7.5 Filtrate Water Characterization

Dewatering of the concentrate slurry will continuously produce large quantities of water (filtrate). It is paramount that water management facilities be built to ensure clean discharge to the environment and to provide a measure of handling upsets in the process and prevent uncontrolled spills.

A fraction of the filtrate produced from the dewatering process will be re-used as process water at the port. The excess will be discharged to the St-Lawrence River. Based on the detailed analysis of actual filtrate samples obtained from filtration tests, this effluent is expected to meet applicable discharge criteria set out in the *Ministère du Développement durable, de l'Environnement et des Parcs* ("MDDEP") Surface water environmental guidelines. A complete analysis of the chemical composition of the actual filtrate water was performed and shown not to exceed the applicable standards. Refer to Section 18.0 for further details on water management.

17.7.6 Consumables and Utilities-consumption overview

Consumptions figures will depend on the characteristics of the ore but typical values are provided in Table 17.8.

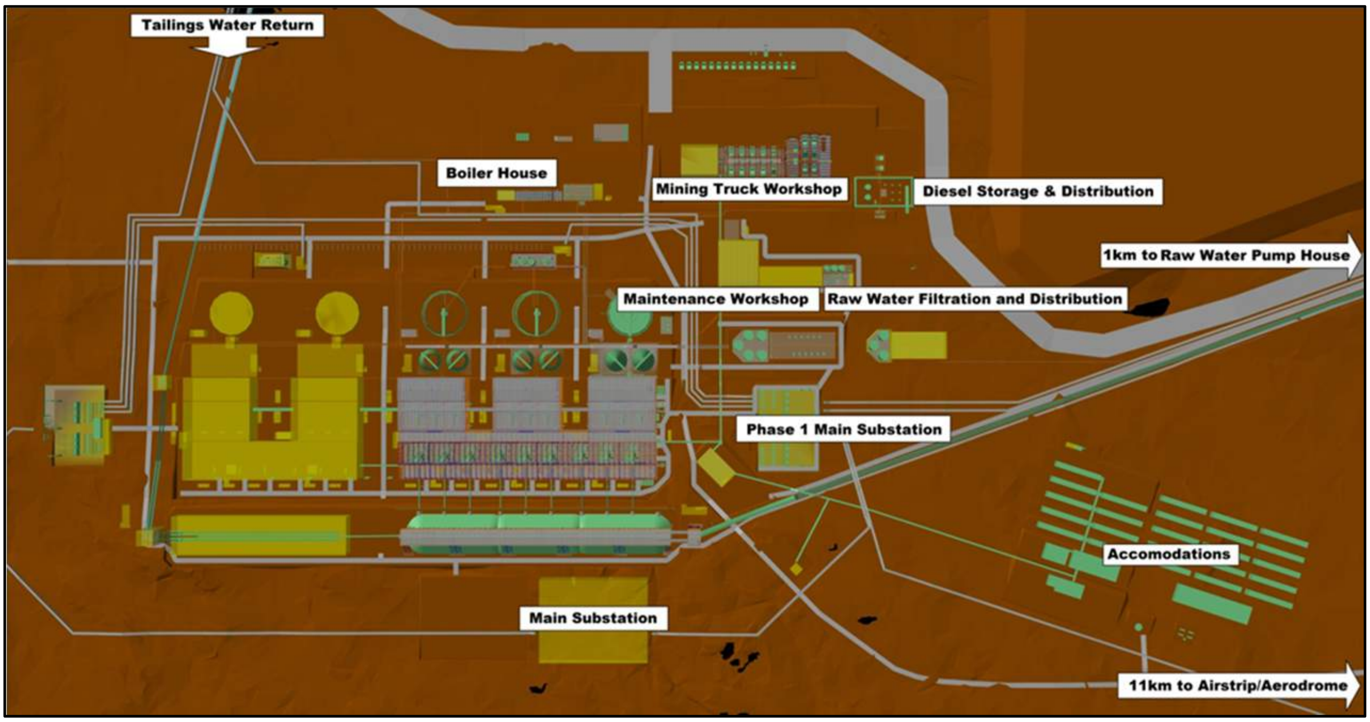
Table 17.8 – Typical consumptions of utilities

Utility	**Unit**	**Value**
Flocculant	Kg/t tailing	0.05
SAG grinding media	Kg/t concentrate	2.83
Ball mill 1st stage grinding media	Kg/t concentrate	1.34
Ball mill 2nd stage grinding media	Kg/t concentrate	0.69
Electrical consumption, beneficiation	Kwh/t concentrate	136
Fuel, mobile equipment in beneficiation plant	l/t concentrate	0.20

18.0 PROJECT INFRASTRUCTURE

This section provides a summary of the infrastructure and logistic requirements for the project, which includes power transmission & distribution, access roads, communication & automation systems, support infrastructures and utilities, camp site accommodations, airstrip, tailings management facility (TMF), dams, water management system, stockpiles, product delivery system (PDS) and port facilities, required for the Lac Otelnuk deposit located in the Nunavik region of the province of Quebec, about midway north in the Labrador Trough iron range. Figure 18.1 provides an overview of the major infrastructure location

Figure 18.1 – Overview of Infrastructure Location



18.1 Power Transmission Lines

It is expected that three (3) incoming power transmission lines will be supplied, one at each of the main Project locations these being the mine and process plant, PDS pump station PS3 and the Port Facilities in Sept-Îles.

18.1.1 Power Interconnection Overview

The Lac Otelnuk Iron Ore Project requires an estimated power supply of 1,078 MW for operation at full capacity at the mine and process plant area.

Lac Otelnuk Iron Ore Project interconnection with the Hydro-Québec transmission network is based on supplying power to the Project at 735 kV from the existing Tilly substation located less than two (2) km from La Grande-4 (LG-4) power generation station and 466 km from the Project Site. Figure 18.2 below shows the location of the high voltage power interconnection.

Figure 18.2 – 735 kV Power Interconnection Location

Source: Hydro-Québec 2008

The 735 kV power interconnection includes a 735 kV transmission line between the Hydro-Quebec's existing 735/315 kV substation at Tilly and a new 735/230 kV substation at Lac Otelnuk Iron Ore Project. The new substation transfers the power to two (2) new 230/34 kV substations, one for Phase 1 and the other for Phase 2, by means of two (2) 230 kV transmission lines with double circuits and approximately one (1) km long.

The 735 kV overhead transmission line is approximately 466 km long and includes one (1) single circuit, one (1) overhead shield wire, and one (1) optical ground wire.

As per Hydro-Québec (TransÉnergie, September 2014), the power interconnection requires the addition of 330 MVAR inductive compensation at Tilly substation and 330 MVAR inductive compensation at LeMoyne substation. These are the responsibility of Hydro-Québec. There are also two (2) 230/34 kV substations required for Phase 1 and Phase 2 respectively and these are described separately in Section 18.2 of this report.

18.1.2 Equipment Requirements

Based on preliminary analysis, the following list of major equipment is expected to meet the load demand at the ultimate stage:

- 1 x 735 kV overhead transmission line (approximately 466 km);
- 2 x 735/230 kV power transformers capable of carrying the full 1,084 MVA load, while meeting the required reliability criteria, during contingency;
- 1 x 735 kV bay located at Tilly substation;
- 1 x 330 MVAR inductive shunt compensation located at Tilly substation;

- 1 x 330 MVAR inductive shunt compensation located at LeMoyne substation;
- 1 x 330 MVAR + 2 x 165 MVAR inductive shunt compensation located at the load-side;
- 100/+300 MVAR SVC located at the load-side.

18.1.3 735/230 kV Main Substation

The 735/230 kV substation has the principal function of receiving the electrical power from the Hydro-Québec Tilly substation at 735 kV and transferring it to the plant substations at 230 kV.

The main substation consists of two main sections, the 735 kV section and the 230 kV section. They are located next to each other but on two different elevations in order to facilitate construction adapted to the natural elevations of the area.

The entire substation will be fenced in.

a) 735 kV Section

The 735 kV section including the connection of the main transmission line together with the main autotransformers are located on the upper level. Two sloping overhead spans interconnect the upper and the lower sections.

Also located on the upper level will be the station auxiliary transformers, the main control building, and the shunt reactors.

b) 230 kV Section

The 230 kV section is arranged as a breaker-and-a-half scheme, with two (2) diameters of three breakers each, and provision for a future additional diameter for Phase 2 of the Lac Otelnuk Iron Ore Project. This future additional diameter can serve to connect additional reactive power compensation systems for Phase 2 consisting of an SVC (static VAR compensation) system.

18.2 Power and Energy Systems

18.2.1 Process Plant Power Distribution

Each of the two process plant substations consists of outdoor air-insulated switchgear (AIS), and includes 230 kV switches, circuit breakers, protection and instrument transformers, power transformers and electrical rooms for 34.5 kV Gas Insulated Switchgear (GIS) and 230 kV protection relays.

The substations comprise all harmonic filters required to mitigate harmonics generated by large cyclo-converters used for SAG mills and ball mills.

To supply the process plant, the Phase 1 substation includes nine transformers, 90/120/150 MVA, 230/34.5 kV, and the Phase 2 main substation comprises six transformers of the same rating. The 34.5 kV network is configured for 100% redundancy to allow full operation in the case of a transformer outage.

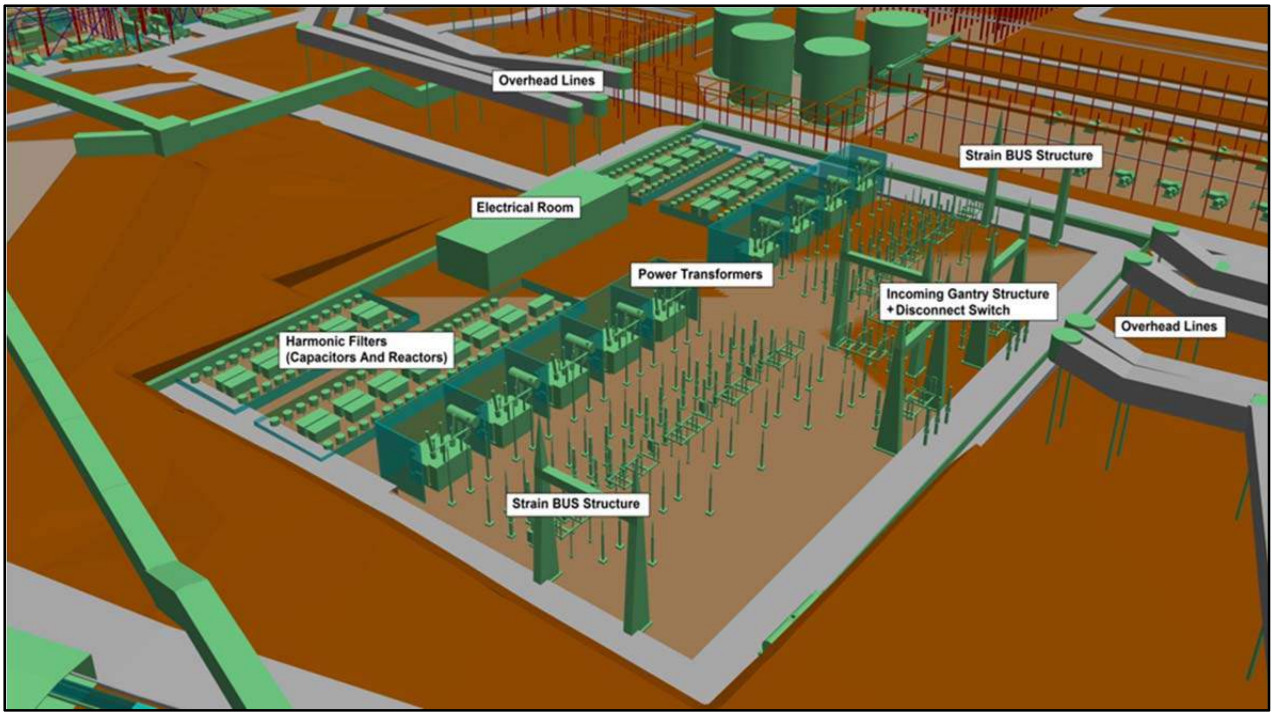
Figure 18.3 illustrates in 3D the general arrangement of the Phase 1 Process Plant substation.

a) Secondary Power Distribution

Electrical rooms in the process plant and in the non-process areas will be supplied at 34.5 kV by cable and overhead lines. Where possible, power cables will be mounted on pipe-racks or in cable trays, to minimize trench work.

Non-process areas located outside the process plant area – e.g. the airstrip, tailings water return pumping station, and the mine site will be supplied by 34.5 kV overhead lines directly from the process plant substation.

Figure 18.3 – Phase 1 Process Plant Substation



The electrical rooms will comprise of a step-down transformer and 34.5 kV and/or 4.16 kV switchgear and a motor control centre (MCC).

Each gearless SAG and ball mills will be equipped with a prefabricated electrical room (ER) containing a cyclo-converter to power the mill's gearless synchronous motor and to control mill speed. The ERs will accommodate all mill control, heating, cooling, lighting and others services. The ERs will have a capacity of 25 MVA, and be fed directly at 34.5 kV. Phase shifting transformers will be localized as close as possible to the ER which will in turn be as close as possible to the SAG/ball mills.

For dual pinion ball mills, a voltage source inverter drive required to maintain equal torque on both pinions will be used and installed in an ER near the mills.

18.2.2 Process Plant Overhead Lines

Two (2) 34.5 kV ACSR (aluminium conductor steel-reinforced) pole-mounted overhead lines will distribute power to the following areas:

- Tailings water return pumping station and booster pumping station
- Garage and warehouse, and diesel storage.

Two (2) 34.5 kV ACSR pole-mounted overhead lines will distribute power to the following areas:

- Airstrip
- Explosive preparation plant building
- Raw water supply pumping station
- Workers accommodation complex.

The electrical power will be distributed to the following area via double circuit 34.5 kV ACSR pole mounted overhead lines, connected to the main power substations located in the concentrator area:

- The water supply pumping house, conveyor transfer towers and primary crushers (Phase 1) will be energized by a 34.5 kV double circuit overhead line, powered by one (1) of the circuit with the second circuit as a redundant power source;
- The conveyor transfer tower and primary crushers (phase 2) are energized by a 34.5 kV double circuit overhead line, powered by one (1) of the circuit with the second circuit as a redundant power source;
- The camp, explosives facility and the airstrip are energized by a 34.5 kV double circuit overhead line, powered by one (1) of the circuit with the second circuit as a redundant power source;
- The garages and the two (2) tailing water tailing return pump houses are energized by a 34.5 kV double circuit overhead line, powered by one (1) of the circuit with the second circuit as a redundant power source.

18.2.3 Mine Site Power Supply

Electric power will be supplied to the mine from two (2) main mine electrical substations (one for each phase of production). The substations will provide power to the fleet of electrical mining equipment which includes shovels, drills and mine dewatering pumps.

The two substations will be located in the general area of the primary crushers and will be connected to the Project's main transformers via 34.5 kV overhead power lines that will run parallel to the conveyors. In order to provide redundancy, the substations will both be connected by two (2) overhead lines.

Each main electrical substation will be equipped with two (2) 15 MVA/20 MVA transformers. The delta-wye connection on the secondary side will be equipped with a 25 A neutral grounding resistor in order to comply with mining code M421 which requires provision of ground fault protection for mobile and moveable equipment.

Each main electrical substation will be connected to four (4) trailer mounted moveable substations via a 34.5 kV overhead power line. The moveable substations will be strategically located at the edge of the open pit to be in the vicinity of the active mining areas. They are expected to be relocated as needed −approximately once a year− to maintain their proximity to shovels and drills.

The moveable substations will be equipped with the following:

- Incoming circuit breaker;
- 10 MVA, 34.5 kV/7.2 kV dry type transformer with a 25 A natural ground resistor;
- Walk-in type switch gear with seven (7) outlets to connect shovels, drills, and dewatering pumping station trailing cable. The couplers are sized at 400 A for shovels and 250 A for drills and dewatering pumping stations. As required by the mining code M421, each output circuit will be protected with measures including ground check and ground fault protection.

The mine electrical infrastructure includes trailing cable, cable stands for haul road crossings, as well as pylons that will be used to lay down the trailing cable.

18.2.4 PDS Pump Station PS2 and PS3

PS2 requires 39.2 MW of at the full nominal capacity of 50 Mt/y of slurry and will be connected to the main plant secondary substation via a 200 km long, 230 kV line.

PS3 also requires 39.2 MW and will be supplied from a 161 kV transmission line originating from Hydro Quebec's substation situated in Fermont Quebec.

18.2.5 Port Facilities

The port facility requires a 27.6 MW at the full nominal capacity. Electric power will be provided by provincial utility Hydro-Quebec from Arnaud Station. The selected interconnection scenario is to build a new 161 kV line (2 km long) in derivation from one of the two 161 kV lines (line no. 1618 or the new 161 kV line to be built in 2015) from the Arnaud 735/315/161 kV station.

The port electrical main substation and electrical room are located on the north side of the dewatering plant and an additional remote electrical is located on the wharf to feed power to conveyors and ship loader.

18.2.6 Emergency Power

a) Process Plant

Emergency power generation for the process plant area is provided by diesel generators (stand-by rated for 4 MW / 5 MVA) in sufficient quantity to supply the emergency power requirements as given in the Electrical Load List. The emergency generators will be operated in parallel on a common bus rated at up to 21 MVA. The emergency power will be distributed by cable at 4.16 kV, powering the medium voltage switchgear in the necessary electrical rooms.

b) Product Delivery System Intermediate Pump Stations

Power back-up for the intermediate pump stations is provided by 4 MW turbo-charged diesel generators each rated at 4.16 kV, as per the following Table 18.1.

Table 18.1 – Number of 4 MW Back Up Generators per Phase

Phase	PS2	PS3
1	5	5
2	3	3

c) Port Facilities

About 2/3 of the electrical load will be on emergency power supply in order to ensure continuity of operation in case of a major electrical power supply shut down. Emergency power is provided by 4 MW turbo-charged diesel generators, four (4) for Phase 1 and two (2) for Phase 2.

18.3 Access Roads

18.3.1 Main Access Road

The main access road linking the process plant site to the town of Schefferville, Quebec is designed for delivery of consumables during operations as well as to provide access during construction for delivery of construction material and equipment.

The road is approximately 185 km long, 10.3 m wide (including shoulders), and constructed of granular materials or crushed rocks suitable for traffic travelling at a nominal speed of up to 60 km/h.

18.3.2 Plant Roads and Secondary Road

The plant and secondary roads are 8 m and 4 m wide respectively, and constructed of granular materials suitable for traffic traveling at a nominal speed of up to 30 km/h. In Phase 1, there is about 60 km of plant and secondary roads required for the process plant including tailing, tailing water reclaim, mine, accommodation camps and explosive building access. 20 km more will be added in Phase 2.

18.3.3 Haul Roads

The mine haul-truck roads required for off-highway mine 400-ton (short ton) haul trucks will be designed to be 30 m wide, and to feature a 2 m high rock safety berm on each side. The same road design applies to the access road for mine haul trucks from the crushers to the maintenance area.

Beside the mine pit roads, there is about 12 km of haul roads required for Phase 1 and 10 km will be added in Phase 2.

18.3.4 Drainage

Surface runoff will be diverted through open ditches and trench drains. The ditches will mostly be excavated in the overburden. Interceptor and diversion ditches are provided on the higher ground south of the main substation and the runoff water is directed to existing streams naturally flowing into the tailing management facility (TMF) area.

Diversion ditches will also be provided around the perimeter of the stockpiles (mine rock, overburden, and low grade) areas. Water that is collected in this perimeter ditch network will be discharged into the water management system and the TMF pond.

18.4 Explosives Preparation and Storage

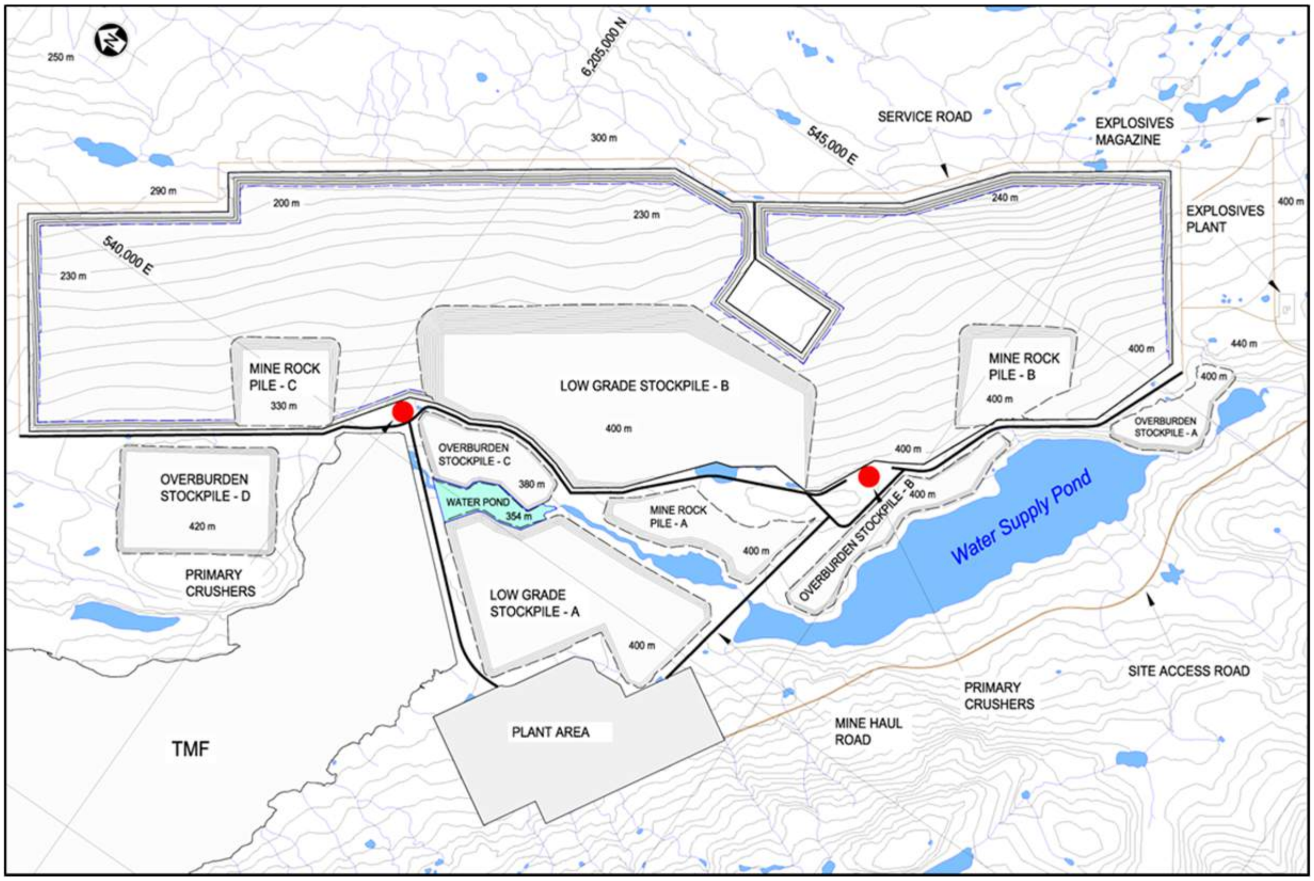
Blasting will be carried out with bulk emulsion which will be manufactured in a facility that will be built and operated on site by a properly licensed explosives supplier. The explosives supplier will be responsible for transporting the raw materials to site, manufacturing the bulk emulsion and loading the blast holes using his own fleet of pumper trucks. The explosives supplier will also be responsible for supplying the blasting accessories such as detonators, boosters and priming cord, although it will be the mine's responsibility to transport these accessories from the magazines to the drill patterns and to tie-in the blast holes. The blasts will be triggered using electronic detonators.

The explosives plant and the magazines to store the accessories will be located at the south end of the open pit and are shown on Figure 18.4. The location of these sites accounts for the minimum distance requirements that are specified by the Canadian Explosives Regulations. Approvals and permits will be required from government regulating bodies prior to the construction of these facilities.

The explosives plant will be composed of predesigned / prebuilt modules that are easily transported and assembled. The facilities include storage silos for raw materials, the offices and garages, as well as the emulsion plant and pumper truck loading area.

In order to support the explosives supplier, the mine operator is required to build and maintain the access road to the two (2) sites and to supply electric power, communications and diesel fuel for the manufacturing of the emulsion as well as the operation of mobile equipment. The mine operator is also required to mobilize and house the contractor's workforce.

Figure 18.4 – Mine Layout



18.5 Communication and Automation

The telecommunication systems will need to be available 24 hours per day, 365 days per year. All active core components to be installed in each telecommunication systems will be designed in order to achieve 99.999% availability.

The telecommunication and physical security equipment will be supplied with the latest software releases. All systems will be reliable, flexible, 100% expandable and developed from proven technology.

The Lac Otelnuk telecommunications and physical security network will be divided into six (6) major areas:

- Primary crusher area;
- Process plant area;
- Airstrip area;
- Tailings water return pumping station;
- Product delivery system (PDS);
- Port.

The major components of the telecommunication and physical security network will be installed in the process plant administration complex data centre.

The Lac Otelnuk telecommunication and physical security network architecture will be Ethernet TCP/IP based. The IP network architecture will be developed on a star topology.

18.5.1 Control Systems

The PCS and the PDCS communication network will not be physically fully redundant. The programmable logical controller (PLC) communication card and the telecommunication active equipment dedicated to these networks will be redundant. The telecommunication passive equipment and the telecommunication cabling will not be redundant.

The network architecture must be able to be divided physically and virtually in order to provide separate local area network (LAN) services for each of the following systems:

- Data and audio communication network
- Video network
- Physical security systems
- Office plant network
- Process control system (PCS) network
- Power distribution control system (PDCS)
- Electrical equipment.

18.6 Support Infrastructures and Utilities

18.6.1 Fuel Systems

a) Diesel Fuel Storage and Distribution

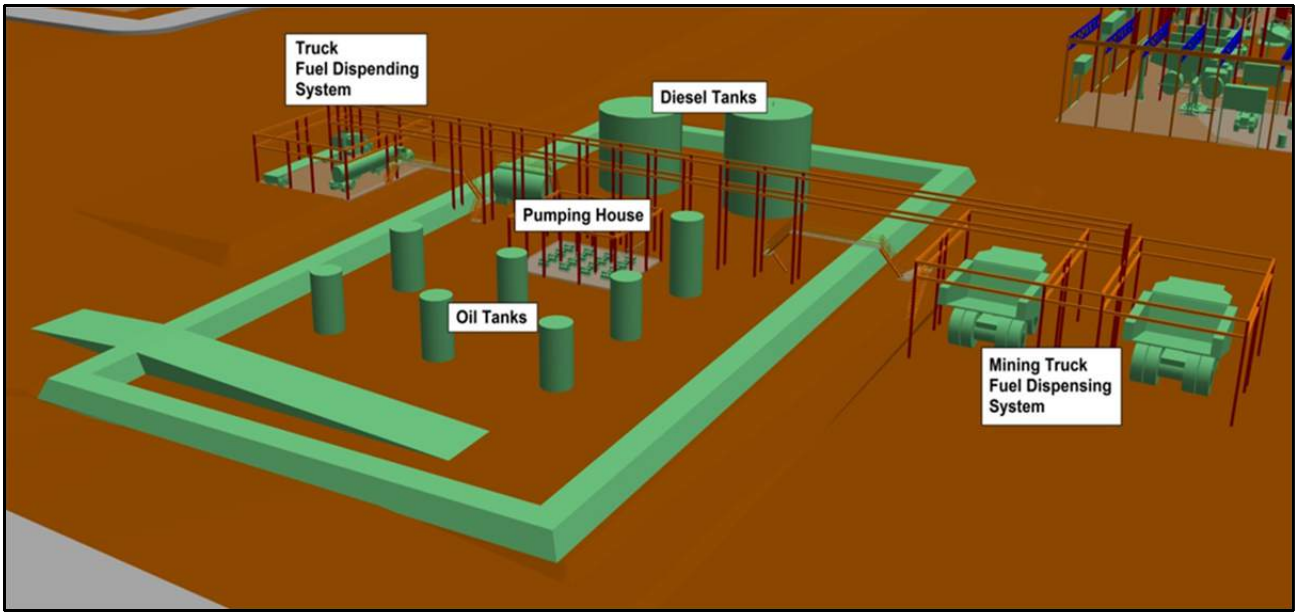
Diesel will be delivered to Schefferville by train and then transferred to tanker trucks and transported to the plant site where it will be stored in large diesel storage tanks of about 1,300,000 litres each. There is one tank of same size installed for each phase of the project.

In order to control any fuel spillage to the environment, a containment berm is provided around the tank farm. The waters collected by drainage-sump system are connected to an oil/water separator from which water is pumped to the tailings thickener while oil is transported by vacuum trucks to the incinerator.

Diesel is pumped via a piping system from the diesel storage tank farm to the vehicle diesel dispenser, mine truck diesel dispenser, emergency generators day tank and to mine diesel day tanks for mine truck refuelling system close to the mine primary crushers.

Figure 18.5 shows the 3D layout of the diesel storage and distribution facility.

Figure 18.5 – Diesel Storage and Distribution



b) Gasoline Storage and Distribution

Gasoline is also delivered to the plant site by tanker trucks and is stored in a gasoline storage tank (common for both phases). The gasoline storage tank of about 80,000 liters capacity is a double wall tank as per ULC S601 and API 650 standards. Gasoline is pumped from the storage tank to the vehicle gasoline dispensers.

c) Jet Fuel Storage and Distribution

Jet fuel is delivered to the plant site by tanker trucks and is stored in a jet fuel storage tank of about 65,400 litres capacity (common for both phases).

The jet fuel storage tank is a double wall tank as per ULC S601 and API 650 standards. The jet fuel is pumped through a loading arm into jet fuel trucks, which are used to refuel aircrafts at the airport.

18.6.2 Steam Generation and Distribution

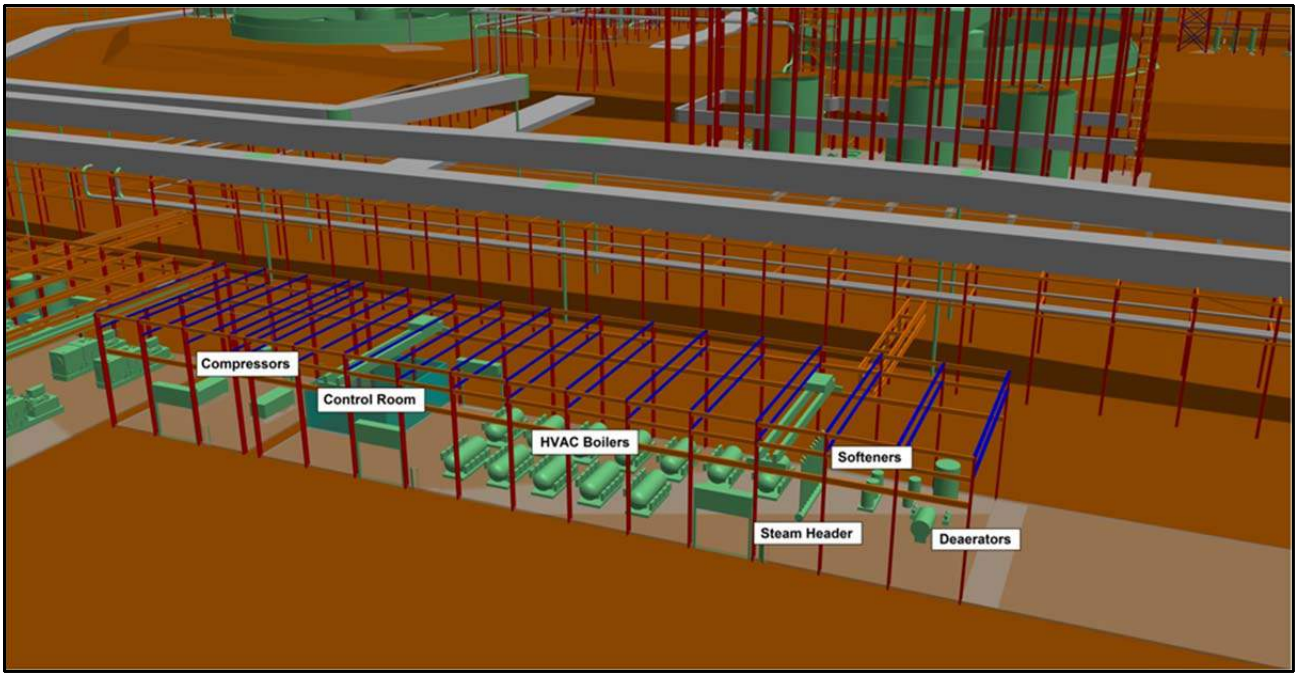
Electrical steam boilers are used for heating the process plant, workshops and the warehouse. Eleven boiler trains for Phase 1 and five for Phase 2 are necessary to fulfill the heating requirements of the buildings. Each boiler train includes one deaerator, one feed water pump, one boiler, one flash drum (for blowdown), and one condensate drum. One demineralised water package and one boiler feed water chemical injection package are common for all the boiler trains. Filtered water from utility distribution pumps is used as make-up water for the boilers.

The generated steam is sent to the process plant and to the workshops and warehouse. The condensate is returned to a condensate drum and is pumped to the deaerator to close the loop.

Another boiler is used to produce steam for the utility stations in the process plant and also to provide steam to thaw the ore stockpile near the apron feeders to facilitate the free flow of ore when the material is frozen. For this specific purpose, one boiler train per phase is needed. The boiler train is similar to the one for heating the building, except there is no condensate return. One utility boiler is installed in Phase 1 and a second one will be added in Phase 2.

Figure 18.6 shows the 3D layout of the boiler house for Phase 1 only (12 boilers).

Figure 18.6 – Phase 1 Boiler House



18.6.3 Plant Air Compressor

A central compressed air station provides all the necessary compressed air for the plant air distribution system adjacent to the boiler building. The air compressing station is located in the same area as the boilers. It includes two (2) compressors per phase (one operating and one on standby), each featuring a capacity of approximately 1,556 Nm^3/h. The compressors are supplied with air dryer and receiver.

18.6.4 Maintenance Workshop and Warehouse Complex

a) Maintenance Workshop

The maintenance workshop is used for the maintenance of small vehicles, rubber lining, and process equipment and can be accessed by utilidors. It includes workshop equipment and machine tools to perform regular mechanical, piping, electrical and instrumentation maintenance on equipment.

The maintenance workshop building is approximately 2,700 m^2 and is provided with overhead cranes and jib cranes. It includes vehicle access doors and fresh air heating/ventilation and gas/fumes exhaust systems.

b) Mine Equipment Maintenance

The mine equipment maintenance complex includes all necessary facilities and equipment required for mine equipment maintenance. It includes several mine truck wash bay stations, repair stations, welding shop, tire press shop, as well as a storage and a handling systems for all new and used oil and lubricants.

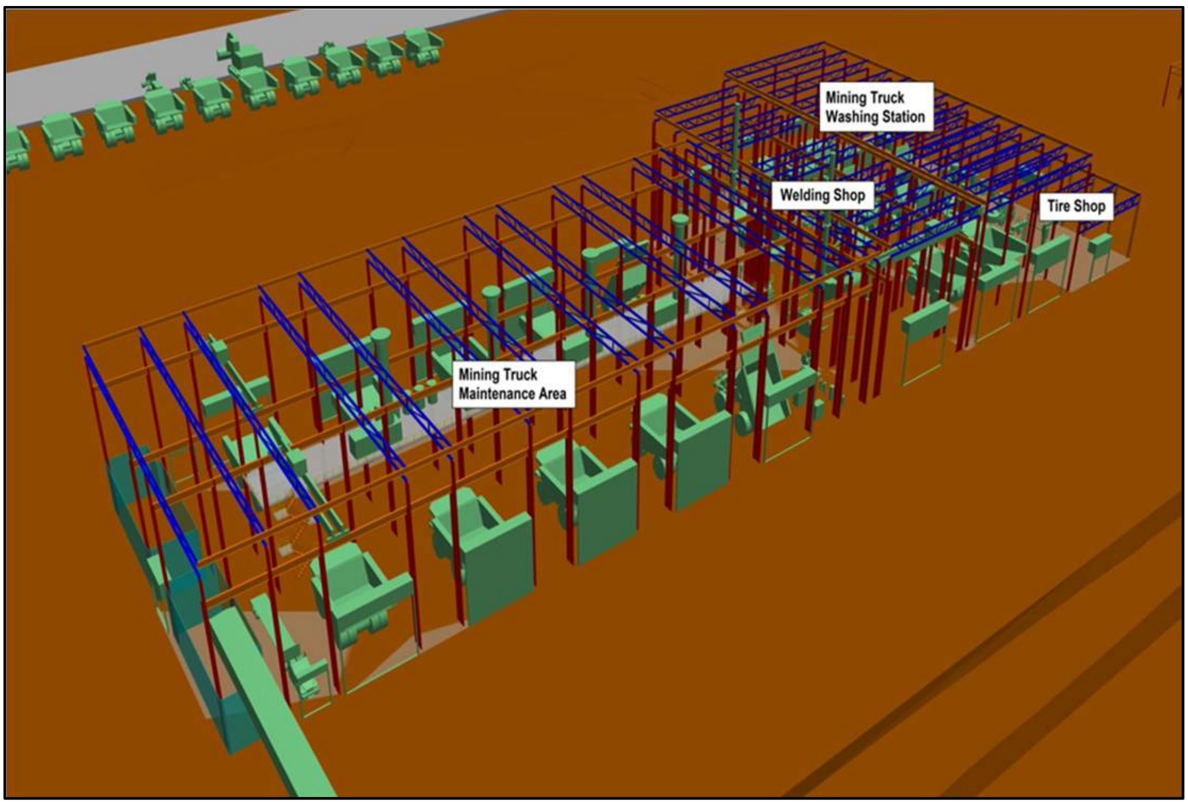
The building is provided with overhead cranes and jib cranes covering the floor work space. It includes large and small vehicle access doors and fresh air heating/ventilation and gas/fumes exhaust systems.

The mine truck washbay area is provided next to the welding shop/maintenance area.

Space for haul truck parking is reserved north of the maintenance building.

Figure 18.7 provides a 3D representation of the mine equipment maintenance complex.

Figure 18.7 – Mine Equipment Maintenance Complex



c) Warehouse

The main warehouse used for storage for consumables, spare parts, tools is located next to the maintenance workshop building. The building area is approximately 4,160 m^2.

18.6.5 Administration Complex and Laboratory

The administration complex located adjacent to the process plant substation and next to the SAG mills area will house all management offices, meeting and control room on a two story building. Each building floor consists of approximately 1,440 m^2.

The laboratory required for the process plant and environmental analysis is included within the same complex. It covers a floor area of about of 800 m^2.

18.6.6 Domestic Waste Water Treatment

The domestic waste water treatment plant will be built in modular units. Two units are needed during the construction phase, and one unit will be required during the production phase. The plant is located to the north of the permanent camp. Waste water from the process plant, the camp, and the maintenance areas is collected and transferred using five pumps to the waste water treatment plant. Wastewater from the primary crushers' area and the airport is collected and transported to the plant via a vacuum truck.

Waste water is directed to an equalization tank to first homogenize the effluent and regulate the flow to the treatment plant. Submersible pumps transfer the waste water through a screen to eliminate coarse debris. The effluent then flows by gravity to the primary treatment system which eliminates fats, oils, and grease and allows for settling of settable particles.

18.6.7 Solid waste disposal

Non-hazardous solid waste from the mine, the process plant, and the accommodation complexes and the residue from the wastewater treatment plant are collected by truck and sent to waste disposal cells. Hazardous solid waste is sent by truck to a dedicated hazardous solid waste disposal cell.

Once the solid components are removed from the effluent, it then flows to a biological treatment system as required. The treated water is discharged into a ditch which directs it towards the tailings management pond area.

The final sludge produced in the sewage water treatment plant is directed towards a storage tank and dewatered to a concentration above 20% solid. The resulting solid waste sludge will be disposed in the landfill.

18.6.8 Water Systems

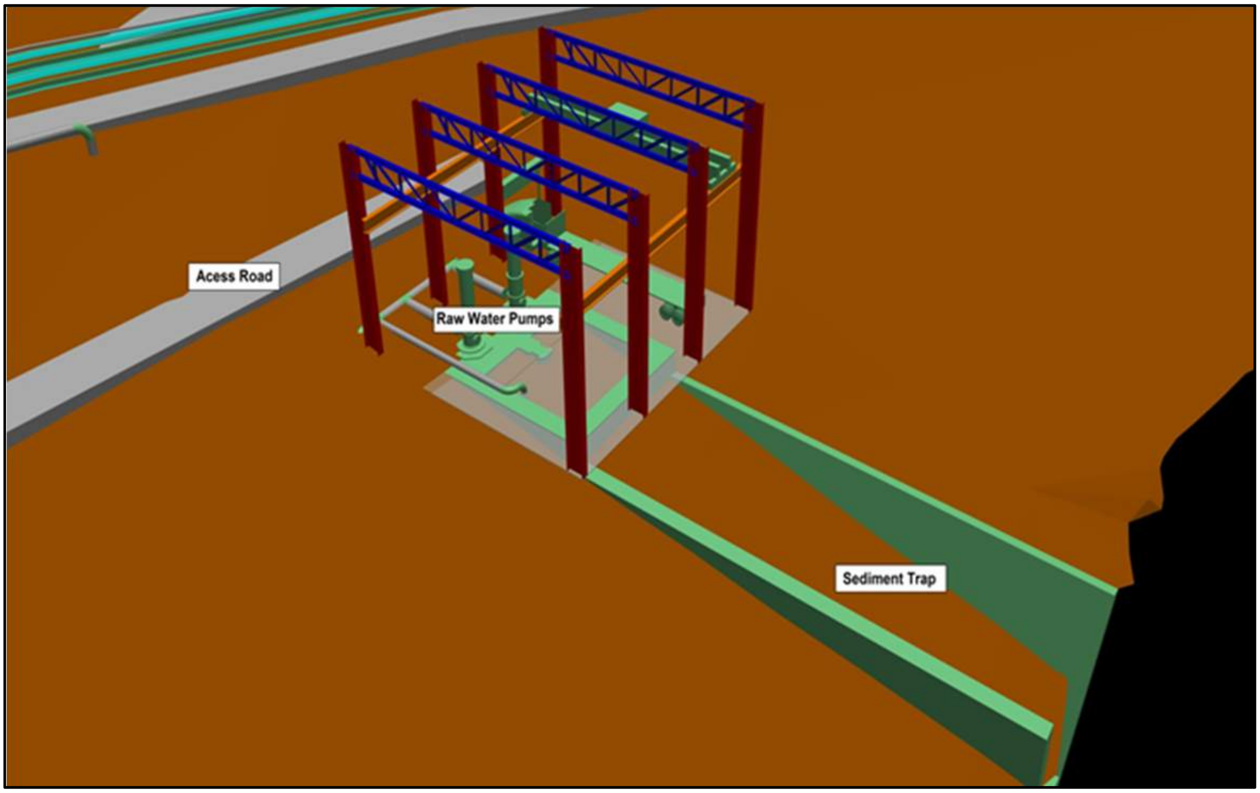
Raw Water Intake and Pump House

Raw water from the water supply pond (WSP) is pumped to the process plant with two pumps (one operating and one on standby) at a nominal flow rate of 3,631 m^3/h and a design capacity of 4,172 m^3/h.

- A portion of this water feeds the raw water filtration system, from which water is directed to the filtered water storage tank and to the firewater tank.
- The remaining water is not filtered; rather it is used as a make-up for the five process water ponds.

The detail arrangement of the raw water pump house is presented in 3D in Figure 18.8.

Figure 18.8 – Raw Water Pump House



a) Raw Water Filtration and Filtered Water Distribution

From the raw water pumped from the WSP through the filtration system, a fraction of the water is directed to the fire water tank, and the remaining filtered water is collected in a water storage tank and then pumped to the following utilities:

- Potable water treatment plant;
- Gland seal water distribution network;
- Utility stations (for wash down stations);
- HVAC and utility steam boilers;
- Product delivery system for seal water;
- Tailings flocculent preparation package;
- Dust suppression systems in the primary crushing areas and in transfer towers.

The raw water filtration plant consists of multiples filtration units. The filtration plant will be built in two (2) phases: six (6) filters will be required for Phase 1 and four (4) filters will be added for Phase 2.

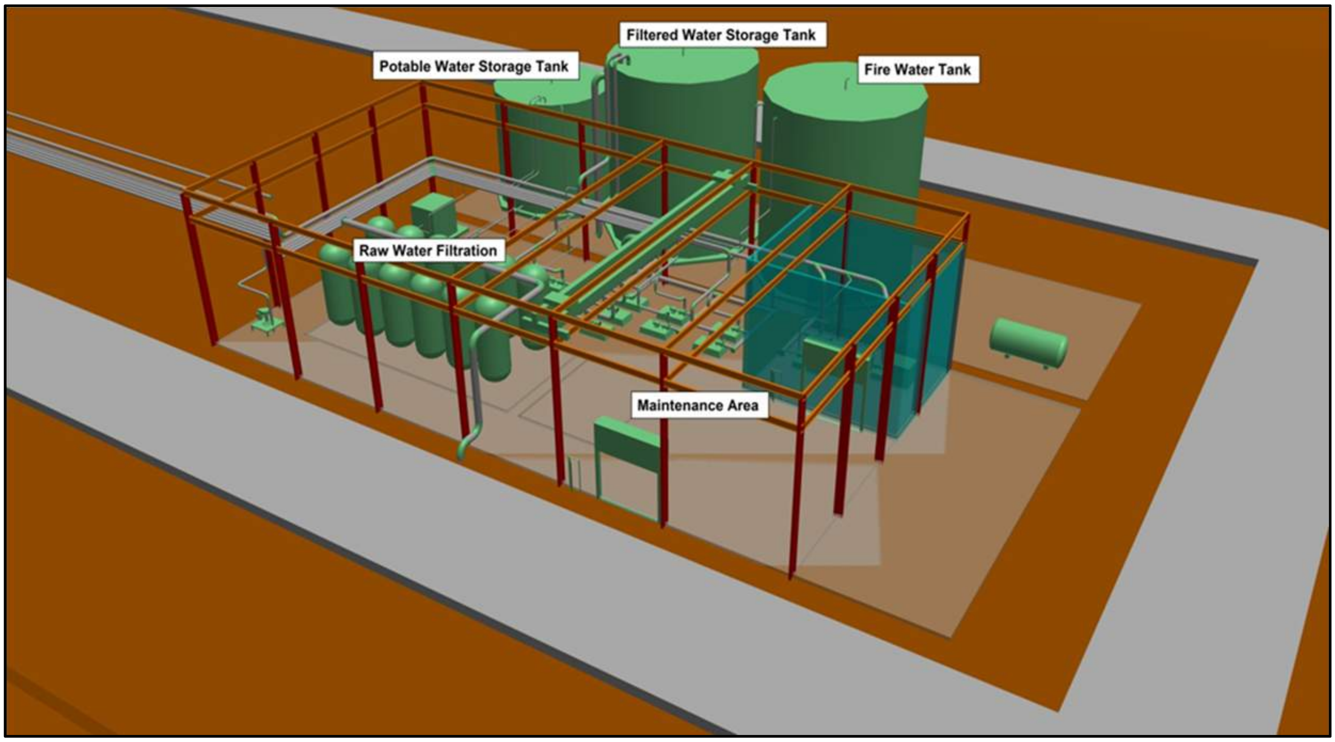
The filtration plant consists of cross-flow microsand filters. The water first enters the system going towards the filters and is directed parallel to the media, producing a cross flow condition and allowing the particles to be captured.

During the automatic backwash cycle, one filter is washed with the water produced by the other filters. Filtered water is still produced during the backwash sequence but the overall flow rate is slightly smaller. The backwash cycle is followed by a rinsing step.

The filter backwash water will be sent to a water building sump pump, which will direct the water towards the TMF.

Figure 18.9 provides general arrangement details.

Figure 18.9 – Raw Water Filtration and Distribution



b) Potable Water

The potable water treatment plant will be built using three modular units. Two units are needed during the construction phase, and one unit will be used during the production phase. Each unit consists of two pre-assembled skids.

The source of potable water at the process plant is from the water supply pond (WSP). Water treated by the potable water system is first pre-filtered by the raw water filtration package.

c) Firewater

A portion of the water from the raw water filtration plant is sent to the firewater tank. One firewater tank is common for both phases.

Three firewater pumps are connected to the firewater network: one jockey pump and two pumps (one electrical and one diesel) which supply water to hydrants, sprinkler systems, and other fire protection systems. The firewater requirements per building are provided in the building list in accordance with design criteria.

18.6.9 Incinerator

The following waste materials are stored in a tank and incinerated periodically:

- Oil from the oil water separators;
- Used oil generated by the operations and maintenance;
- Grease;
- Soiled rags in a dedicated department.

The incinerator is designed to handle the quantity for the full production.

The ash residue from the incinerator is trucked to the hazardous solid waste cell for disposal.

18.6.10 Landfill

Acting as a waste management facility on site, a landfill will receive trash, sludge, and garbage which consist mostly of food scraps. The landfill area will be expanded throughout the life of the mine and will also be capped progressively.

The landfill consists of non hazardous standard cells and hazardous cells.

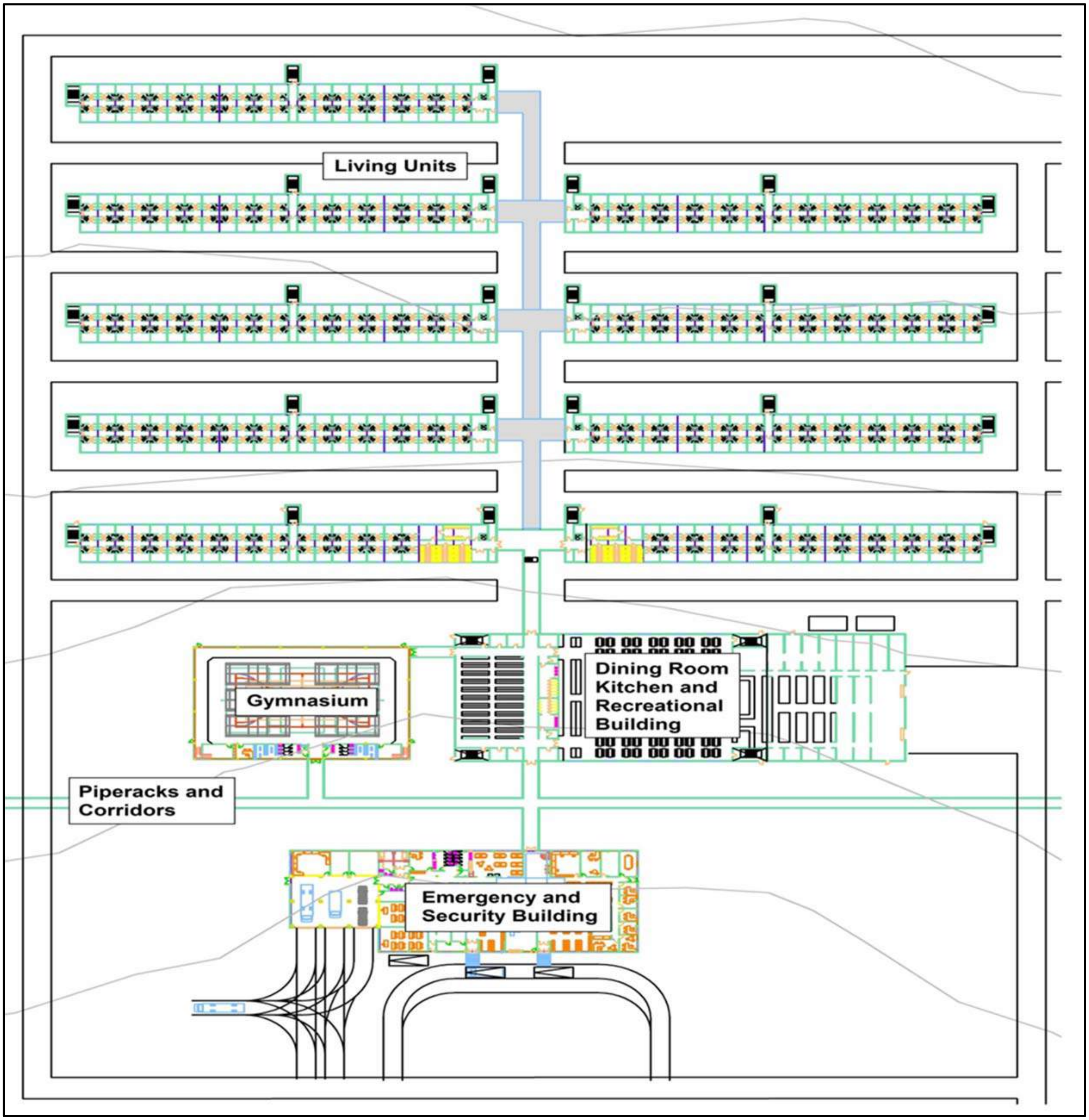
Another fenced landfill area will also be designed to receive construction debris and other solid materials which are not contaminated.

18.7 Camp Site Accommodations

The permanent accommodations are located approximately 0.5 km from the process plant. They consist of: living units (dormitories), kitchen / dining room / recreational facilities building, gym and fitness center, and security / first aid / emergency services building.

For further details, refer to Figure 18.10 here below.

Figure 18.10 – Accommodations



18.7.1 Emergency and Security Building

The emergency and security building is a single level building consists of approximately 1,550 m^2 of offices, and about 380 m^2 for the emergency response centre. The height of the emergency response centre considers the clearances required by the fire-fighting equipment. A 2-hour-fire-rated wall separates the emergency response vehicle area from the office structure.

The security building includes:

- Reception area;
- Arrival and departure areas with luggage storage and distribution facilities;
- Medical facilities fitted with a nurses'/doctor's office, examination rooms, and washroom;
- A helipad for emergency evacuation.

First aid stations will be provided through-out the process plant complex.

18.7.2 Kitchen / Dining Room / Recreational Facilities

The building has two (2) floors, with the kitchen and dining area on the ground floor, and recreational facilities on the first floor. These facilities are included in Phase 1.

The building has a floor area (ground) of approximately 3,450 m^2. The kitchen has a capacity of serving approximately 500 meals/hour. The kitchen is suitable for a 2-choice hot meal with two (2) service lines.

The dining area can accommodate a minimum of 750 persons in one seating, with additional space planned for vending machines, lunch room area, coat room, and washrooms for men and women.

18.7.3 Living Modules

The living modules (dormitories) are arranged in nine (9) separate wings of three (3) floors each. Altogether, the complex comprises 1,134 one-person rooms, each with a surface area of approximately 17.4 m^2.

The dormitories will be constructed in two phases:

- Phase 1 (wings 1 to 6, total room capacity: 756)
- Phase 2 (wings 7 to 9, total room capacity: 378).

18.7.4 Corridors / Egress Stairs

Within the accommodation complex, the minimum width of the corridors giving access to the rooms is approximately 1.5 m. Egress stairs have a minimum width of 1.1 m.

Fire rating and acoustical separations between corridors / bedrooms / technical rooms are designed per building codes. The sound transmission class of all bedroom separations is rated at 55 STC minimum.

18.7.5 Utilidors (Subarctic Corridors)

Utilidors are used to protect people from the harsh weather. They consist of one-storey corridors interconnecting all buildings of the accommodation complex, as well as the administration, process, and maintenance areas. The utilidors also carry interconnecting utilities such as piping, electrical, and communication cables between buildings.

Minimum utilidor width is 2.45 m. Ceiling height minimum is 2.7 m.

The utilidors are in compliance with national code requirements regarding fire protection, fire rating, exposure and egress capacity / distribution.

18.7.6 Construction Camp

The construction camp will be located next to the permanent accommodations camps and consist of similar facilities. The construction camp is however designed for temporary purpose.

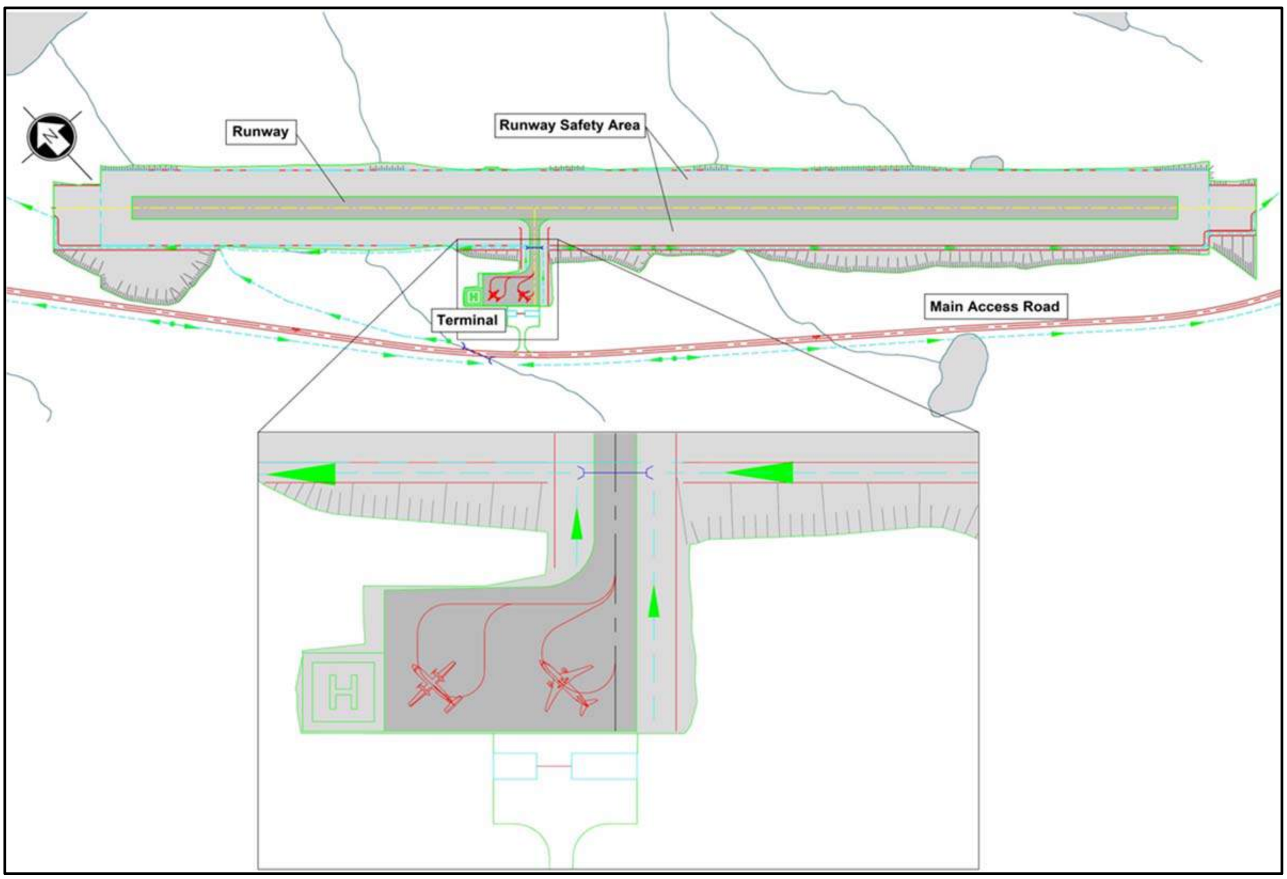
The size of the living modules (dormitories) is of a similar design as the permanent one. It is arranged in fifteen (15) separate wings of three (3) floors each. Altogether, the complex is sized to accommodate the peak manpower of 1,900 persons for Phase 1.

18.8 Airstrips / Aerodrome

The design and construction of the airport facility will comply with standards contained in the newly released 5th edition of Transport Canada TP 312 that has recently been released in draft form and is expected to take effect from early 2015.

Further details on the airstrip / aerodrome are shown in Figure 18.11 below.

Figure 18.11 – Airstrip / Aerodrome



The aerodrome site is located at latitude of 55° 53’ 38" N and longitude 68° 12’ 53" W, and approximately 12 km south of the proposed process plant. The main access road connecting the mine site and Schefferville will have a secondary road connecting to the aerodrome.

The aerodrome area includes a pre-fabricated terminal building, maintenance garage, helicopter pad, and an apron area. The buildings will be heated by an electrical heating system. The apron sizing is designed for a Boeing B737-200C. The airstrip will be built of a granular material.

18.9 Tailings Management Facility

The general arrangement of the TMF and surrounding area is shown on Figure 18.12. The final tailings footprint covers an area of approximately 45 to 50 km^2 and is shown in Figure 18.13.

18.9.1 Concept for Tailings Management

The volume requirements for tailings storage are summarized in Table 18.2 based on the characteristics of settled tailings. The table gives the volume requirements for the design case for 60 % solids in the tailings slurry as well as for a sensitivity case for 50 % solids. The average (struck level) tailings surface at about elevation 353 m provides the required tailings storage volume of about 2,300 Mm^3. This results in a maximum tailings thickness at the end of operations of about 150 m at the West Dam.

Table 18.2 – Tailings Storage Volume Requirements

Year	Concentrate Production (Mt)	Incremental Tailings Production (Mt)	Cumulative Tailings Production (Mt)	Volume 60 % Solids in Tailings Slurry (Mm^3)		Volume 50 % Solids in Tailings Slurry (Mm^3)	
				Inc	Cum	Inc	Cum
1	10	27.7	27.7	17.0	17.0	17.9	17.9
2	25	69.4	97.1	42.6	59.6	44.9	62.8
3	30	83.2	180.3	51.1	110.8	53.8	116.7
4	30	83.2	263.5	51.1	161.9	53.8	170.5
5	30	83.2	346.8	51.1	213.0	53.8	224.4
6	30	83.2	430.0	51.1	264.1	53.8	278.2
7	30	83.2	513.2	51.1	315.2	53.8	332.1
8	40	111.0	624.2	68.2	383.4	71.8	403.9
9	50	138.7	762.9	85.2	468.6	89.7	493.6
10	50	138.7	901.6	85.2	553.8	89.7	583.4
11-15	250	693.5	1595.1	426.0	979.8	448.7	1032.1
16-20	250	693.5	2288.6	426.0	1405.8	448.7	1480.8
21-25	250	693.5	2982.1	426.0	1831.8	448.7	1929.6
26-30	250	693.5	3675.6	426.0	2257.8	448.7	2378.3

Note: Inc = Incremental Cum = Cumulative

Figure 18.12 – General Arrangement of TMF and Surrounding Area

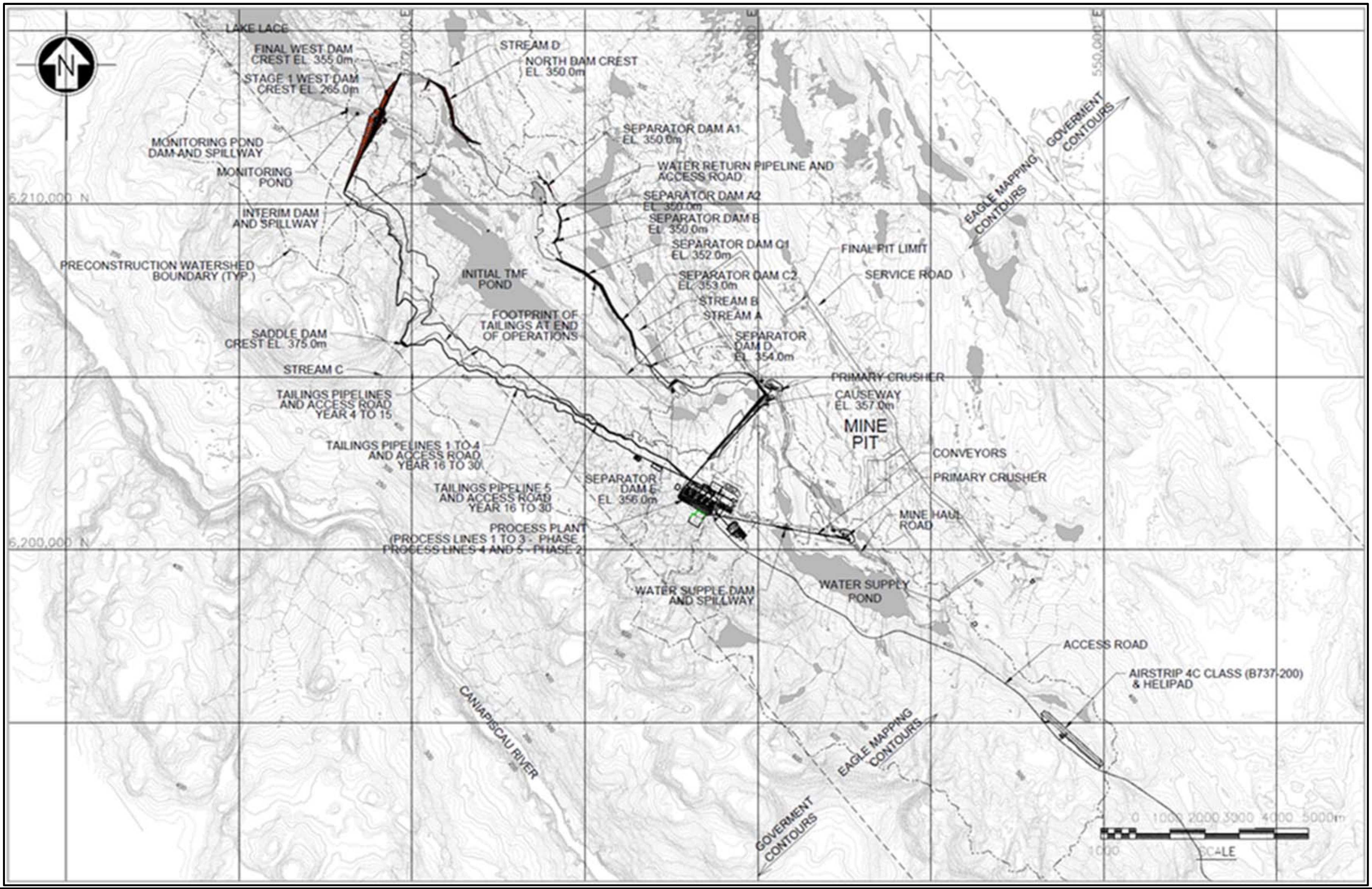
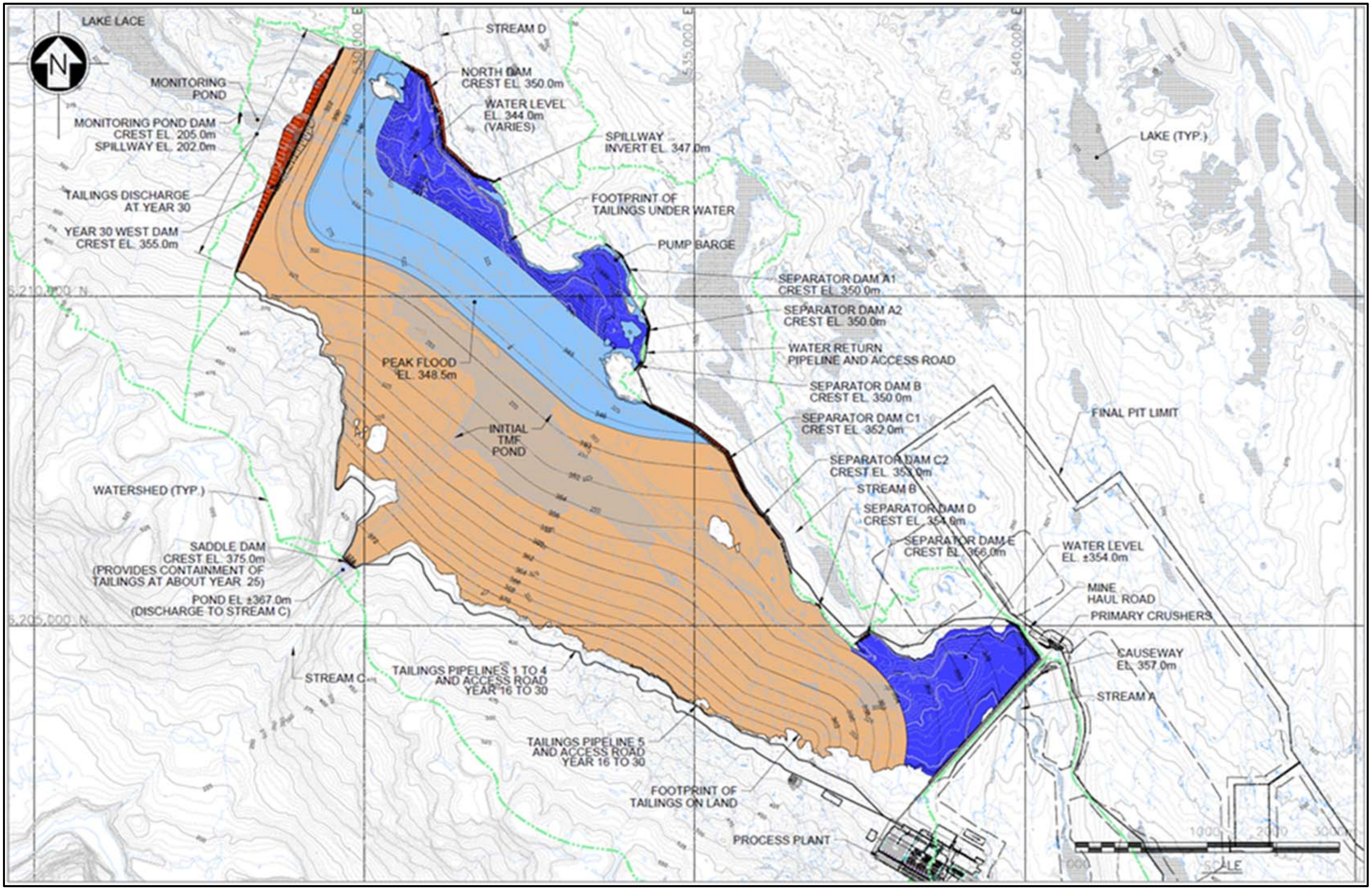


Figure 18.13 – Plan of TMF at Year 30



18.10 Water Management

18.10.1 Site Drainage Plan and Water Management Strategy

a) Site Drainage Plan

Figure 18.14 shows the regional watersheds and drainage paths in the area of the Project. The drainage plan for the mine site is developed to maintain discharge to the Caniapiscau River watershed.

Figure 18.14 – Regional Watersheds and Discharge Paths in Project Area

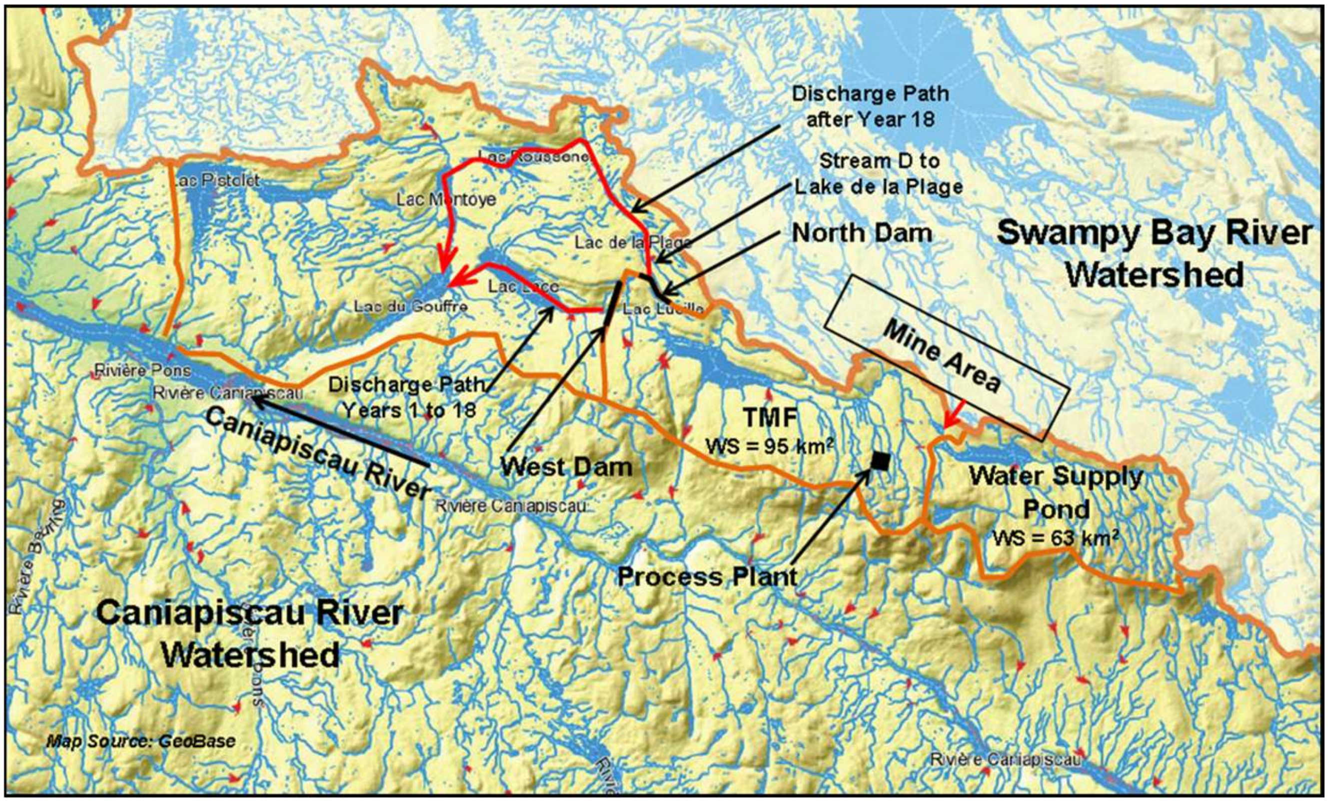


Table 18.3 provides a general description of the main water ponds at the mine site.

Table 18.3 – General Description of the Main Water Ponds at the Mine Site

Main Water Ponds at the Mine Site	General Description
Water Supply Pond	• Source for site raw water and process make-up water • No discharge for average runoff year based on the design • During wet years, excess water discharges to the TMF.
Stream A Drainage Course Upstream of TMF	• Receives open pit dewatering flow which then reports to the TMF.

Main Water Ponds at the Mine Site	General Description
Water Pond(s) within the TMF	• Serve as water management center for runoff water from the mine site • Receive tailings slurry, runoff water from process plant, mine pit dewatering flow, local runoff and any excess water discharged from the water supply pond • Water in the TMF Pond is reclaimed for process use • Excess water from the TMF discharges from the West Dam into the monitoring pond for years 1 to 18 • Excess water from the TMF discharges from the North Dam into Stream D for years 19 to 30.
Monitoring Pond	• Final effluent discharge point of the mine site for flow quantity and quality monitoring for years 1 to 18.
TMF Pond at North Dam	• Final effluent discharge point of the mine site for flow quantity and quality monitoring for years 19 to 30 and for mine closure.

18.10.2 Overall Site Water Balance

Based on the review of available information on hydrology, water balances of the water supply pond and TMF, and based on inputs from process, mining and tailings disposal, complete site water balances have been developed for years 15 and are presented in Figure 18.15. The year 15 have been selected to represent the full Phase 2 for average annual water balances when concentrate production rate is at a nominal capacity of 50 Mt/y.

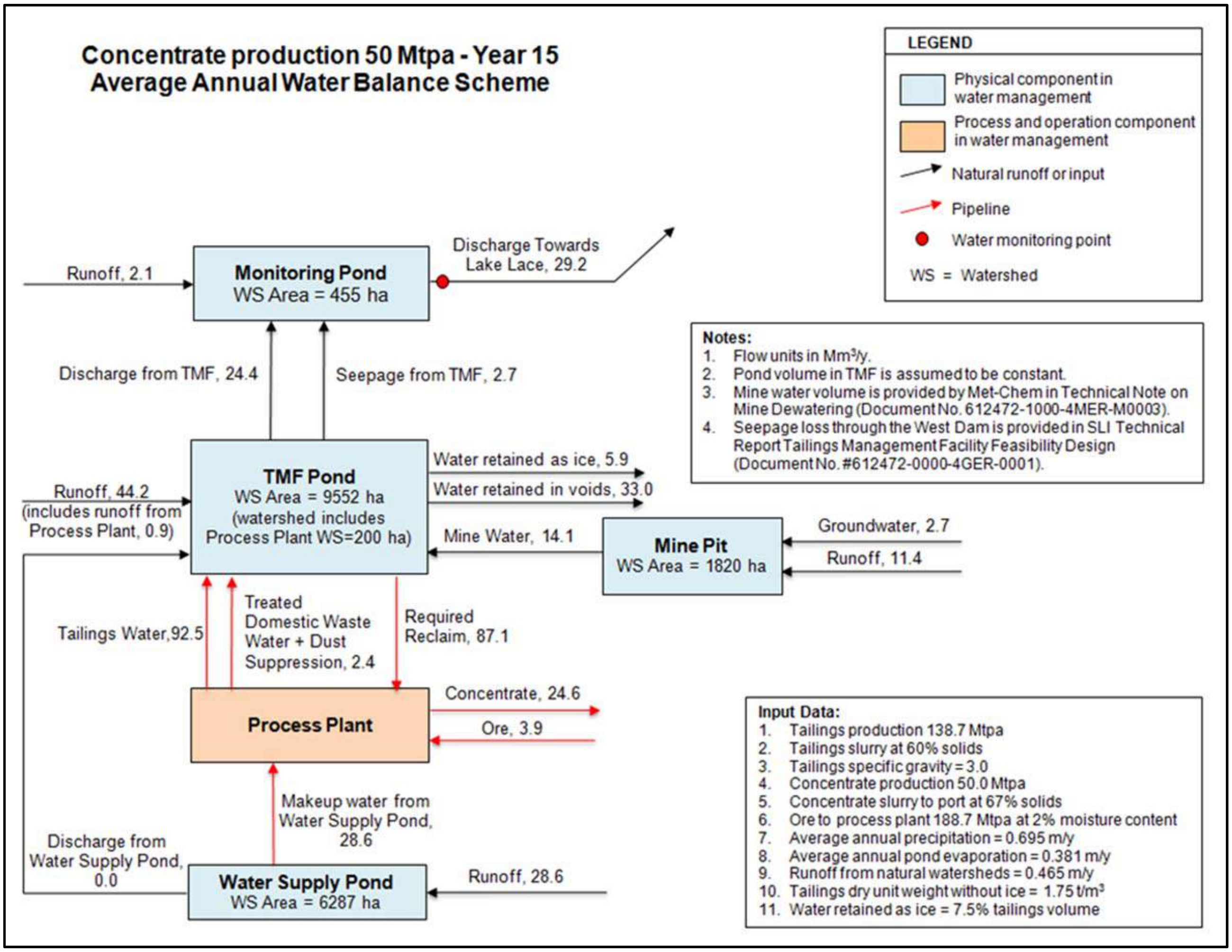
The inflow for the overall site water balance includes the following components:

- Water from all watersheds contributing to the water supply pond and TMF (includes direct precipitation on water body, surface runoff and process water)
- Mine pit water reports to the TMF.

The outflow for the overall site water balance includes the following components:

- Water losses from evaporation and seepage through the dams and their foundations
- Water in the concentrate slurry that is pumped off site (the slurry has an expected solids content of 67 %) and water used for PDS batching operations.

Figure 18.15 – Overall-Site Average Year Annual Water-Balance Scheme – Year 15



(Note: Year 15 is selected to represent Phase 2 operation condition when concentrate production rate is at 50 Mt/y)

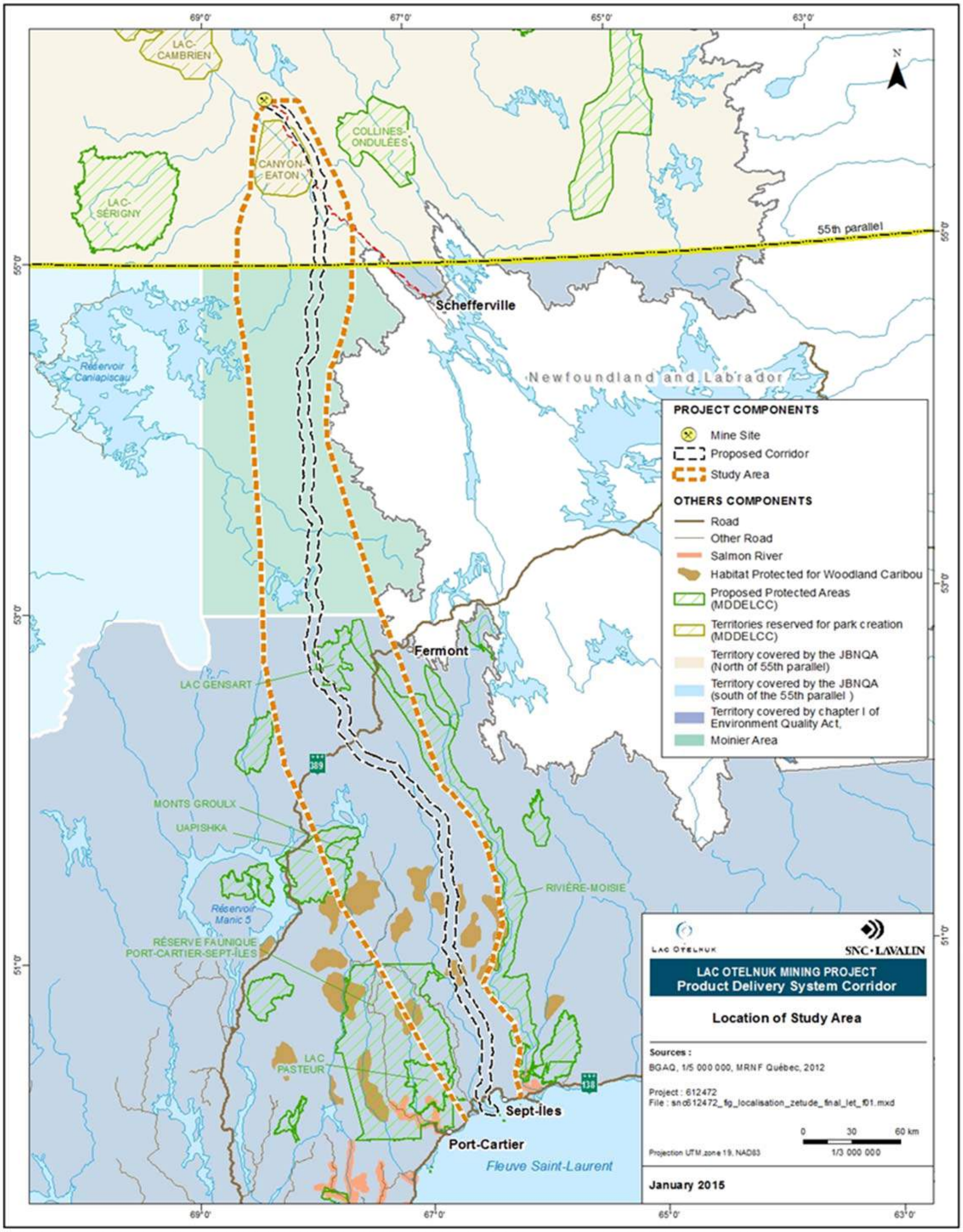
18.11 Product Delivery System

18.11.1 Product Delivery System Description

The Product Delivery System ("PDS") refers to a pair of slurry pipelines (one with a 30 Mt/y capacity for Phase 1 and the other having a 20 Mt/y capacity for Phase2) required to bring iron ore concentrate in the form of a slurry from the Lac Otelnuk Iron Ore Project mine site to the Pointe Noire Terminal at the Port of Sept-Îles. The pipeline route is approximately 755 km and the pipelines are mostly shallow buried except when crossing water bodies, where they are on piles or on top of culverts for small crossings.

The PDS routing remains within the province of Quebec as shown in Figure 18.16. The PDS starts uphill through the Caniapiscau watershed, passes over the Central Plateau, and crosses the Groulx Mountains to reach the coastal plain.

Figure 18.16 – Product Delivery System Alignment



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Source: BGAQ

18.11.2 Slurry Pipeline Design

LOM engaged the services of Ausenco PSI for the study and design of the PDS system based on the alignment established by SNC-Lavalin as per Figure 18.16.

The predominant design principle is to bury the PDS at the proper depth so as to optimize the insulation costs while minimizing rock excavation. The design has adopted a "No-Freezing, No-Plugging philosophy", which means that the slurry transportation system will have the necessary reliability and emergency back-up equipment to maintain no-freeze conditions during an emergency stoppage and the slurry transportation system slope will be restricted to avoid plugs even if the plant is shut down while the line is full of slurry.

Table 18.4 summarizes the results of the concentrate pipeline systems design.

Table 18.4 – Concentrate Pipeline Systems Summary

Item	Value in Phase 1	Value in Phase 2
Design throughput	30 Mt/y	20 Mt/y
Design flow rate	2,635 m3/h	1 757 m3/h
Pump discharge pressure (design pressure)	21,000 kPa	20 000 kPa
Concentration (by weight)	67% nominal	67% nominal
Pipe length (design)	755 km	755 km
Outside diameter	KP 0–689: 30" KP 689–735: 26"	KP 0–689: 26" KP 689–735: 22"
Average wall thickness	0.70" / 0.68"	0.62" / 0.59"
Pipe material	Steel, API 5L X70	Steel, API 5L X70
Pipe steel tonnage	254,930 t	194,760 t
Insulation thickness	1.5"	1.5 "
Number of pumping stations	3 with 10 + 2 positive displacement (PD) pumps	3 with 6 + 1 PD pumps
Agitated tanks at concentrator plant (repeat at terminal)	5; 20 m diameter, 18 m high	3; 20 m diameter, 18 m high
Burial depth	600 mm top of pipe	600 mm top of pipe
Number of pressure monitoring stations	20	20
Transit time from mine to port	5.8 days	5.25 days

18.11.3 Pipe Bench Design

The pipe bench addresses the northern portion of the PDS which is approximately 400 km long between route 389 and the intersection of the permanent access road 70 km south from the mine site.

There are two access requirements for the PDS, one is to allow transportation of equipment, manpower and materials during construction and the other requirement is for servicing of the intermediate pump stations and re-fueling the emergency power backup generators at the intermediate pump stations.

The civil design of the southern portion from Route 389 to Sept-Îles of the PDS focused on the trench design only. Construction access was assumed to be created by the pipeline contractor for the exclusive use during construction.

From kilometer post KP 70 to KP 465 where the PDS intersects respectively the mine permanent access road from Schefferville and route 389 coming from Baie-Comeau, a temporary access trail (the pipe bench) will be first constructed along the path of the PDS to allow a complete access to the site for construction and delivery of mining or any other equipment too large to be brought by rail. The pipe bench design is based on the road limitations specified in the modularisation specification and logistic report produced during the feasibility study which provides the following limitations for transporters:

- Weight = 10,000 kg/axel load
- Width = 0 - 2.60 m
- Length = 0 – 23 m

The selected method is to bury the PDS in a shallow trench so as to minimize rock excavation while providing support and freeze protection so as to utilize a relatively thin (1.5 inches) insulation jacket.

Figure 18.17 and Figure 18.18 below show the typical cross sections of the entire right of way when the PDS is constructed alongside the pipe bench.

Figure 18.17 – Typical Cross-Section of PDS Installation along Pipe Bench, Phase 1

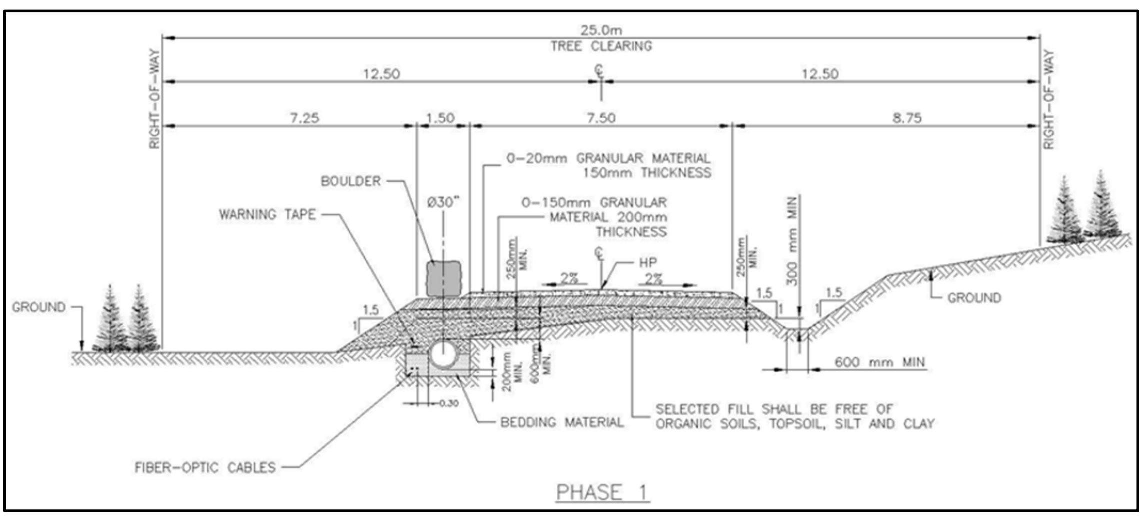
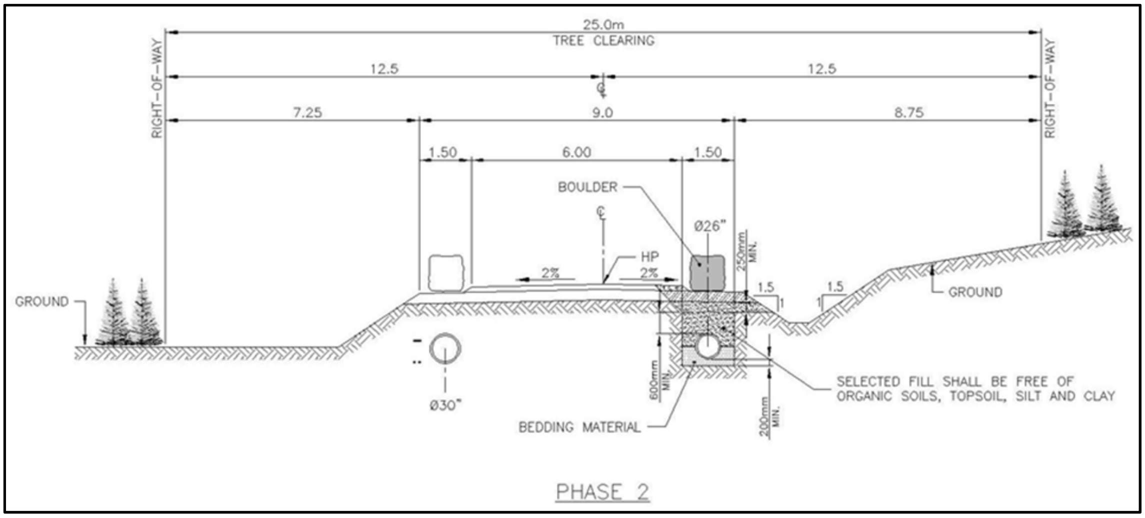


Figure 18.18 – Typical Cross-Section of PDS Installation along Pipe Bench, Phase 2



Over the course of the PDS lay many streams and lakes that will be crossed by supporting the pipeline on piles. In the access road section, the road will deviate from the pipeline path when there is a more favourable roadway that permits avoiding large spans of bridges over water. Small streams (<5m wide) that often become intermittent in summer and winter will be trenched or crossed on top of the culverts.

18.12 Port Facility

18.12.1 Port Facility Concept

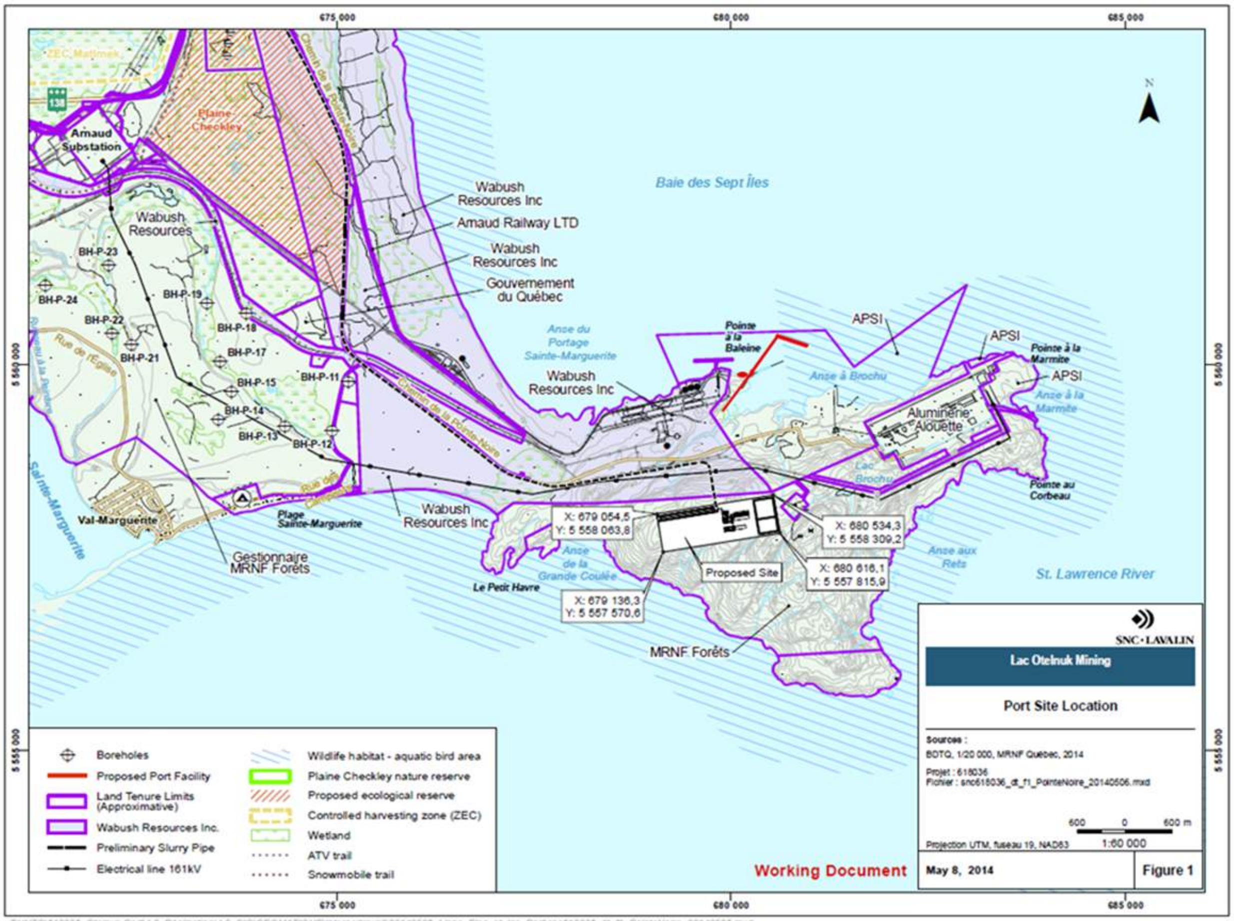
The concept for the port area which includes product dewatering, storage, reclaiming, and shipping of the concentrate is summarized as follows:

- The concentrate slurry, at a solids content in the design range of 65 % to 67 %, will be delivered by the PDS to terminal station (described in Concentrate transportation, PDS Section 18.11 above) in the Sept-Îles Port Area.
- The port will be located at the Pointe Noire terminal at the Sept-Îles Port area, where the concentrate slurry will be received at the product dewatering facility for dewatering of the slurry in order to achieve a dry concentrate at 8% moisture content, transferred to the concentrate storage building, and ultimately reclaimed and conveyed to the ship loader on the wharf to be constructed and operated by the Sept-Îles Port Authorities (SIPA)
- The shipping concept for the dry concentrate is based on the use of the future Phase II deep-water wharf at the Port of Sept-Îles for the future potential use by LOM where the ship loaders will be installed. The installation is designed to load bulk carriers ranging from 180,000 DWT to 400,000 DWT in capacity.
- In support to the FS, a MOU (Memorandum of Understanding) has been prepared and signed between LOM and the Sept-Îles Port Authority for the future potential use by LOM of the Phase II deep-water dock for the shipping of LOM concentrate product while being operated by the Sept-Îles Port Authorities.

18.12.2 Localisation of the Product Dewatering, Storage and Reclaiming Facilities

The product dewatering, storage and reclaiming facilities are located at the Pointe Noire Terminal at the Port of Sept-Îles. The location of the facility is shown on the map in Figure 18.19. This location is on public land that is under the jurisdiction of the Ministry of Natural Resources of the Province of Quebec.

Figure 18.19 – Port Site Location

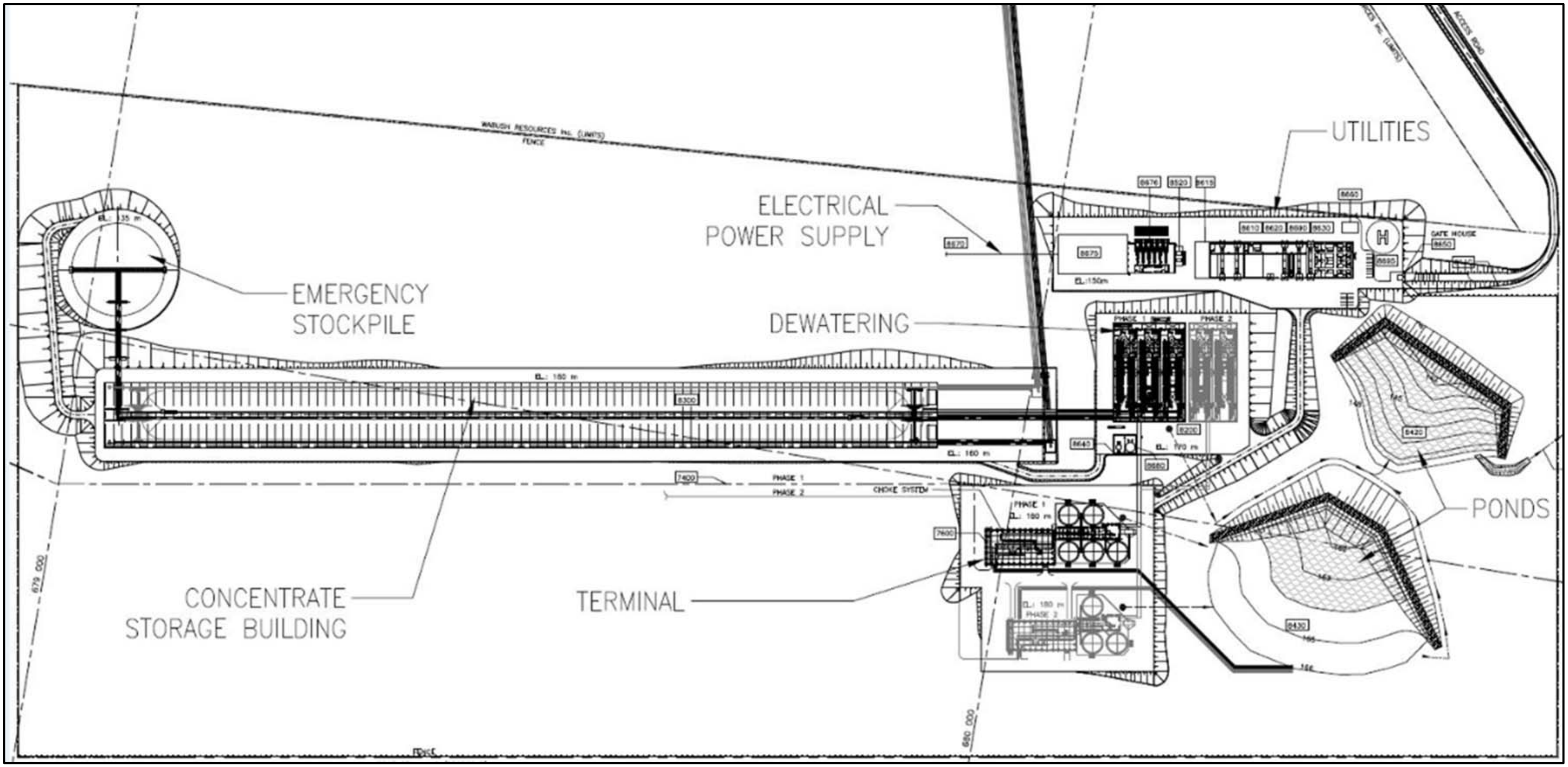


Source: BOTQ 1/20 000, MRNF Quebec, 2014

18.12.3 General Layout of the Product Dewatering, Storage, Reclaiming Facilities

General site layout of the port area facilities are illustrated in Figure 18.20 in the next page here below.

Figure 18.20 – Site Layout



18.12.4 Facility Description

a) PDS Terminal and Slurry Dewatering

Refer to section 17.0 for the description of the PDS terminal and product dewatering plant.

b) Concentrate Storage

The storage/reclaim facility consists of two stacking conveyors, a fully enclosed concentrate storage building, a bridge style bucket-wheel reclaimer, an outdoor emergency stockpile and an overland conveyor and ship loader.

Since the design capacity of each stacking conveyor is 8,000 t/h, one stacking conveyor may be shut down without reducing the dewatering throughput.

For Phase 2, a second reclaimer is added, while the covered stockpile building remains the same. Its holding capacity is based on the tonnage required to load two of the largest ships (380,000 DWT) plus a reserve of 25 %. This adds up to 950,000 tonnes (or 12 days) for Phase 1 and 7 days for Phase 2, regardless of ship availability.

An uncovered discharge area, close to the concentrate building, acts as an emergency overflow in case of problems with ship loading or vessel availability. It can accumulate up to 250,000 tonnes of concentrate in a single cone-shaped pile. Any iron ore concentrate stacked in the overflow discharge area is transported back into the storage building by front-end loaders.

c) Water Management Pond

Water and slurry management in case of upsets in the process will be handled by two ponds which will allow the suspended fine-concentrate particles to settle. As shown in Figure 18.21, the ponds will be created by building retaining dams in a naturally occurring, downward-sloping depression. During upset conditions or during water batching operation, water will be filtered through the upper (emergency concentrate handling) pond to the lower (water management) pond and then released to the St-Lawrence estuary. During normal operation, the filtrate will be discharged directly to the lower pond, the water management pond, from there released in a controlled manner. Refer to Figure 18.21 below.

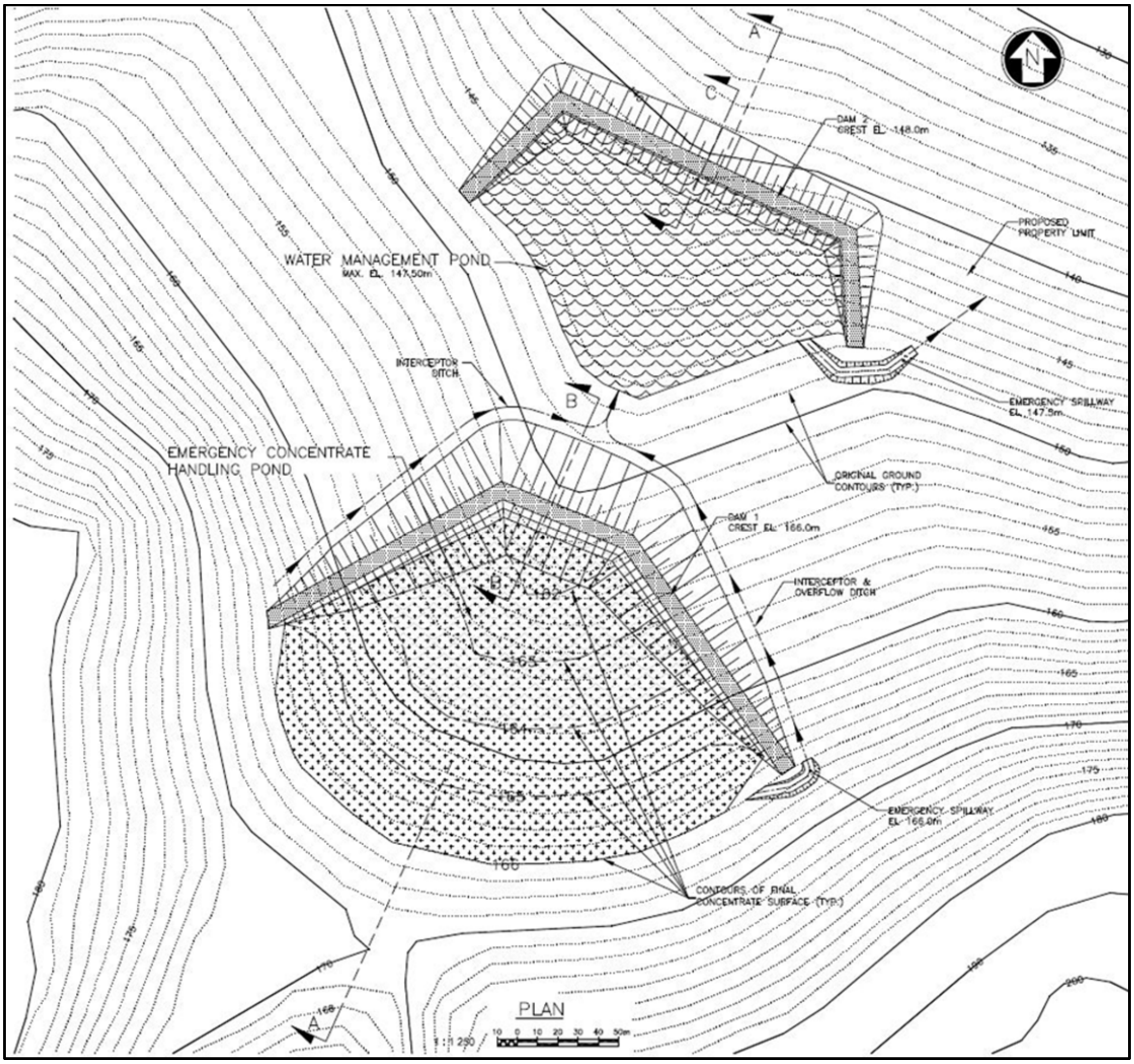
d) Emergency Concentrate Handling Pond

An emergency concentrate handling pond will be constructed upstream of the water management pond. Its dike will be composed of different grades of granular material such that the dike acts as a filter dam.

This pond will be used to either capture slurry from an emergency dump of the PDS or transition water (batch water) containing high amount of suspended solids during batching.

The emergency concentrate handling pond has an effective volume of approximately 89,000 m^3. The pond is sized to hold the volume of concentrate from one PDS pipeline after it has settled out. Refer to Figure 18.21 below.

Figure 18.21 – Water Management Ponds



e) Concentrate Reclaiming and Shiploading

While stockpile replenishment will operate virtually 24/7 (matching the concentrator production rate), the system required to supply material to the ship loaders is only activated once a vessel is moored and ready for loading. The design of this system is based on parameters that minimize demurrage fees and maximize the use of the marine infrastructures.

The ship-loading material-handling system, including the reclaimer, conveyors and ship loaders, is designed for 9,000 t/h nominal capacity and a single shipping berth. This design yields berth occupancy of 69 % when one ship loader is used in Phase 1 and 74 % when a second ship loader, operating simultaneously, is added in Phase 2.

During the ship loading process, iron ore concentrate reclaimed from the stockpile via the travelling bridge reclaimer is loaded onto a reclaiming conveyor, which runs the full length of the concentrate storage building. The concentrate is then transferred to a 1.25 km long covered overland conveyor towards the marine installations. A pier conveyor receives iron ore concentrate from the overland conveyor and transfers it to the ship-loading tripper conveyor.

For Phase 2, a second bridge reclaimer will be installed, and reclaiming will be from both ends of the stockpile. A second overland conveyor and a second ship loader will also be necessary.

18.12.5 Port Infrastructures

a) Firewater

Since no firewater infrastructure is available at the proposed location of the Port facilities, the Project will need to provide its own pumping and distribution network for fire protection. The firewater tank will be replenished with municipal potable water. The firewater tank will have a capacity of 1,200 m^3. Fire pumps, a jockey pump and a diesel pump will be installed near the fire tank. The fire network will be underground and include hydrants and a sprinkler system in the buildings.

b) Sanitary Network

Since the sanitary network does not extend to the Pointe-Noire area, the Project will be required to provide sewer management and disposal. A gravity sewer system will be installed for the collection of all sewage effluent from dewatering plant area. A septic tank will be installed in the plant, along with a septic field and filtration system. The septic sludge will be emptied periodically by local companies.

c) Potable Water

Potable water is available for industrial use. It is expected but not confirmed by the municipality that it would install a pressurized line up to the property limits. In any case, potable water will be stored into a transfer tank. The size of the potable water storage tank will be 100 m^3. The potable water network will be underground and ground-mounted pumps will provide the pressure head for the water systems.

d) Storm Water Network

As the site is currently undeveloped, there is no storm water network. Storm water run-off (from roofs, parking area, etc) needs to be collected and discharged in a controlled manner to the river system. The site layout is configured with a surface network to collect storm water. Sand traps will be installed in the ditches.

e) Solid Waste

It is expected that domestic solid wastes collection and disposal will be provided as part of the city of Sept-Îles domestic waste management; industrial waste has to be managed by the onshore facilities.

f) Fencing

The perimeter of the onshore facilities will be fenced with a single security fence with overhang. Additional fencing will be provided in sensitive areas such as generator stations and main substation.

g) Telecommunications

All telecommunication infrastructure services are available locally; optic fiber runs through the area.

h) Power

Refer to Section 18.2.

i) Incoming Freight for Mine Operations

Incoming freight in the form of containers can be received at La Relance, where a roll-on/roll-off (Ro-Ro) system will load flatcar wagons directly from ships. Because storage is limited at La Relance, the plan is to focus on Pointe-aux-Basques (another terminal managed by Port of Sept-Îles) in case the container freight increases to the levels expected by the Project. A ro-ro system would have to be installed at Pointe-aux-Basques along with dredging, but the area has better access to the railway and room for storage.

Incoming fuel would be handled at the fuel dock operated by Shell.

j) Access Roads

Access roads currently do not reach up to the dewatering facilities; 1.3 km of paved road is expected to be constructed between Chemin de la Pointe Noire and the entrance to the Port facilities site. On the port premises, 3.0 km of gravel roads 6.00 m wide will allow access and servicing of the facilities.

k) Buildings

A single building will house maintenance, services and management for the Pointe-Noire and intermediate slurry pump stations.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Preamble to the Market Study

SNL Metals & Mining is engaged in mining research and analysis. On behalf of its clients, SNL Metals & Mining conducts market and industry surveys, prepares regional exploration and project reports, monitors and analyzes production, ownership and mergers and acquisitions and conceptual and prefeasibility studies. SNL Metals & Mining assists governments with exploration and mining investment promotions, as well as policy studies, and specializes in detailed monitoring and forecasting market development for mining equipment manufacturers and service providers.

This report is based on (i) information and data provided to SNL Metals & Mining by Lac Otelnuk Mining Ltd., (ii) information and data provided to SNL Metals & Mining by third parties and (iii) SNL Metals & Mining's proprietary data. In performing its analyses and preparing this report, SNL Metals & Mining has relied upon the accuracy, completeness and fair presentation of all information, data, advice, opinions and representations obtained from public sources or provided to it from private sources, including Lac Otelnuk Mining Ltd. SNL Metals & Mining has not independently verified such information and has assumed that information supplied and representations made by Lac Otelnuk Mining Ltd. are substantially accurate. No representation or warranty, expressed or implied, is made by SNL Metals & Mining as to the accuracy, completeness or fairness of such information and nothing contained herein is, or shall be relied upon as, a promise or representation, whether as to the past or the future. Neither SNL Metals & Mining nor any of its affiliates takes any responsibility for the accuracy or completeness of any of the accompanying material. SNL Metals & Mining's maximal liability for whatever reasons is limited to total fee paid for this study.

To the extent that any of the assumptions or any of the facts on which this report is based prove to be untrue in any material respect, this report cannot and should not be relied upon. Furthermore, in SNL Metals & Mining's analysis and in connection with the preparation of this report, SNL Metals & Mining has made numerous assumptions with respect to industry performance, general business, market and economic conditions and other matters, many of which are beyond the control of any party involved.

SNL Metals & Mining has prepared this report effective as of the date hereof. This report is necessarily based upon market, economic, financial and other conditions as they exist and can be evaluated as of the date hereof, and SNL Metals & Mining disclaims any undertaking or obligation to advise any person of any change in any fact or matter affecting this report which may come or be brought to the attention of SNL Metals & Mining after the date hereof.

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The analyses, opinions, and conclusions presented in this report apply to this engagement only and may not be used out of the context presented herein. This report is valid only for the effective date(s) specified herein and only for the 10 purpose(s) specified herein. This report is furnished solely for the use and benefit of Lac Otelnuk Mining Ltd. and is not intended to, and does not, confer any rights or remedies upon any other person, and is not intended to be used, and may not be used, by any other person or for any other purpose, without SNL Metals & Mining's express consent.

19.2 General

19.2.1 Iron Ore and Its Uses

Iron ore is the raw material used to make pig iron, which is one of the main raw materials used to make steel. 98 % of all iron ore mined in the world is used for steel production. Iron ore comprises mineralization of iron from which metallic iron can be economically extracted. These ores are usually rich in iron oxides and vary in colour from dark grey, bright yellow, deep purple to rusty red.

The iron itself is usually found in the form of magnetite (Fe_3O_4), hematite (Fe_2O_3), goethite, limonite, siderite (carbonate ore), pyrite (sulphide ore) or other less common iron sources such as chamosite, lepidoccite and chalybite. Hematite (also haematite) is also known as "natural ore", in reference to certain hematite ores containing 66 % iron that can be fed directly into iron making blast furnaces. Hematite is, together with magnetite, the main form of iron ore used in steelmaking.

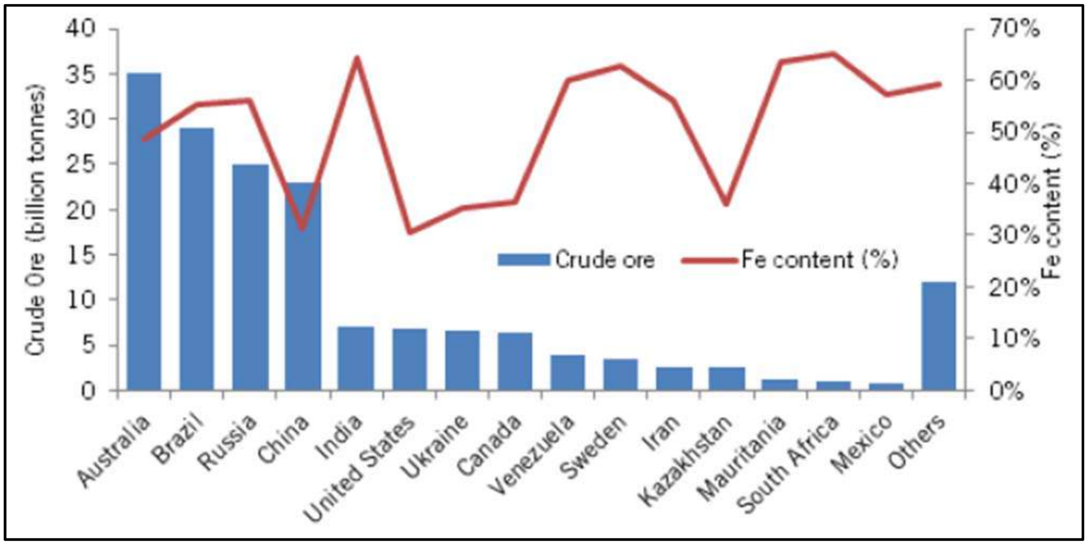
Table 19.1 – Major Iron Compounds

Compound name	Formula	% Fe
Hematite	Fe_2O_3	69.9
Magnetite	Fe_3O_4	72.4
Goethite/Limonite	$HFeO_2$	~63
Siderite	$FeCO_3$	48.2
Chamosite	$(Mg,Fe,Al)_6(Si,Al)_{414}(OH)_8$	29.61
Pyrite	FeS	46.6
Ilmenite	$FeTiO_3$	36.81

Source: SNL Metals & Mining

Pure magnetite and hematite contain 72.4 % and 69.9 % iron, respectively, while siderite contains only 48.2 % iron. Because of its variable hydrated nature, the iron content of pure limonite can range between 38 and 51 %.

Figure 19.1 – Global Iron Ore Reserves



Source: USGS

The US Geological Survey has estimated that there are around 170 billion tonnes of iron ore resources worldwide, containing more than 81 billion tonnes of iron. The largest iron ore reserves are found in Australia, Brazil, Russia, China, India, United States, Ukraine and Canada.

Iron ore deposits almost always occur as an intimate mixture of one or more of the iron bearing minerals referred to above plus a number of unwanted or 'gangue' constituents. Typically, high-grade iron ore will contain 60-68 % iron content whereas low to medium grade ores can contain 20-60 % iron.

In almost all mining operations some degree of beneficiation (i.e. ore upgrading) is carried out before the ore can be used to produce iron. This may involve a simple crushing, washing and screening operation in the case of high-grade ores. Lower grade ores require more sophisticated methods of beneficiation to produce the high iron grades demanded by steel makers. Hematite iron is rarer than magnetite, but it is considerably cheaper/easier to beneficiate requiring less energy to crush and grind. However, hematite typically contains significantly higher concentrations of impurities attracting penalties in trade.

19.2.2 Iron Ore Products

Iron ore is supplied to the world's iron and steel industry in four main forms:

- Lump ore: added directly to the blast furnace (BF);
- Sinter fines: used to make sinter at the steel mill; sinter is an agglomerated product added to the blast furnace;
- Pellet feed/concentrates: used to make pellets;
- Pellets: an agglomerated ball-shaped product (about 1 cm in diameter), generally produced at the iron ore mine, shipped to the steel mill and added to the blast furnace to produce pig iron.

Pellets generally have a higher iron content and are therefore more productive than sinter or lump ore in a blast furnace. They are, however, usually also more expensive to produce than the other products. The iron content in lump ore may vary, but it is sought after by steel mills as it also increases the productivity of a blast furnace, although not to the same extent as pellets.

a) Lump ("Direct Shipping") Ore

Hematite is the mineral form of Iron (III) oxide, (Fe_2O_3). High grade hematite is often also referred to as ‘direct shipping ore’ or DSO, because it is mined and beneficiated using a relatively simple crushing & screening process pre-exporting for use in steel mills.

Lump is produced by crushing the mined ore to a top size of around 60 mm which is then screened at both 50 mm and 6 mm. The >50 mm material is recycled to be re-crushed while the >6 mm and <50 mm size fraction is extracted as the usable lump product. It is supplied in the nominal size range >6 mm<100 mm and is produced in a range of different iron grades.

Lower grade lump varies in iron content up to 64 % and is generally used as feed material to blast furnace (BF) iron-making operations, together with sinter, in proportions from 0-100 %, although in modern, high productivity BF operations, 5-30 % lump ore is typical. Because of lump’s particle size, air/oxygen can circulate around them easily thus efficiently melt them.

High-grade lump typically contains between 64 % and 68 % Fe and (assuming the correct metallurgical properties) is almost exclusively used in direct reduction (DR) iron-making where it usually commands a premium price. It is common to limit the use of lump in DR operations to between 30 % and 40 % of the total feed, with the remainder being DR pellets.

b) Sinter Fines

Fines are iron ore particles with size less than 6 mm, which require some form of agglomeration (sintering, pelletizing) before being fed into a blast furnace. The <6 mm fraction of the lump ore screening and crushing processes is commonly deslimed (assuming water is available) during which some of the <1 mm material may be removed.

The resulting >1 mm to <6 mm product is known as sinter fines and is supplied to sinter plants. These are generally located at integrated steel works where the sinter fines are fused with fluxes and coke breeze to produce the finished sinter product for use in blast furnace iron making operations.

Sinter fines can vary significantly in iron content, but generally they tend to be within the range of 58 % to 62 % Fe. For reasons related to sinter quality, the vast majority of sinter fines are hematite, siderite, limonite or goethite ores, although some operators in China are known to use significant amounts of magnetite ores in their sinter-making process.

Fines, due to their smaller size, block air movement in the blast furnace leading to inefficiencies. Lump is preferred to fines in blast furnaces due to their ability to melt more efficiently and therefore generally trade at a premium. The advantage of using fines, however, is that it can be blended to ensure the desired concentration of iron ore and contaminant materials to be fed to the blast furnace.

c) Pellet Feed and Concentrates

Certain ores are not of sufficiently high quality to generate a useable product without significant beneficiation. Almost all of these ores require the crushed product to be finely ground before it undergoes physical beneficiation to remove excess gangue material.

The iron grade of the concentrate will depend on the original grade of the feed material and the degree of beneficiation carried out. The main difference between pellet feed and other iron ore concentrates is the grain size, where pellet feed is ground in order to achieve optimal production in the pelletizing operation.

d) Pellets

After the iron ore is beneficiated, the concentrate tends to have a fine average particle size (almost all <1 mm), and it therefore requires pelletizing (agglomeration) to allow ease of handling and use in the blast furnace or Direct Reduction (DR) shaft. While pelletization refers to ultra-fines (<0.15 mm), sintering refers to fines (0.15 mm to 6.3 mm in diameter). Sinter plants are usually located near the steel mills since the sinters tend to be friable and cannot be transported to large distances.

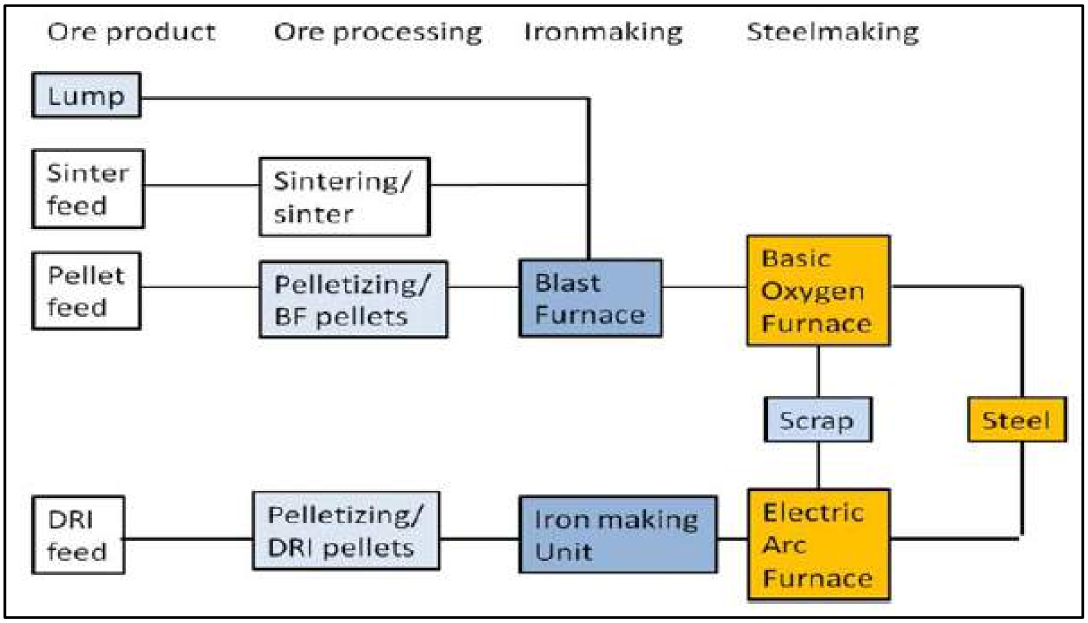
Pelletizing involves the balling using a binding agent (clays) and subsequent drying and firing of the fine, concentrated material in order to produce a hard but porous spherical ball, generally of between 6 mm and 18 mm diameter. These operations prefer a close size-range distribution and consistent physical and chemical properties for optimum performance in use.

Concentrates containing between 62 % and 65 % Fe are suitable as feed material for producing blast furnace pellets. Those of higher grade, usually containing between 65 % and 68.5 % Fe are highly sought-after as feed material to make high-grade pellets suitable for DR iron-making processes.

The quality of output is influenced by the nature of the binding reagents employed. Pellets tend to differ in composition, as the product specifications are set by the steel mills. Pellet plants tend to be located near the source mine site. The problem with producing pellets is the high energy required to produce pellets from fines, high capital costs of constructing a pellet plant and the stringent and consistent requirement of fines feed. Pellets are considered a high "value-in-use" iron ore product and can carry a significant premium in price, particularly the high-grade pellets normally associated with DR use.

An outline of different process route for steelmaking and where different iron ore products are found in the routes is found in Figure 19.2.

Figure 19.2 – An Outline of Different Production Routes to Steelmaking



Source: SNL Metals & Mining

e) Iron Ore Requirements

There are certain minimum physical, metallurgical and chemical properties that are required to ensure that the ore is useable and therefore saleable. The physical and metallurgical properties of each ore are fairly specific to a particular product; however, there are some characteristics common to all:

- Lump ores and pellets should demonstrate good reduction characteristics for use in the blast furnace or DR processes.
- The demands on sinter fines are less exact, but they must exhibit good agglomeration characteristics in order to provide sinter with the required strength and reducibility.

f) Iron Ore Quality and Prices

The price paid for iron ore reflects mainly the Fe content and the content of impurities in the product. A host of other factors also influence the price, most importantly the size of the ore and its physical properties and its detailed chemical composition, including its moisture content. In some cases the steel plant may also be interested in some of the alloying metals that might be present, such as manganese.

Given the complexity of the operating parameters of the blast furnace and the varying properties of the final steel product, it is difficult to forecast the exact effects of all the properties in any iron ore in advance. A key aspect is always the consistency of the quality of the product. A higher content than ideal of some of the un-wanted elements may be mitigated by addition of a certain flux or other operational procedures, but if the quality varies this becomes difficult. In addition, ore prices are of course influenced by market parameters.

Table 19.2 – Comparison of Established Iron Ore Indices and the Composition of the Lac Otelnuk

	MBIOI	TSI 62	Platts IODEX	Lac Otelnuk
Price	USD/dry metric tonne, CFR China	USD/dry metric tonne, CFR China	USD/dmt, CFR main Chinese ports	
Fe content	Base 62 %, range 56-68 %	62 %	62 %, range 60-63.5 %	69 %
Silica	Base 3.5 %, max. 6.0 %	4 %	4.50 %	2.95 %
Alumina	Base 2.0 %, max. 4.0 %	3.5%	2.00%	0.02%
Phosphorus	Base 0.05 %, max. 0.1 %	0.07 %	0.08 %	0.02 %
Sulphur	Base 0.02 %, max. 0.05 %	0.05 %	0.02 %	0.01 %
Moisture	Base 8 %, max. 10%	8 %	8 %	8 %
Granularity	>90 %:<6.3 mm, <10 %:<0.15 mm	>90 %: <10 mm, <40%: <0.15 mm	>90 % <10 mm	

Sources: Metal Bulletin, The Steel Index, Platts Lac Otelnuk Mining Ltd

i) Fe Content

Iron ore, as stated earlier in the report, occurs naturally in a number of forms. In Lac Otelnuk, the principal form is magnetite (Fe_3O_4) with a theoretical iron ore content of 72.4 % Fe. Most blast furnace operators are looking for an iron ore of at least 62-64 % or higher and in the case of Lac Otelnuk, the concentrate has a grade well above this with an indicated Fe content of 69 %.

ii) Other Elements in Iron Ore

The inclusion of even small amounts of some elements in the iron ore can have profound effects on the behavioural characteristics of the blast furnace. These effects can be both good and bad. Some chemical elements are deliberately added, such as a flux to make a blast furnace more efficient.

The choice of ore, fuel, and flux also determines how the slag will behave and the characteristics of the iron produced. Ideally iron ore contains only iron and oxygen. In nature this is rarely the case. Typically, iron ore contains a host of elements which are often unwanted in the iron making process. Examples of such elements are:

– SiO_2 Silica, Silicon Dioxide
– Al_2O_3 Aluminium Oxide
– CaO Calcium Oxide

– MgO	Magnesium Oxide
– S	Sulphur
– P	Phosphor
– TiO_2	Titanium Dioxide
– Mn	Manganese
– Zn	Zinc
– Na_2O	Sodium Oxide, Soda
– K_2O	Potassium Oxide

<u>Silica</u>

Silica (SiO_2) is almost always present in iron ore. Most of it is slagged off during the smelting process. But at temperatures above 1,300 °C some will be reduced and form an alloy with the iron. The hotter the furnace, the more silicon will be present in the iron. The silica content of the Lac Otelnuk concentrate is 2.95 %, which is well within the acceptable limits and better than many other products on the market.

<u>Aluminium</u>

Alumina is very difficult to reduce and therefore aluminium contamination of the iron is not a problem. However, alumina does increase the viscosity of the slag and requires the addition of basic flux to maintain slag viscosity within the appropriate range. At extremely high levels of alumina, the high slag viscosity could lead to a difficulty in tapping the blast furnace, disrupting production. The alumina content of the Lac Otelnuk concentrate is 0.02 %, which is very low and well within acceptable limits.

<u>Magnesium Oxide</u>

Besides contributing to the basicity of the slag in the furnace, magnesium oxide has other beneficial traits. Exchanging calcium oxide for magnesium oxide raises the softening temperature of pellets in the furnace and thus contributes to better aerodynamic conditions in the furnace and higher productivity. This is, however, not relevant to Lac Otelnuk concentrate, which contains 0.15 % MgO.

<u>Sulphur</u>

Sulphur in iron ores fed to sintering or pelletizing plants is mostly oxidized and reports to flue gas. Depending on the efficacy of pollution controls and local emission regulations, elevated levels could be an environmental issue. Sulphur in charge materials to iron making operations will be distributed between hot metal and slag. A desulphurization step is included in the hot metal processing to reduce sulphur in steel to acceptable limits. The sulphur content of the magnetic separation concentrate from Lac Otelnuk is negligible at an average of 0.01 %.

<u>Phosphorus</u>

Phosphorus comes from iron ore mixed with phosphate deposits. It is reduced chemically and is dissolved in the liquid iron. Much of the phosphorus is subsequently oxidised in the steel making process and passes into the slag where it remains. In the finished steel phosphorus contributes to hardening of the steel but it also makes it more brittle, especially in the quenched and tempered condition. This results in reduced ductility and impact toughness. A low content of phosphorus in the iron ore products, as in the Lac Otelnuk case (maximum 0.02 %), is thus desirable.

Manganese and Titanium

Manganese is used as alloying element in some steel products. Accordingly, it may be considered as beneficial. However, in today's steel making, precise compositions on a ppm basis makes the steelmaker inclined to adjust the final compositions as late in the steelmaking process as possible. Because of the need to control the content of alloying elements as precisely as possible, manganese may entail a penalty rather than a premium. For DRI pellets, the manganese content needs to be low. In the Lac Otelnuk case, the manganese content is on average 0.16 %.

Alkali Metals

Alkali metal group oxides (usually referred to as "alkalis") should be as low as possible. In particular K_2O and Na_2O are typical found in low concentrations in iron ores. K_2O and Na_2O are reduced and volatilized in lower part of the blast furnace, and then condense in the upper shaft where they cause scaffold buildup and degradation of coke. To control alkali recirculation, the slag basicity ratio $(CaO+MgO)/(SiO_2+Al_2O_3)$ can be lowered which improves alkali output in slag but decreases the sulphur capacity of the slag and increases sulphur content of hot metal. Lac Otelnuk concentrate has a low alkali content, with K_2O and Na_2O both lower than 0.01 %.

iii) Consistency of the Chemical Analysis

When setting up production in the blast furnace, what the producer is looking for is consistency in the chemical analysis of the iron ore. This is to ensure that production can run at the most efficient pace. Often the steel mill buys ore in contracts spanning several years to ensure that it receives consistent quality. Buyers can also buy iron ore from several different producers and then blend the ore themselves to in order get the desired properties of the iron ore. Price considerations may also argue in favour of blending.

iv) Conclusions Regarding the Lac Otelnuk Composition

Swerea MEFOS says in its report[1]: "Lac Otelnuk concentrate is a fine-grained high grade magnetite concentrate. The main element composition is favourable for blast furnace iron making. Technical it can also be considered for use in direct reduction pellets if blended with higher grade material, but the acid gangue content is somewhat high which can make this commercially unviable. The ore is predominantly magnetite with Fe contents over 68 % and generally low to very

[1] See separate report from Swerea MEFOS (not available as part of this report).

low in elements deleterious for iron and steel making including P and Al_2O_3. Silica is the main diluting compound at circa 3 %." Swerea MEFOS concludes that the most suitable application of the concentrate is as a pellet feed to produce blast furnace pellets, it might also be considered for DR pellet feed if blended.

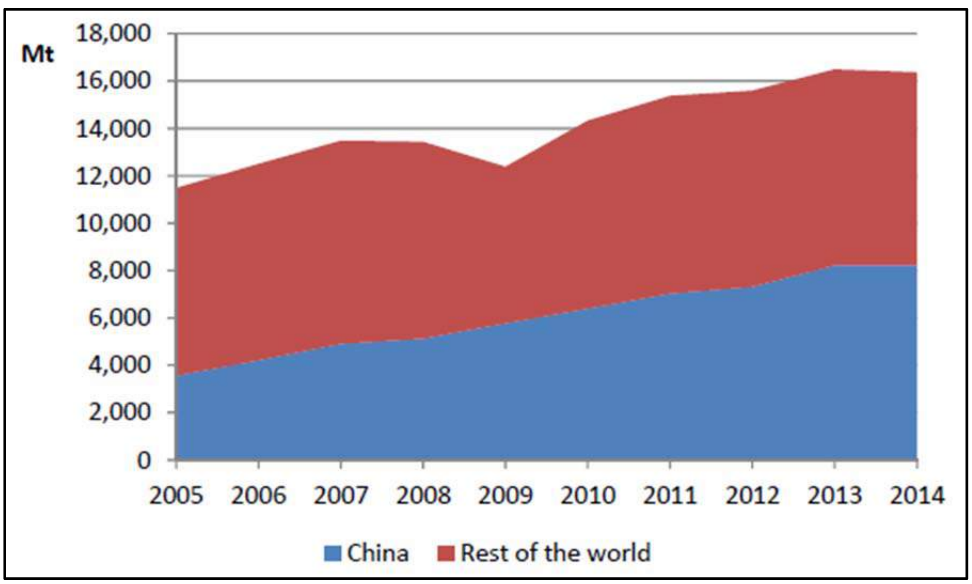
19.3 The World Steel and Iron Ore Industries

19.3.1 Steel

The world steel industry consumes more than 98 % of all iron ore. After having grown at a rapid pace after the Second World War until the mid-1970s, it shifted into a slow growth mode, with annual growth ranging from 1 % to 2 %, from the mid-1970s to the late 1990s. The rebuilding of infrastructure in the West had long finished and growth in steel use reflected incremental increases in material standards.

At the onset of the 21st century the market changed. China overtook Japan to become the world's leading steel producing country in 1996, ushering in a period of exceptionally robust growth at an annual rate of 6 % to 7 % from 2000 to 2007, with world crude steel production growing from 750 Mt in 1996 to 1,348 Mt in 2007 and to 1,637 Mt in 2014, despite the impact of the financial crisis and recession. Almost all of the increase in production was accounted for by China, where crude steel output grew seven-fold (from 101 Mt to 823 Mt) over the same seventeen-year period (see Figure 19.3).

Figure 19.3 – Crude Steel Production, 2005 to 2014 (Mt)

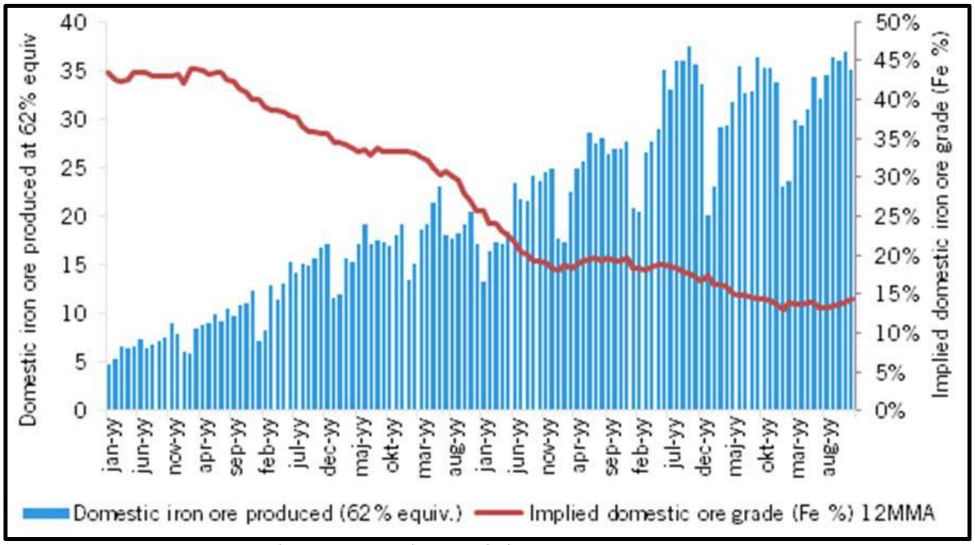


Source: World Steel Association

19.3.2 Iron Ore Production

The world iron ore market is dominated by Australia and Brazil. China is also a large producer, although China's production is exclusively for domestic use. It should be noted that unless otherwise stated all figures concerning China's production in this report have been converted so as to be comparable with production volumes in other countries (See Figure 19.4). The reason for this is that although the total volume of China's production is by far the largest in the world, the grade is so low as to make comparison with other countries on a gross tonnage basis meaningless.

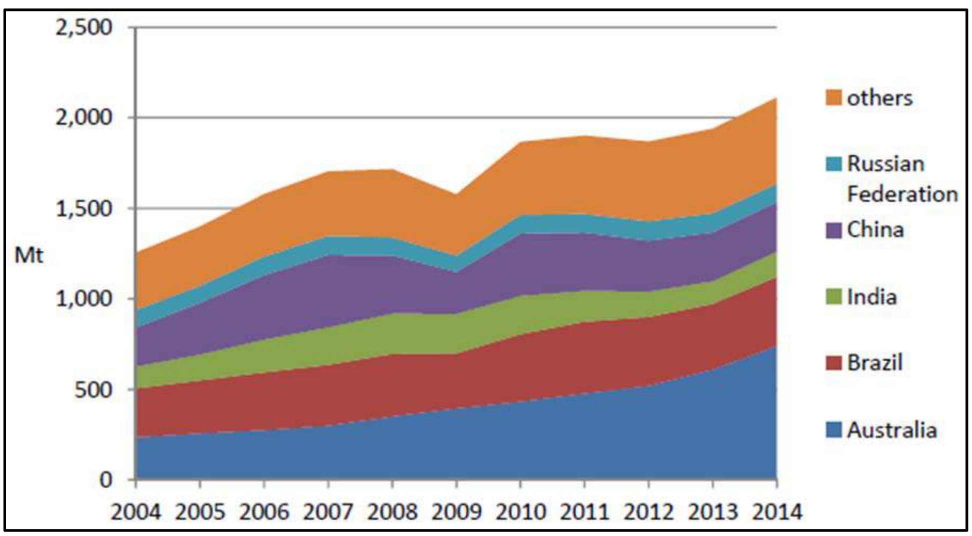
Figure 19.4 – Chinese Domestic Iron Ore Production (62 % Equivalent) and Implied Iron Ore Grade, January 2004 – December 2013



Source: CISA, MySteel, SNL Metals & Mining

Accordingly, Chinese production is recalculated using reported iron ore imports and pig iron production so as to correspond to 62 % Fe content, which is roughly the average grade of internationally traded iron ore (see Figure 19.5).

Figure 19.5 – Iron Ore Production, 2004 to 2013 (Mt)

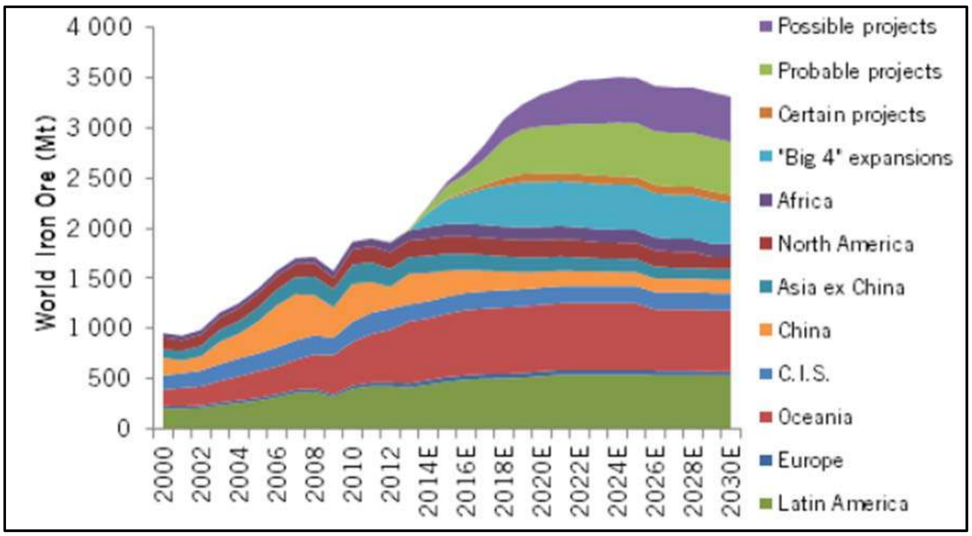


Source: UNCTAD, RMD Iron Ore

Global iron ore production is currently around 2.11 billion tonnes having increased substantially from 1.16 billion tonnes over the past decade. The majority of this supply growth has been from Brazil and Australia with the Vale, Rio Tinto and BHP Billiton being largely responsible. Australia and Brazil are expected to contribute to the majority of near term supply. However towards the end of this decade new supply from West Africa could start to emerge with the anticipated start-up of Rio Tinto's Simandou project in Guinea scheduled for 2018 at the earliest. In addition increased output from Sierra Leone and Liberia and potential new supply from Gabon and Cameroon could contribute to this.

Developing countries accounted for 16.4 % of world iron ore production in 2014 (down from 17.3 % in 2013), the CIS countries for 9.2 % and the industrialized economies almost 83.6 %. The decrease in the share of the developing countries has been due mainly to growth of production in Australia, up by 130 Mt, and Brazil (up by 18 Mt) while Ukraine fell by 10 Mt. Chinese production was 273 Mt, 12.9 % of total world production, down from 13.9 % in 2011.

Figure 19.6 – World Iron Ore Historic and Potential Supply, 2000 - 2030E (Mt)



Source: UNCTAD, SNL Metals & Mining (China (adjusted to world average Fe content)

In general terms, Australia and less so Brazil will be the dominating forces in future supply of iron ore driven largely by the Big Four producers which accounted for 70 % of seaborne supply in 2013: Rio Tinto, Vale, BHP Biliton and Fortescue. The Big Four are forecast to collectively increase iron ore output by over 200 Mt over the next two years having already added around 170 Mt since 2010 (see Figure 19.6 above and Table 19.3 below).

India, which has large and good quality resources of iron ore, will be hampered by red tape, export restrictions and an on-going internal struggle of use. As of early 2014, India's Supreme Court maintained a 14-month ban on iron ore mining in the state of Goa, although it will allow the sale of more than 11 Mt of material that has been sitting in stockpiles. Nonetheless, over time, as the Indian steel industry grows, we expect most of the iron ore will be used domestically; however given the large number of small scale iron ore mines in India (c.100) we anticipate that it may be a number of years before India becomes a net importer of ore.

Elsewhere, exports from Iran, which account for around 20 Mt/y, could dry up as the country implements a 25 % export duty to preserve domestic supply. As we have previously forecasted Chinese iron ore output will slowly fall, as low grade high-cost mines are shut, but the rate of decline may increase with sustained low iron ore prices.

Table 19.3 – The "Big Four" Dominate Near Term Supply Growth

Trade (Mt) (100% basis)	2011	2012	2013	2014E	2015E
Rio Tinto (Pilbara)	**231**	**239**	**251**	**279**	**324**
y-o-y		4%	5%	11%	16%
Increment		8	11	28	45
Vale	**312**	**309**	**300**	**321**	**348**
y-o-y		-1%	-3%	7%	8%
Increment		-3	-9	21	27
BHP (Pilbara)	**162**	**175**	**219**	**224**	**244**
y-o-y		8%	25%	2%	9%
Increment		13	43	5	20
Fortescue	**48**	**61**	**96**	**145**	**155**
y-o-y		28%	57%	51%	7%
Increment		13	35	49	10
Other	**367**	**359**	**365**	**444**	**496**
y-o-y		-2%	2%	22%	12%
Increment		-8	6	79	52
Total seaborne	**1,120**	**1,144**	**1,230**	**1,412**	**1,566**
y-o-y		2%	8%	15%	11%
Increment		24	86	182	154
Big Four	**753**	**785**	**865**	**968**	**1,070**
y-o-y		4%	10%	12%	11%
Increment	56	32	80	103	102
Share of total growth		na	93%	57%	66%
Share of total supply	67%	69%	70%	69%	68%

Source: Company reports, SNL Metals & Mining. (*Production on 100% basis)

19.3.3 Iron Ore Supply Issues

On a global level, the "Big 3" face increased competition in the medium to long term, but this competition is not likely to come from complete newcomers to the industry, as the magnitude of an investment into iron ore is larger than in any other metal. The average iron ore project continues to grow, and it is now over 1.6 billion USD including extensive infrastructures such as ports and railroads.

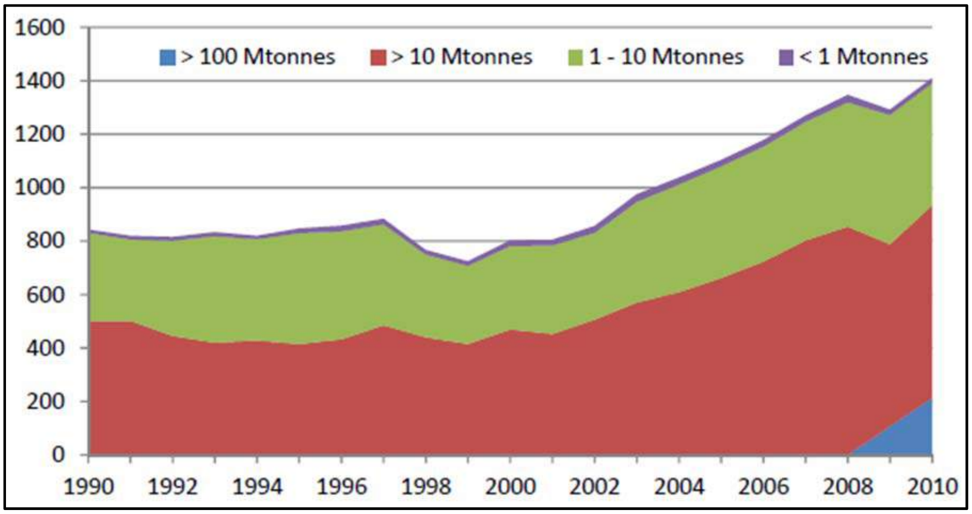
The high costs are partly a reflection of the physical size of an iron ore mine, which are often 10 times the size of a gold project; 10 Mt compared to 1-2 Mt in annual ore production. But also the demand for power, transport facilities for the final product and other infrastructure, increase the investment costs as iron ore is a bulk commodity. This makes it difficult for companies without large financial resources to enter the sector. Furthermore, the majority of green field iron ore projects, not owned by a major, are often located in challenging areas due to lack of infrastructure or political uncertainty.

Iron ore projects are often burdened with investments in transport infrastructure, frequently constituting the major part of the investment costs. The battle fought in and out of courts concerning the rights to use the existing railway systems in Western Australia was but one good example of the importance of the infrastructure. Therefore, in addition to financial strength, considerably bolstered by the recent year's large iron ore price increases, the established major producers have advantages that decrease their marginal costs of expansion. Firstly, they are able to expand existing operations, often through relatively minor additional investments in upgrading their transport systems. Secondly, they can open up new deposits close to existing mines, thus reducing the costs by expanding already existing process plants. With falling prices those who will be hit the hardest are the newcomers who started production in the last one to three years when prices was high and rising.

In addition, it is more difficult to market an iron ore product than a comparable copper or base metal concentrate, not to mention a gold doré bar. Iron ore is a much more heterogeneous product and physical trade on commodity exchanges is still too thin for them to function as markets of last resort. A new producer has to sell directly to a consumer and convince the potential customer that the product will be of consistent quality over a long period of time and that it will be delivered regularly and in time. Accordingly, any completely new producer is likely to meet with major difficulties in entering the market. Instead, more serious competition may come from established second tier producers, particularly in countries where the transport situation is similar to that of the dominating companies.

Iron ore mines handle by far the largest volumes of ore and rock of all metal mines. As can be seen in Figure 19.7, large (those with production volumes from 10 Mt to 100 Mt) and very large (those with production volumes exceeding 100 Mt) mines account for roughly 60 % of all iron ore production (i.e. of all mines with production figures reasonably available, excluding most small mines in China). The very large mines (Hamersley, owned by Rio Tinto, and Carajas, owned by Vale) only entered this bracket during the last couple of years. The closest in size, Robe River (also controlled by Rio Tinto), currently produces around 60 Mt. Medium size mines with an annual production of less than 10 Mt comprise around 40 % of all iron ore output. It is also in this size bracket that we expect the bulk of new capacity coming on stream.

Figure 19.7 – Iron Ore Production, 1990 to 2010, by Mine Size (Mt)



Source: RMD Iron ore

a) Pellets

The share of pellets in total iron ore production has decreased since the late 1990s. In 2001 the share was almost 25 %; in 2009 it was 19 % and it has remained at roughly this level since then, mainly because steel plants that have some flexibility in the choice of materials tend to use less pellets when demand is low. A major factor behind the decrease in the share of pellets in world production has been the decline in United States iron ore output (pellets account for a large share of blast furnace feed in the United States). Worldwide, several new pellet plants are under construction or are being planned. Quite a few of these were idle when the global financial crisis first began, but have since recommenced production.

It appears fairly certain that pellets production in China has been underestimated in the past and may still be so. Our current estimate is that China produced 135 Mt of pellets in 2012. However, this does not seem to be sufficient to make up for the relatively low sintering capacity. We currently assume that there are at least 24 pellet plants in China of which 7 or 8 are not connected to companies owning iron ore mines. This means that there is ample room for imports of pellet feed.

b) Hematite

Hematite and magnetite each account for about half of world production. Where possible, hematite deposits are the first choice since developing magnetite deposits is costlier. The added costs relating to developing magnetite come from the required fine grinding and beneficiation needed to bring up the Fe content.

The Fe-content is generally lower in magnetite ore but when ground the gangue (such as CaO and SiO_2) can be separated from the ore. There is, however, growing interest in magnetite because of the strong growth in global demand for iron ore, which cannot be met from hematite deposits alone. China's domestic iron ore resources are mostly magnetite, which is also the case in the United States, the CIS and Sweden. In order to feed China's growing number of pellet plants, magnetite deposits are being developed in Australia, where in the past the focus has been on hematite direct shipping ore, in the forms of lump ore and sinter feed. Magnetite deposits may also be developed in Ukraine, Kazakhstan, Mongolia and Bolivia and, for the long term, India. While magnetite is mainly used in pelletizing plants in China, sintering plants use hematite, of which the major part is imported from Australia and Brazil. Pelletizing plants can also accept a certain proportion of hematite.

European and Japanese sinter plants tend to prefer hematite to magnetite, but flexibility and openness to options is necessary for cost reasons and because of diminishing availability of high grade hematite. Alternatives to iron ore from Brazil or Australia, as well as ways to use lower grade ores, are being looked at in the steel industry.

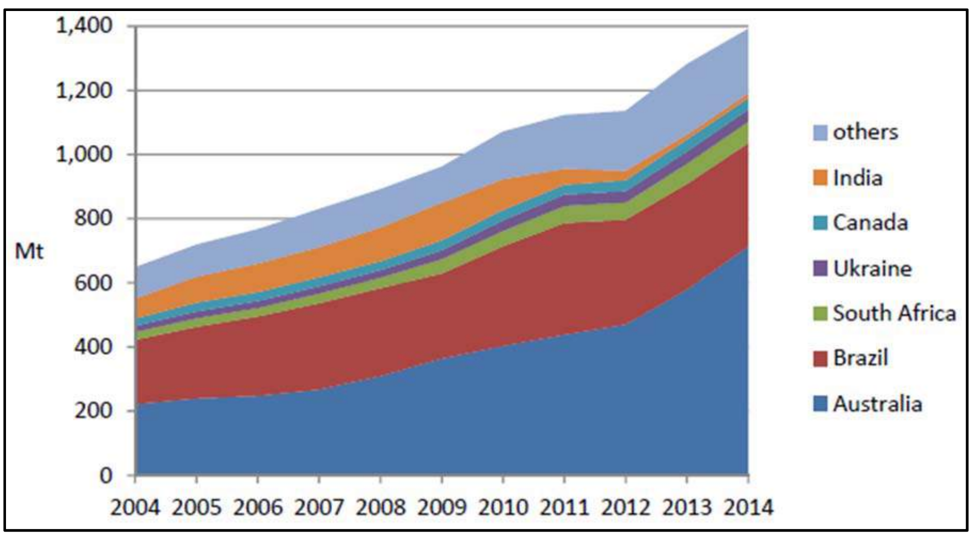
c) Conclusions: the fit for Lac Otelnuk products

In conclusion, the main use for the Lac Otelnuk concentrate would appear to be in pelletizing plants, either for blast furnace pellets or direct reduction pellets.

19.3.4 Iron Ore Trade

Total iron ore exports have increased by 115 % since 2004 and reached 1,392 Mt in 2014. Figure 19.8 and Figure 19.8 show the development of exports and imports since 2004. It deserves to be noted that most importing countries have not yet regained their 2008 import levels.

Figure 19.8 – Iron Ore Exports 2004-2014 (Mt)

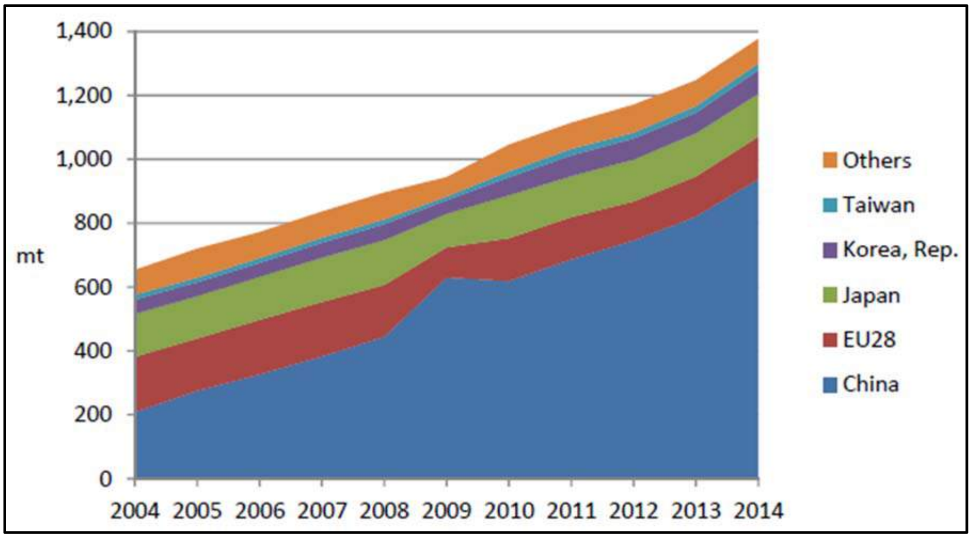


Source: UNCTAD; SNL Metals & Mining

Australia and Brazil together account for more than 87 % of world exports. All other countries are much less important, with the third exporting country being South Africa, which exports only a fifth as much as Brazil. India used to be a large exporter, but regulatory problems and export taxes have resulted in a dramatic fall in both production and exports.

In 2003 China passed Japan to become the world's largest iron ore importer and it has remained by far the most important importing country, accounting for two thirds of total world imports in 2014. The EU, Japan and South Korea are the other significant importers, while no other importing country accounts for more than 2 % of total imports.

Figure 19.9 – Iron Ore Imports 2004-2014 (Mt)



Source: UNCTAD; SNL Metals & Mining

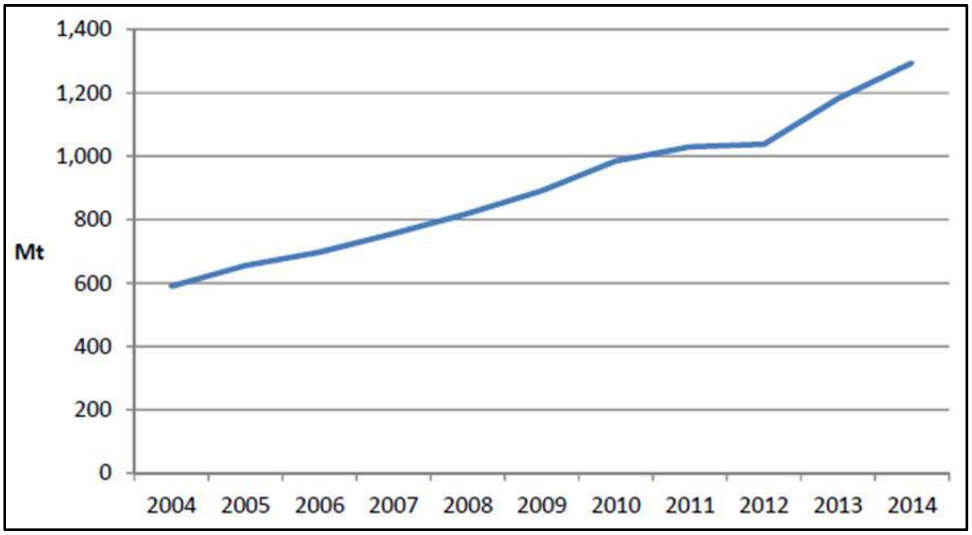
a) Seaborne Iron Ore Trade

We estimate that seaborne iron ore trade was about 1,294 Mt in 2014. The increase in the last several years has been almost entirely due to higher Chinese imports, with trade in other parts of the world stagnating. Figure 19.10 shows the development of seaborne trade over the past several years.

Iron ore and coal each account for about 40 % of total dry bulk trade, with grains the only other commodity of major importance (15 %). Total dry bulk trade declined in 2009, when trade decreased in almost all commodities except iron ore due to the financial crisis. Freight rates then fell dramatically as a result of the drop in demand. A turning point in freight rates was due to happen in any case, since deliveries of new ships was just catching up with the growth in demand after several years when demand for freight services increased at a faster rate than the available fleet. However, the financial crisis brought it on sooner than it would otherwise have happened. During the good years, very large numbers of ships were ordered and these were delivered just as demand had fallen.

It will take several years to work off the excess tonnage, in spite of the fact that many orders have been cancelled. Deliveries of new vessels are particularly large for "Cape Size" type ships, that is, the size most commonly used for iron ore freights. The increase in fleet size will continue, partly because iron ore producers, particularly Vale, are increasingly running their own shipping operations, either through long time charters or through acquisition of their own ships. Fortescue has also decided to acquire its own ships.

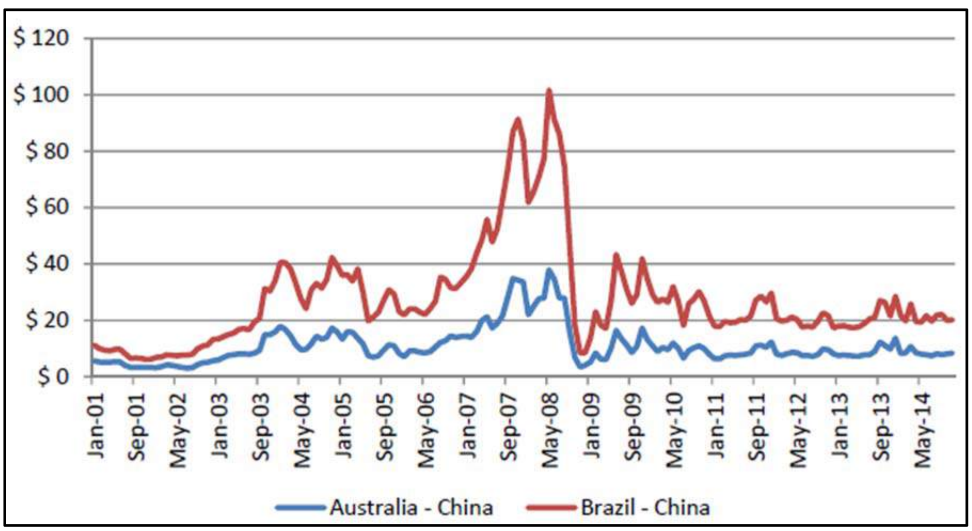
Figure 19.10 – Seaborne Iron Ore Trade 2004-2014, Mt



Source: SNL Metals & Mining

With world trade growing only slowly as a result of weak economic growth in all major economies, any rises in freight rates – caused, for instance, by port congestion – are likely to be temporary and the market will likely remain weak until 2015 at least. Eventually the excess capacity will be absorbed, provided that new orders do not pick up again, a possibility that cannot be excluded. The development of freight prices can be found in Figure 19.11.

Figure 19.11 – Average Freight Rates for 150,000-160,000 dwt Ships, 2001 – November 2014 (USD/Wet Tonne)



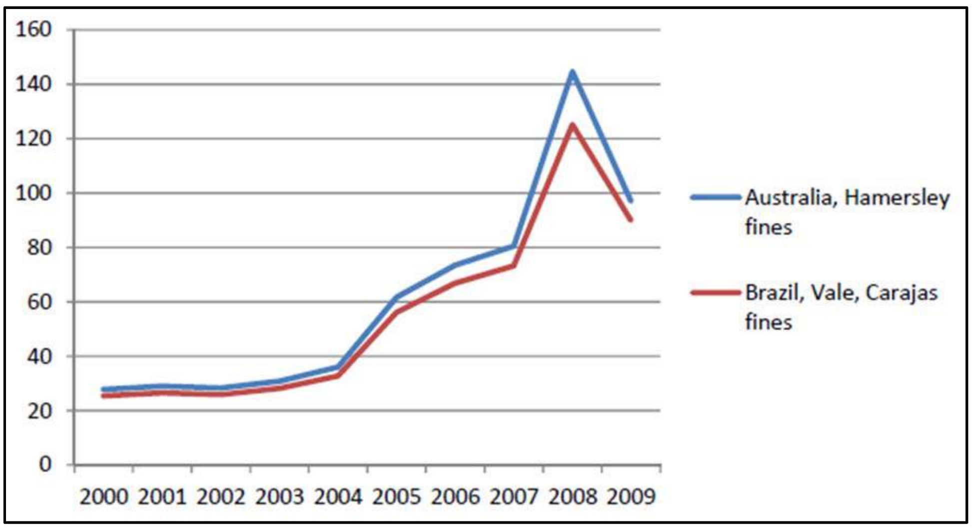
Source: SSY.

19.3.5 Iron Ore Prices and Pricing

Until 2009, iron ore prices were set under a benchmark system, where annual prices under long term contracts were set by the large buyers and sellers during a short negotiation period. This was then used as a standard, i.e. a benchmark, for negotiation between other actors in the iron ore market where the individual ore products were priced in relation to the benchmark product with premiums or penalties depending on quality. Price changes from year to year were relatively small and demand growth was slow at 1-2 % annually up to the year 2000. Variations in demand were accommodated by a system of off-take variations in long term contracts. Only very small volumes were traded on the spot market.

As iron ore demand began to grow more rapidly and the seaborne trade was reoriented, with China assuming an increasingly important role, the pricing system came under strain. Benchmark prices rose, driven by growing demand, and volatility became more important, helped by a rise in freight rates (see Figure 19.12). In addition, spot pricing became more common, since it was the preferred pricing method for many Chinese steel mills.

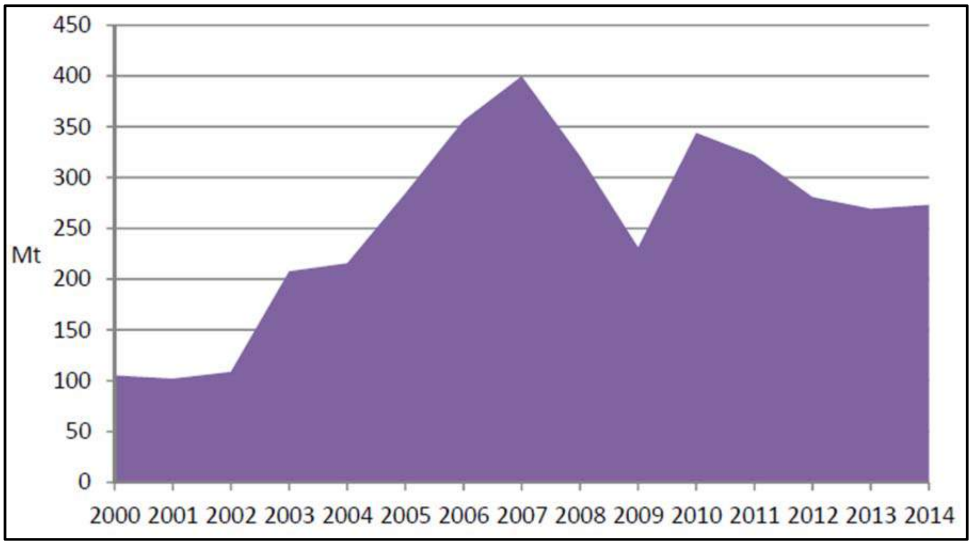
Figure 19.12 – Benchmark Prices for Iron Ore 2000-2009, USD per Dry Metric Tonne, FOB



Source: UNCTAD Iron Ore Statistics 2010, recalculated to US$ per dry metric tonne

The surge in Chinese demand allowed all producers to raise output, but the most dramatic increase took place in China itself, where many small mines exploiting low grade deposits opened in the first years of the twenty-first century. These mines benefitted from their closeness to steel mills and were sheltered by the very high shipping costs that raised the price of competing, imported ore. For these reasons they were viable in spite of very high operating costs. As a result, the industry cost curve came to resemble the famous hockey club shape familiar from graphs of global warming, with a large portion of Chinese producers bunching together at the far right of the curve with significantly higher costs than the rest of the industry. Figure 19.13 shows the rapid growth in Chinese iron ore production. It also shows that Chinese miners, with their high costs, bore a major share of the adjustment when demand and prices plunged in 2008-2009. After prices recovered in 2010 and 2011, production rose again, but it has declined since, the victim of lower prices and rising costs, particularly as ore grades continue declining.

Figure 19.13 – Iron Ore Production in China 2000-2014, Mt, Reported Gross Figures Converted to 63 % Fe



Source: UNCTAD Iron Ore Trust Fund

During the 2007-2009 period, rapidly increasing spot prices and the role played by Chinese iron ore producers who rapidly entered and left the market depending on prices created tensions in the benchmark system. Finally, the difficulty of ensuring market discipline resulted in the breakdown of the benchmark system in 2009-2010.

Currently, annually negotiated contracts are practically no longer used. Of the largest producers, Vale and Rio Tinto apply quarterly determined prices to most of their sales, with the price being set on the basis of spot prices during the preceding months. There have, however, been deviations from this practice, particularly in instances when spot prices have fallen rapidly, as in late 2011, and buyers have insisted that the new lower prices be applied. BHP Billiton has been reported to sell most of its production on a monthly basis and a sizeable proportion on the spot market. Other, smaller producers price on a monthly basis, sometimes using indices as a reference, or sell spot. It should be emphasized, however, that the situation is extremely fluid and changes fast.

The spot market has grown in importance and now accounts for a sizeable proportion of deals. There is, however, widespread discontent with price volatility, and none of the pricing techniques used at present can deal with this. Annual contracts do provide a considerable measure of certainty, but if they are not observed, the certainty is illusory. More frequent price changes offer greater flexibility, but they mean that prices are less foreseeable. It is unlikely that any solution that satisfies a majority of market participants will be found until markets for iron ore swaps and futures become more liquid and more widely understood by the industry. At present there are several ways open to those who want to manage their price risk, including options, futures and swaps. Trading is however still relatively thin, although volumes are growing rapidly. At some point, however, price risk management, as practiced for other commodities, will become sufficiently widespread to offer foreseeability with respect to prices for those who value it, while still ensuring that prices respond to market events.

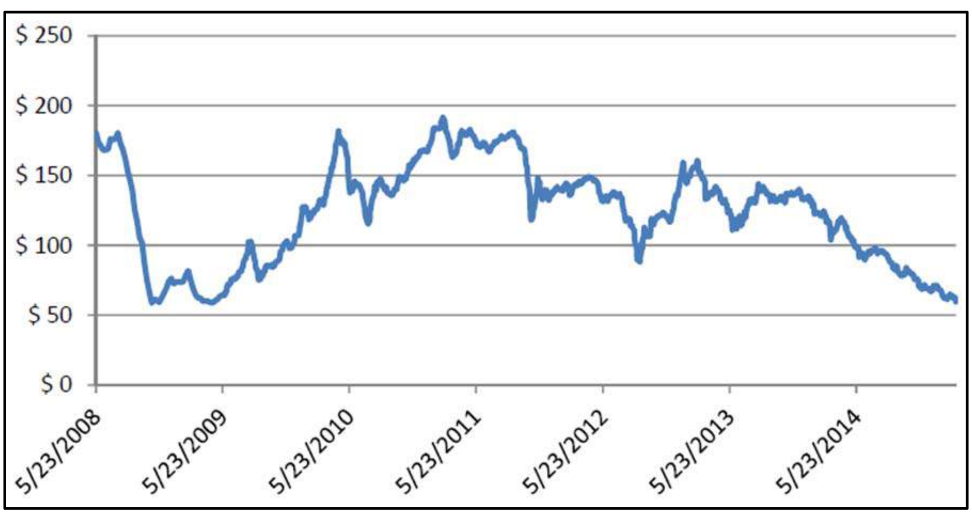
All pricing systems take into account differences in value in use (i.e. grades and other quality parameters), although the size of premiums and penalties, and their method of calculation, may differ. As has already been explained, different users, that is, different steel plants, have different quality requirements, depending not only on the type of equipment used, but also on the intended end product and on operational factors that may change over time. For instance, when steel demand is high, steel plants prioritize productivity and, other things equal, will prefer a high grade raw material that allows a higher metal output. Consequently, the premium paid for pellets, which have a higher iron content and are generally more consistent in composition, will be higher.

Nevertheless, the process of pricing has to start somewhere and a useful starting point is often the published price indices. They also provide the most relevant "average" quality references, although the value in use of a particular iron ore product will still vary from one steel plant to another.

The indices are published by Metal Bulletin (the Metal Bulletin Iron Ore Index or MBIOI) and The Steel Index (TSI). They differ slightly in how they are constructed but, by and large, reflect the markets for different grades. For instance, the MBIOI is published for ten different products and premiums while TSI reports prices for five different qualities.

The premium for higher grade ore (i.e. ~66 % Fe or above) is not necessarily proportionate to the premiums achieved by medium grade above low grade. The development of the Metal Bulletin index, medium grade (62 % Fe) since mid-2008 is seen in Figure 19.14.

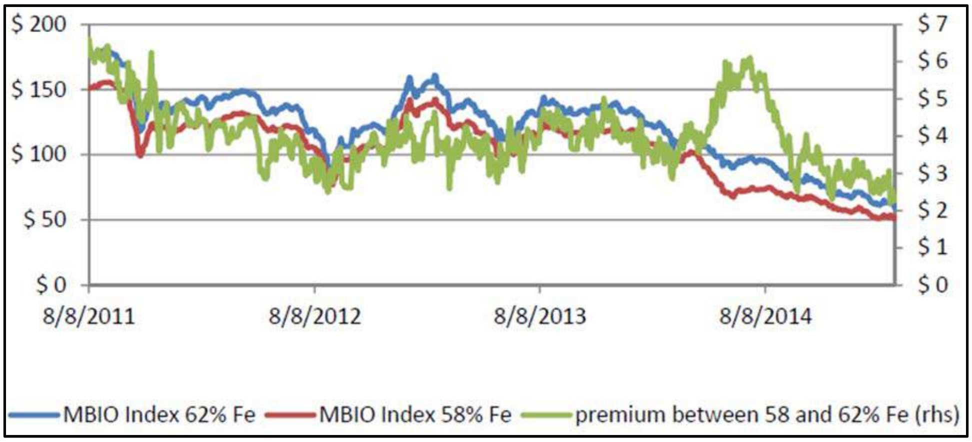
Figure 19.14 – Development of the Metal Bulletin Index (MBIOI 62 %), May 2008 – March 2015, USD per Dry Metric Tonne



Source: Metal Bulletin

The premium for higher grade ore over low grade one is shown in Figure 19.15. As shown, the premium has trended downwards but been fairly stable during the last couple of years.

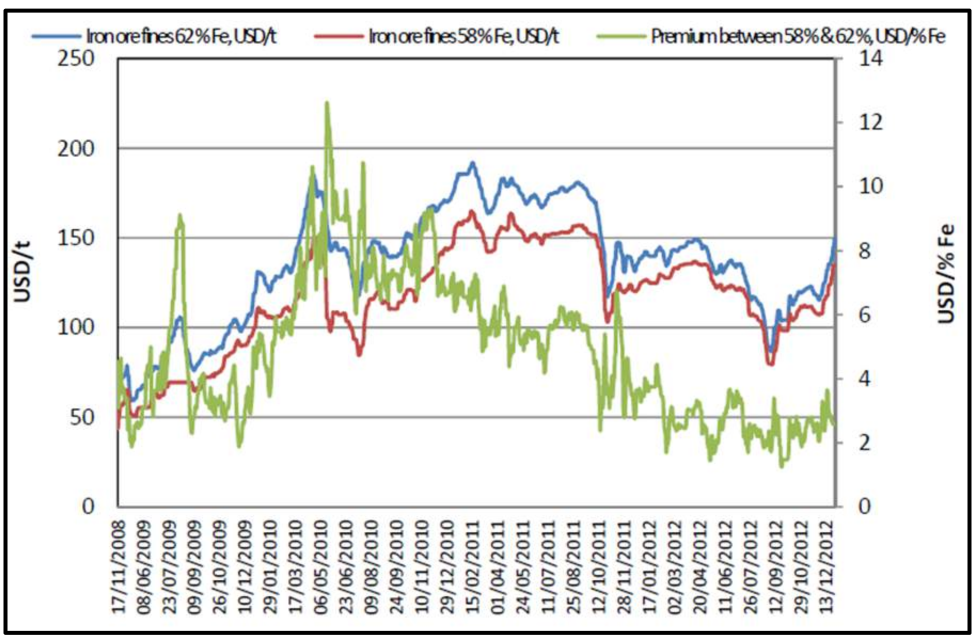
Figure 19.15 – Premium for MBIOI 62 % Fe Ore Over MBIOI 58 %, USD per % Fe CFR, August 2011 – March 2015



Source: Metal Bulletin

An alternative picture is shown in Figure 19.16. As seen from the figure, the premium trended downwards after mid-2010. It is believed that this is a consequence of the relative abundance of material, which has meant that steel producers do not have to prioritize high grade material in order to achieve maximum throughput in blast furnaces.

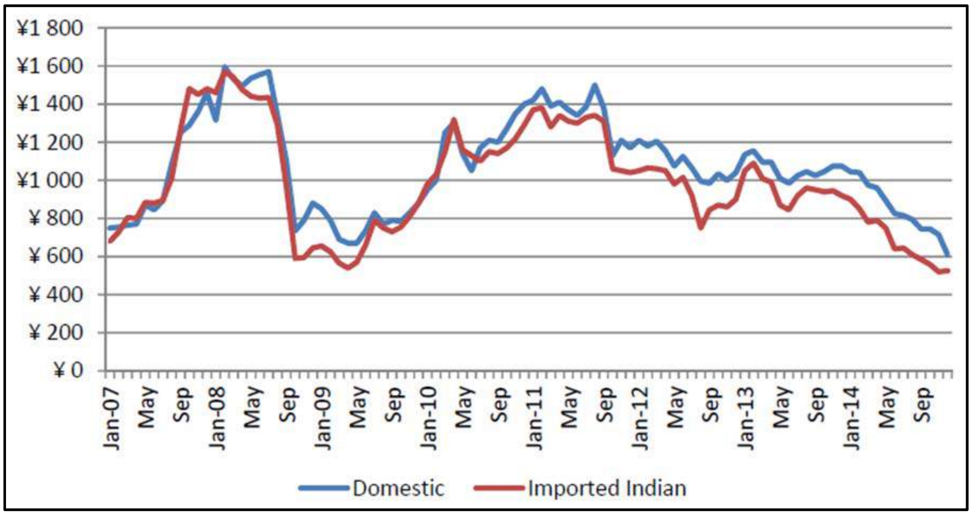
Figure 19.16 – Premium for TSI 62 % Fe Ore Over 58 %, USD per % Fe, 2008 – January 2013



Source: The Steel Index

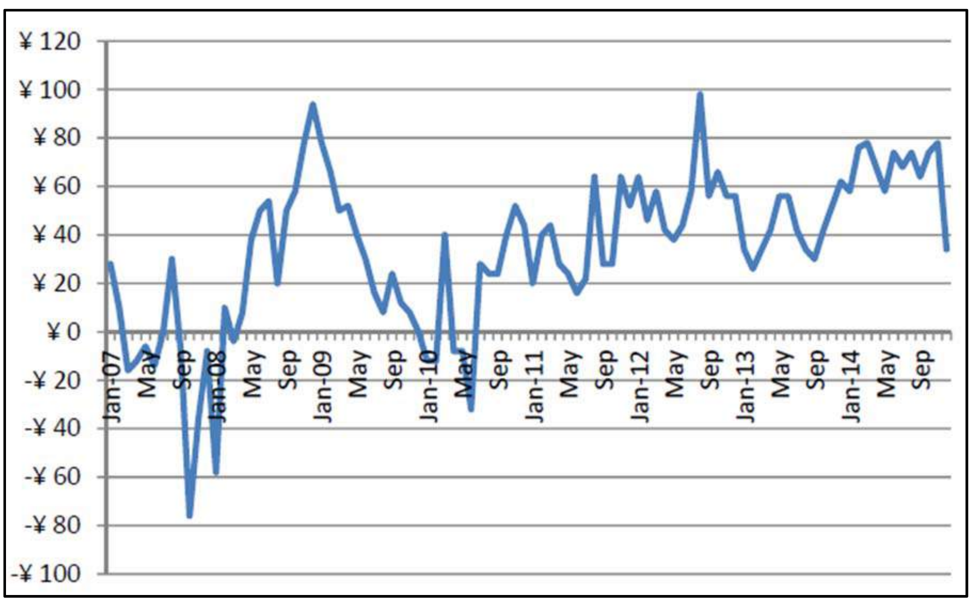
Data from an alternative source are also shown in Figure 19.17 and Figure 19.18. The latter figure shows the implied premium. As seen from the figure, there is no obvious similarity between that premium and the one shown in Figure 19.15 and Figure 19.16. Since the two price series shown in these figures refer to imported and domestic ore differences between them are likely to reflect other factors in addition to grade. The imported ore is mainly fines used in sintering, while the domestic one consists mainly of concentrate used as pellet feed. Imported concentrates and pellet feed with high Fe content is also used to correct silica levels in Chinese pellet plants.

Figure 19.17 – Iron Ore Prices in China, Domestic (66 % Fe) and Imported (Indian, 63.5 % Fe) Iron Ore, 2007 – Dec 2014, Yuan Renminbi per Dry Metric Tonne



Source: China Metallurgical Newsletter

Figure 19.18 – Premiums Between Domestic and Imported Ore, Yuan Renminbi per % Fe, 2007 – December 2014



Source: China Metallurgical Newsletter

One should be careful when trying to draw any conclusions from the evolution of the difference between the two (2) prices, that is, the premium for higher grade material. As seen from Figure 19.18, the premium has varied widely, sometimes turning into a discount.

It does appear, however, that the premium has been higher when the overall price level has been high, as in mid-2008 and the autumn of 2011, when steel works have presumably been anxious to acquire higher grade material to increase the throughput in their blast furnaces. Somewhat paradoxically, the premium has also been high in periods characterized by lower prices, as in the first half of 2009 and mid-2012. One interpretation of this could be that after the price has fallen rapidly, as it did in both cases immediately before the premium rose, domestic iron ore miners respond by closing down production and in the process of doing so they tend to overshoot and close down too much. However, if there is such a relationship, it is not statistically significant.

Other iron ore products such as lump and pellets generally command a premium over sinter fines. The size of this premium is not constant and has varied from 25 to 150 $/tonne in recent years.

19.3.6 Direct Reduced Iron

Direct reduced iron (DRI) is a secondary route to steel making. When producing direct reduced iron, a DRI feed is rolled into DRI pellets in the same way as blast furnace pellets. The DRI pellets are subsequently fed into a direct reduction furnace of some kind (the Midrex type shaft furnace is the most common) and the iron ore in the pellets is reduced by a reformed natural gas in a solid state reduction. At the output end of the direct reduction furnace, a 97 % (roughly) iron product, Direct Reduced Iron, DRI is produced. The remaining 3 % are unreduced iron oxides.

a) Demand for DRI

DRI is primarily (>90 %) used for the production of carbon steels by EAF-based steelmakers. The steel products that require higher-quality metallics are as follows:

- Sheet or coil
- Coil plate
- Line or transmission pipe, especially for severe weather or environment applications such as that found in northern Europe
- OCTG energy drilling pipe and tube, especially in more severe geologic conditions
- Engineered bar (primarily automotive and industrial applications)
- Engineered rod and wire (primarily automotive and industrial applications)

In regions where scrap is scarce and expensive such as the Middle East, North Africa, Mexico and parts of Southeast Asia, steel plants use almost 100 % DRI for their EAF charge, even for the production of rebar.

Integrated steel plants represent a secondary, but significant, user of DRI/HBI (Hot Briquetted Iron, a specialized product made of DRI). The two most important applications are:

(1) Blast furnace productivity enhancer to increase the volume output of hot metal,

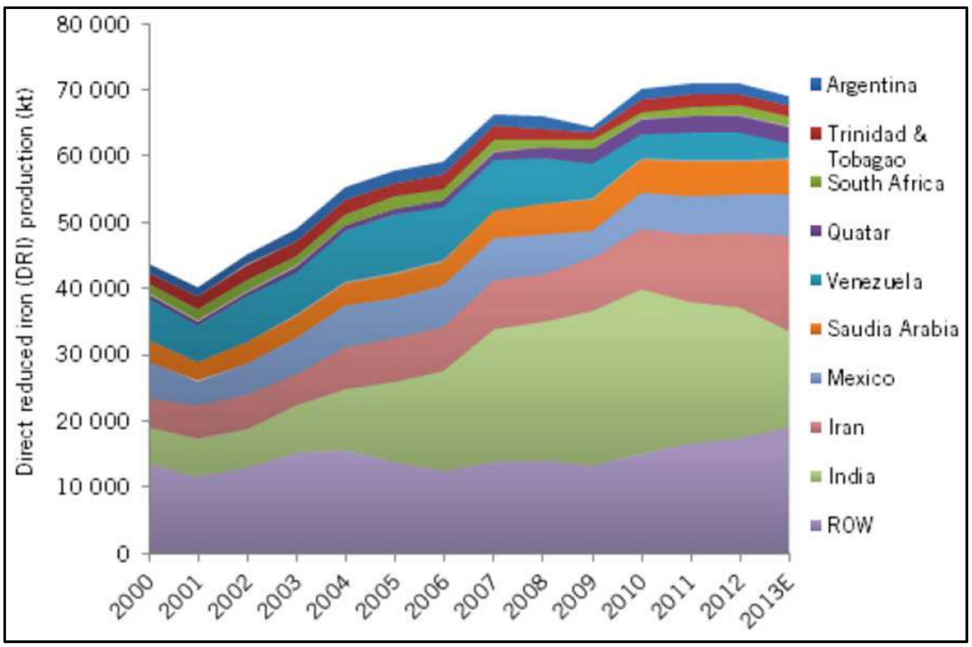
(2) BOF coolant – mainly as a cold charge (in place of scrap) and formerly (not as common any longer) as a post-tapping trim coolant for plants without Dynamic

Control cooling on their BOFs. There have been various points in the cycle since late 2003 where BOF steelmakers in the U.S., Europe and the Far East (China especially) have used greater amounts of DRI/HBI as a charge coolant. The primary reasons are related to changes in relative prices and lack of availability of higher quality scrap.

b) DRI Output

Global installed DRI production capacity reached a peak of 70.9 Mt in 2011 but production growth has decreased since then by -0.8 % (CAGR) in 2013 largely due to the fall in output from India the largest producer of DRI accounting for around 21 % in 2013. Typically DRI production is mainly located in countries where there is an availability of natural gas; however a significant proportion (73 % in 2010-11) of India's DRI production was from coal-based kilns with the remainder from gas based shaft furnaces. An outline of world production of DRI is found in Figure 19.19.

Figure 19.19 – Development of DRI Production World-Wide 2000 – 2013E



Source: World Steel, SNL Metals & Mining

India is the largest producer of DRI at over 25 Mt/ycapacity. In 2010-11, 17.06 Mt of DRI production was from coal based kilns and 6.2 Mt from gas based shaft furnaces. However, since 2010 DRI production has decreased partly due to ore shortages and economic slowdown. The country is shifting away from BF-BOF (blast furnace and basic oxygen furnace) to increasing use of electric steelmaking process (by arc and induction furnances).

Furthermore, the country is transitioning from DRI production from coal based kilns; however, it is expected that India will continue to develop alternative iron making technologies that do not require coking coal since it lacks sufficient domestic resources of such coal. India is unusual in its heavy reliance on small induction furnaces used to melt coal.

The major producers of DRI in India are:

- Essar Steel, DRI capacity 3,500 t
- Jindal steel and power, DRI capacity 3,220 t
- Ispat Industries, DRI capacity 1,800 t

19.4 Iron Ore Demand Forecast

In the following, we discuss the development of steel use, steel production and hence iron ore demand in different areas of the world. In doing so, we apply slightly different sets of reasoning for different areas. In particular, we use a more elaborate scenario for the development of Chinese steel use than for the rest of the world, where the main conclusions are drawn from historical developments and uncomplicated assumptions concerning the macro-economic outlook. The reasons for paying particular attention to Chinese steel use is first, that China accounts for almost half of global steel use; second, that while steel use in the rest of the world is not likely to change by large magnitudes, there are considerable differences of opinion with respect to future developments in China.

19.4.1 World Macro-Economic Outlook

The purpose here is to set out the basic assumptions underlying our forecasts for steel demand so that the reader can judge whether they constitute reasonably conservative starting points.

We see two crucial questions, each concerning one geographical area, influencing world growth prospects in the medium term:

- Will China manage its transition to a less investment and export driven economy, and will it do so in a manner that avoids a credit expansion and speculative bubbles and still maintain high growth rates?
- Will the United States return to growth or will fear of inflation strangle the recovery?
- Will the euro zone countries manage to avoid deflation and return to growth?

We believe that probable developments in the rest of the world are either consequences of what happens in these three places or too insignificant to be of substantial importance to the world economy.

With respect to the United States, we believe that growth will continue to be slow once the quantitative easing is phased out. An acceleration in growth would be possible, but is likely to be held back by low productivity increases and overly cautious fiscal policy, resulting from continued reluctance on the part of a large part of the economic policy establishment to consider more expansive policies. Accordingly, we anticipate a significantly lower growth trend than in the past for the United States. The impact on steel demand will, however, be less dramatic than could be expected, mainly because there is a need for major investment in infrastructure that will have to be accommodated, at least partly.

As far as Europe is concerned, Euro zone governments have chosen to try to escape the crisis mainly through budget austerity. The consequent depressing effect on demand will exert a heavy toll on growth over the next decade. European governments seem to be prepared to show a certain degree of solidarity and the extensive social safety nets in European countries will serve to contain the damage and cushion the impact on private consumption in the countries with relatively healthy public finances. From the point of view of steel production, the fact that investment is only slowly recovering to its pre-crisis share of GDP is a negative factor.

Economic developments in China are of course the overwhelmingly important variable determining iron ore demand, partly because China already accounts for more than half of world iron ore demand, partly because the rate of economic growth in China is expected to exceed that of most of the rest of the world, and finally because that growth is expected to continue being steel intensive, that is, each additional dollar of GDP will cause a relatively higher increase in steel use than would an additional dollar in, say, the United States.

We believe that China will be able to maintain reasonably high growth rates, although, for a number of reasons, economic growth will be slower than in the past. We believe, however, that scenarios involving a dramatic loss of momentum in China are unlikely to come true, mainly because China will still have a high rate of savings and so will be able to finance a high level of investment without running into any external constraints. We are also of the opinion that the vast increase in construction reflects genuine needs rather than only a speculative boom. The level of private debt in China is still relatively low, partly because banks do not allow households to borrow more than a low portion of the value of their houses or apartments. Nevertheless, because there are potential risk factors, including the level of lending by the "shadow banking system" and the indebtedness of lower level governments, we shall take the possibility of a severe fall in growth rates into account in the sensitivity analysis.

Our base case assumptions for China take as a starting point the simple observation that the very high growth rates in the earlier stages of the expansion of Chinese manufactures exports from a low base cannot be maintained for long. As long as Chinese exports replaced output from other producers, it was relatively easy to maintain high growth. At present, however, with China moving into markets where low wage costs are less of an advantage, and with Chinese labour costs increasing faster than in the rest of the world, the limits of growth for Chinese exports are increasingly set by the rate of growth of the world market itself, that is, by the growth of other countries' economies. Since almost all other countries grow slower than China, export growth rates will therefore have to come down.

In order to maintain economic growth at a higher rate than that of the surrounding world, albeit lower than in the recent past, Chinese growth will have to be reoriented towards private and public consumption rather than exports. One important reason is the need to continue the establishment of social safety nets for an ageing population. At the same time, the share of output going to investment will have to fall in order to make space for consumption, although investment in housing and domestic infrastructure will partly compensate for a fall in investment in export oriented infrastructure and capital equipment (the share of construction in investment already doubled from 2000 to 2010).

Such a reorientation, with an associated rapid increase in private consumption standards, would seem - maybe particularly in the eyes of the Chinese government itself - to be the only way that political stability can be assured. From announcements by political leaders and from the reforms undertaken it appears that this reorientation is already well under way. This does not necessarily mean that it will happen: similar announcements have been made before and the share of investment in GDP has continued increasing. It is true that this increase has partly been the result of stimulus spending to offset the fall in demand for Chinese goods resulting from the world recession, both in 2009-2011 and in 2013. Nevertheless, while it is generally agreed that Chinese growth will have to be reoriented towards consumption, Chinese leaders may not be able to take the necessary steps, in which case the reorientation will come about in an unplanned manner, when world demand proves insufficient to finance Chinese growth.

For our projections, we take as a starting point the need to reduce, in an orderly and controlled manner, the share of investment in Chinese GDP from its present level of about 55 % to a share that better resembles that of other countries at similar levels of income per capita (official statistics are generally considered to understate the share of consumption; the choice of 55 % for the share of investment takes that into account). We assume therefore that the share of investment in GDP will decline to 40 % in 2020 and to 30 % in 2030. We further assume that the decline will be smooth and gradual. While it would probably be more realistic to assume that the change will take place in fits and starts, influenced, among other things, by the rate and composition of growth in the rest of the world, we do not have any basis for making any more detailed assumptions about the exact timing of the change.

We also assume that the overall rate of growth in Chinese GDP will fall from its high present level to a more "normal" annual rate of 7 % from 2013 to 2015 (somewhat lower than the IMF's forecasts in its July 2014 update for 2014 and 2015, at 7.4 and 7.1 % respectively, but in line with the Chinese government's current five year plan) and that it will fall further to 6 % from 2015 to 2020 and to 4 % from 2020 to 2030. These assumptions may appear unduly pessimistic. But it has to be kept in mind that the most important source of Chinese growth has been the movement of labour from unproductive jobs in agriculture, particularly in the poorer central and western parts of the country, to much more productive jobs in manufacturing along the eastern seaboard. However, China is now running out of poor peasants to move. In addition, the total Chinese labour force is growing more slowly than in the past and will cease to grow around 2015. Although it is argued by some that growth in labour productivity can be maintained by removing discrimination of rural workers, who would then more easily move to higher paying occupations, and while there is some movement in this direction, this would only postpone the transition. Consequently, the main sources of productivity gains will be changes in the amount of capital used per employee and technological upgrades. Thus, while our forecast is intentionally conservative, we do not consider it improbable. According to the latest update from IMF, published in January 2015, GDP growth in China for 2014 reached 7.4 % and the forecast for 2015 and 2016 of 6.8 and 6.3 % are in line with our predictions above.

19.4.2 The Development of Steel Use in China

It is generally believed that steel intensity of use increases fast at low levels of income, and that the rate of increase slows as incomes increase and finally becomes negative. Table 19.4 shows the peaks in steel intensity in a number of countries. As seen from the table, the peak was in most cases reached at about 5,000 USD of GDP per capita (at Purchasing Power Parity, PPP), that is, roughly where China is now. Steel intensity at the peak varied, but was higher in Japan and South Korea than in Europe and North America.

Table 19.4 – Peaks in Steel Intensity of Use

	Year of Peak	GDP per Capita (USD at PPP)	Steel Intensity Kg/ 1000 USD of GDP
United States	1920	4594	87
OECD Europe	1960	4616	76
Canada	1965	7966	57
Japan	1970	5623	159
South Korea	1990	5398	100

Source: OECD: Recent Steel Market Developments, DSTI/SU/SC(2011)1, Paris, April 2011.

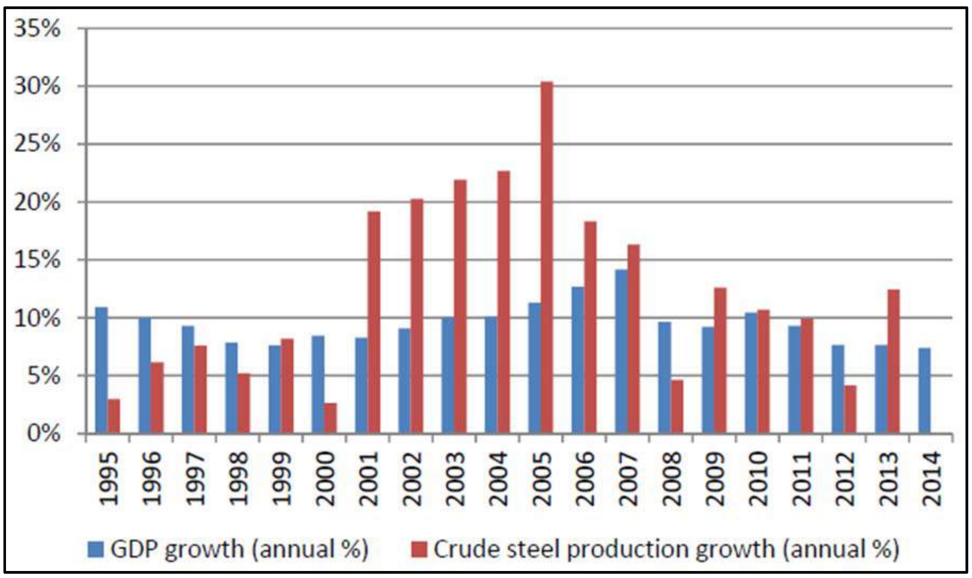
In 2013, steel intensity in China was at 80 kg per 1,000 $ of GDP, down from 88 kg in 2011. On this basis, it could be argued that China's steel intensity has peaked. There are, however, two reasons why it should not be expected to fall much further.

First, capital investment, which is generally more steel intensive than consumption, is very high in China, as it has to be in order to achieve double digit growth rates. As investment in infrastructure moves west towards the interior of the country, although we expect investment to fall relative to GDP, steel demand will continue to be high for a country at its income level, provided that GDP still grows rapidly. The only other reasonably large countries that have followed an investment and export oriented growth path similar to that of China are Japan and South Korea, both of which reached their peak steel intensity at levels much higher than that of China today, despite having lower ratios of investment to GDP. Moreover, Japan's and South Korea's periods of rapid growth were stopped by external shocks: in Japan's case by the first oil price shock in 1973/74, which reduced the demand from the rest of the world for Japanese exports, and in the case of South Korea by the Asian crisis in 1997, which led to credit tightening and planned contraction of overextended conglomerates. If China can avoid such external shocks, its period of high growth rates, high investment and consequently high steel intensity, could prove to last longer than did Japan's or South Korea's corresponding periods. Indeed, China appears to be in a better position than the two other countries with respect to its ability to avoid external shocks. It is much less dependent on the rest of the world than Japan or South Korea were at similar points in their expansion, because it is a large net creditor to the world and therefore has fewer constraints on expansion of domestic consumption. In addition, it is also less dependent on exports and has demonstrated its ability to continue growing even when its export markets are contracting. Therefore, China may be better able to continue growing at a high rate, with consequent high levels of investment and steel use, than Japan or South Korea were able to do.

Second, as China's competitiveness as an exporter of products such as machinery and vehicles improves, the steel content of its exports will rise more than its imports, thus contributing to a rising steel intensity, as traditionally measured.

The high growth rates in steel use that China has experienced in the past cannot continue indefinitely, however, and indeed, there are signs that the growth is slowing down. Figure 19.20 shows the rates of increase in Chinese GDP and steel production. We use steel production as a proxy for demand, since reliable demand data are difficult to obtain, and although China is a net steel exporter, its exports are small relative to its production. The figure appears to point to a slowing down of the rate of growth in steel production relative to GDP and thus to falling steel intensity.

Figure 19.20 – Annual Increase in Chinese GDP and in Crude Steel Production, 1995-2014, %



Sources: China National Bureau of Statistics, IMF World Economic Outlook January 2015, World Steel Association

At present, investment accounts for 75 % of Chinese demand for steel, while consumption (both public and private) and net exports make up the rest. We assume that, for the period 2014 to 2030, steel demand for investment and for the rest of the economy will increase at the same rate as the sectors themselves, that is, we assume that the steel intensity of investment, consumption and net exports all remain unchanged, although overall steel intensity will fall as a result of changes in the relative weight of these three elements of GDP. This is not a very strong assumption. The composition of investment is expected to change somewhat, with greater emphasis on housing construction, but that is not likely to have any dramatic effect on steel intensity. As far as consumption and exports are concerned, most factors argue for a higher steel intensity (growing domestic demand for household capital goods and improved competitiveness of Chinese exports in steel intensive export sectors such as machinery and vehicles). However, substitution effects, arising partly from the need to reduce energy consumption, work in the opposite direction, and we have assumed that the two forces cancel each other out. We have also assumed that steel intensity is equal in consumption and net exports, which means that the exact distribution of GDP between domestic consumption and exports does not affect steel demand. While the arguments put forward in the foregoing would appear to point to the steel content of net exports rising, maybe faster than consumption, we prefer to use the more conservative assumption of equal intensity. It should also be kept in mind that even with the slight slowdown in growth that has been seen in China in 2011-2013, industrial production is estimated to have grown at rates above 10 per cent year-on-year.

The assumptions concerning China are summarized in Table 19.5. For the years 2014 and 2015 we have assumed a very modest growth following the current outlook, however, considering the outlook described above we forecast a return to higher growth rates for steel post 2016 but considerably lower than the growth rates achieved during the last decade.

Table 19.5 – Summary of Projection Assumptions for China

	2014-2015	**2016-2020**	**2021-2030**
GDP Growth	7 %	6 %	4 %
Investment Share of GDP	55 % declining to 51 %	49 % declining to 41 %	39 % declining to 30 %
Share of Steel Use in Investment	75 % declining to 72 %	70 % declining to 63 %	61 % declining to 51 %
Annual Growth in Steel Use	1.2 %	3.2 %	2.9 %

19.4.3 Steel Use in the Rest of the World

With respect to the rest of the world, the assumptions are both less critical and less sensitive, since the probable margin of error is likely to be considerably lower, given the greater stability in growth rates and structure of GDP and in steel use. Our assumptions are conservative and the growth rates are generally lower than the ones achieved historically. With respect to growth in developed countries, we do not believe that higher growth rates can be justified under any likely scenario. Rates of growth in developing countries are intentionally conservative, but it appears prudent to make low assumptions over this very long time period.

We expect a very slow recovery in the OECD area, due to government fiscal restraint and deficit reduction which will likely dampen growth for the next decade at least. However, social spending will somewhat alleviate the impact of fiscal austerity and maintain growth within a narrow (low) range. Moreover, a difference in, for instance, European growth rates of a couple of percentage points would not make a dramatic difference to either the rest of the world economy or to world steel demand. In the Unites States, the size of the fiscal deficit is troubling but the need for infrastructure investment argues in favour of a certain, if limited, increase in steel demand.

The outlook for steel in North America has been depressed for quite a while. We assume that growth in steel use will be slow, averaging only 1 % from 2010 to 2020 with no growth thereafter. It is interesting to note that the US steel industry, which was earlier to a large extent scrap based, has started to switch to DRI as gas prices has dropped considerably, making the DRI route competitive. This has created an increased demand for virgin Fe units in the US.

European steel production declined by 0.2 % annually from 2000 to 2013. We think that the region will show no growth from now until 2030.

Over the past 20 years, Japan has experienced stagnating steel use. We believe the trend will continue, in spite of the need for reconstruction after the tsunami in 2011, with growth at 1 % per year until 2015. Recent growth in Japanese steel production has been directly driven by China's surging demand for steel, and in particular for high quality sheet for automotive and appliance applications. Japanese steelmakers have established numerous joint venture coil processing and distributing operations in China to maximize their share of participation in China's recent and future high-quality coil growth.

Eastern Europe and CIS can be expected to perform better than the rest of Europe and North America, partly as a result of growing commodity exports, with steel use growing due to the need to invest in infrastructure. However the Russian steel market which seemed to have recovered from a decade of uneven development is to suffer from the restrictions in trade following the developments in Ukraine. In the long run, the outlook is better than in most areas due to underlying low per capita consumption and availability of low cost raw materials and energy. We assume that steel use in the CIS countries will grow by on average 1.5 % per year until 2030 with growth slightly higher in the beginning (2 %) but slowing down after 2020.

China is no longer the only fast growing large country, and although the Indian economy is still considerably smaller than the Chinese, it is gradually becoming more important. At present, Indian GDP is generally assumed to grow at 5-7 % per year from 2013 to 2020 (the IMF forecast is for 5.8, 6.3 and 6.5 % for the years 2014, 2015 and 2016) and it may well grow at the same rate during the following 10-15 years. India's demographic situation is better suited for long term high growth than China's, and India's urban population, estimated at 35 % in 2020, is also increasing. However, due to India's different economic structure, with services accounting for 65 % of GDP while manufacturing and construction make up only 18 %, it is likely that economic growth in India will be less steel intensive than that in China. Nevertheless, India has committed a large part of its GDP growth to infrastructure investment – this is also a high priority of the new government - and therefore the current lower steel intensity should not be overemphasised. India will use a substantial amount of steel and iron ore in the upcoming years. Given the declared emphasis on infrastructure investment, the future growth rate is likely to be relatively high. However, steel production in India faces several bottlenecks and the country's tariffs deter imports. For this reason, we think it is unlikely that India will come close to double digit annual growth in steel use. We assume that Indian production will increase at 6 % per year 2015-2025 and by 5 % per year 2025-2030 and that the country will produce about 151 Mt/yof crude steel in 2025. Although this is a considerable increase on the 74 Mt reached in 2013, it remains considerably lower than Chinese steel production at comparable levels of income and economic development.

In South Korea steel production and use have also grown fast the past decades. However, we anticipate that the rate of growth will be more modest in the future, averaging 1.5 % per year to 2025. South Korea is already the most steel intensive of all countries.

Steel use in the Middle East is expected to grow rapidly. On average we assume that this region will grow by 5 % until 2020 and by 3 % thereafter.

Table 19.6 summarizes our assumptions concerning future steel use. While growth rates for individual countries are generally lower than in the past, world demand is still expected to grow at a significantly higher rate than during the last decades of the twentieth century, because poor countries with fast growing demand for steel make up an increasing share of total world demand.

Table 19.6 – Assumptions Concerning Steel Use 2013-2030, Mt and %

	Use Mt	Average Annual Growth, %	Use Mt	Average Annual Growth, %	Use Mt	Average Annual Growth, %	Use Mt	Average Annual Growth, %	Use Mt
	2013	**2013-2015**	**2015**	**2015-2020**	**2020**	**2020-2025**	**2025**	**2025-2030**	**2030**
China	700	1.2	717	3.2	839	2.9	968	2.9	1117
India	74	7	84	6	113	6	151	5	193
Europe	176	0	176	0	176	0	176	0	176
North America	129	1	132	1	138	0	138	0	138
Japan	65	1	67	0	67	0	67	0	67
C.I.S	59	2	61	2	68	1	71	1	75
South Korea	52	2	54	1	56	1	59	1	62
Other Asia	94	3	100	3	116	2	128	2	141
Central and South America	49	2	51	3	59	2	66	2	72
Middle East	57	5	63	5	80	3	93	3	108
Rest of world	26	1	27	2	29	5	37	5	48
Total World	**1481**	**1.7**	**1532**	**2.6**	**1741**	**2.3**	**1954**	**2.4**	**2197**

Source: World Steel Association (2013); SNL forecast (2015-2030)

Globally we forecast that steel production growth will slow to 2.3-2.4 % per year between 2020-2030 compared to 2.6 % annual growth between 2015 and 2020 and 7.9 % growth during the 2001-2007 period. By 2030 we estimate that global steel production will reach around 2.36 billion tonnes compared to 1.64 billion tonnes of output in 2014. We forecast that China's crude steel ouput will reach around 1.0 to 1.2 billion tonnes during 2025-2030. Despite our assumption of slower output growth, demand is still forecast to remain strong for the next decade primarily driven by China. Growth outside China is expected to be driven by India, South East Asia, and to a lesser extent, Latin America and the former Soviet Union states.

19.4.4 World Iron Ore Demand

In general, iron ore demand will grow at almost the same rate as demand for steel. However, in the particular case of China, because of the rapid expansion of Chinese steel production, scrap use did not keep up in the past (since scrap availability at any particular point in time is a function of earlier steel use, the rapid growth in steel use has meant that little scrap was available for recycling). As a result, the share of scrap used in world steel production has declined. However, we are now nearing the point at which goods that were produced during the first high growth years are being scrapped. Accordingly, we assume that scrap use in China will be a constant share of steel production from 2014 to 2020 and that it will increase thereafter. Therefore, while we have assumed that steel intensity remains constant in each sector in China, iron ore intensity declines. We choose to represent this by assuming that the rate of growth in iron ore demand generated by each of the sectors after 2020 will be one percentage point lower than the sector's growth. For the rest of the world, we assume that the share of scrap remains constant throughout the period.

Under these assumptions, world iron ore demand will reach 2,098 Mt in 2015, 2,345 Mt in 2020, 2,601 Mt in 2025 and 2,892 Mt in 2030. The average annual increase over the entire period 2013 to 2030 is 2.5 %, which is below the rate of growth achieved in the early 2000s, but higher than the growth rates in the 1980s and 1990s.

According to this projection, some 233 Mt of production would need to be added until 2020 and about 256 Mt more from 2020 to 2025. These are impressive figures, in spite of being derived from very conservative assumptions regarding overall world economic development. From 2000 to 2007, output rose by 765 Mt, or slightly more in terms of annual additions than what we are projecting for the period 2014 to 2020. It deserves to be noted, however, that 260 of those 765 Mt came from China. As will be explained in the following, it is very unlikely that the Chinese mining industry will be able to put up a similar performance during the next ten years. Accordingly, the additional production per year needed from the world except China during the period 2014 to 2020 is higher than it delivered from 2000 to 2007. During that earlier period, the rate of expansion gave rise to severe bottlenecks in terms of infrastructure investment, equipment deliveries and trained personnel, which illustrates that such a rate of expansion is not something easily accomplished. The additional production needed from 2020 to 2030 is again of a similar order of magnitude, 55 Mt per year, and it is by no means certain that it can be brought on stream easily and without delays.

a) Outlook to 2050

We have also prepared an outlook for the period 2030-2050. The assumptions underlying this outlook are summarized in Table 19.7.

Table 19.7 – Assumptions for Outlook 2030-2050

	2030-2040	2040-2050
China		
GDP Growth/Year	3 %	
Investment Share of GDP	Declining from 30 to 25 %	25 %
Consumption Share of GDP	Rising from 70 to 75 %	75 %
Annual Increase in Steel Use	2 %	2.5 %
Increasing Share of Scrap	2 Percentage Units Lower Increase in Iron Ore Demand	
Annual Increase in Iron Ore Demand	0.04 %	0.1 %
Rest of world		
Annual Increase in Iron Ore Demand	2 %	

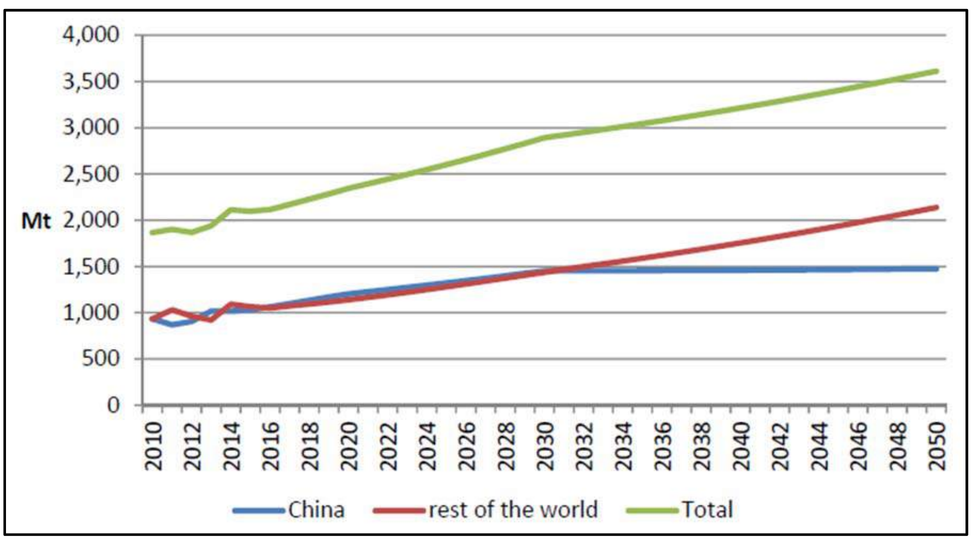
With these assumptions, we arrive at the forecasts shown in Table 19.8 and Figure 19.21.

Table 19.8 – Actual and Projected World Iron Ore Demand 2000-2050, Mt

	2000	2010	2015	2020	2025	2030	2040	2050
China	175	934	1031	1204	1323	1453	1460	1474
Rest of world	784	932	1066	1141	1278	1439	1754	2139
Total	**959**	**1866**	**2098**	**2345**	**2601**	**2892**	**3214**	**3613**

Almost 4 billion tonnes of iron ore in 2050 may sound like a lot, but one needs to remember that this means that demand takes 37 years to double from 2 to 4 billion tonnes. It took only 12 years to double from 1 to 2 billion tonnes.

Figure 19.21 – Actual and Projected World Iron Ore Demand 2000-2050, Mt



Source: SNL Metals & Mining

19.4.5 Iron Ore Supply

It is of course important to have an idea whether the world's iron ore industry will be able to deliver the needed ore and of where the additional production is going to come from, particularly since the location will affect both direct production costs and transport costs and therefore prices.

According to Citigroup, the "Big 4" (Vale, BHP Billiton, Rio Tinto and FMG) will add 229 Mt of capacity in 2014 and 2015. Other Australian and Brazilian producers are expected to add about 40 Mt. Projects elsewhere in the world may add another 20-40 Mt, for a total of about 290-310 Mt. Compared to the more or less stagnant demand that we expect to see, this clearly would point to a massive oversupply developing. For later periods, the number of confirmed projects is much lower and the large producers are reportedly reducing their planned capital expenditure.

In 2010-2013, high prices and encouraging demand prospects for iron ore made many marginal deposits highly interesting and a large number of new projects were announced. At the same time, however, it is also clear that many projects have not been developing as fast as expected and that significant delays have occurred. Many of the projects have been at least temporarily shelved. At the same time, those companies that want to go ahead with their plans have found that it has become considerably more difficult to raise capital.

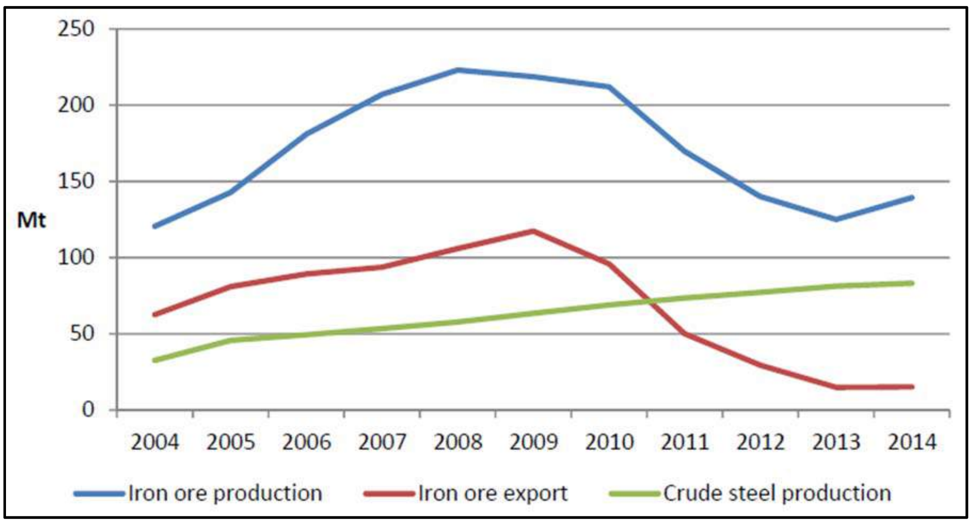
Despite the cancelling of many projects, the mismatch between planned capacity additions and expected demand will have to be resolved by some producers leaving the market. Before analysing the prospects for closures, however, it should be noted that the oversupply wold have developed much earlier had it not been for developments in India.

Indian iron ore exports have declined by almost 90 % from the top level in 2009 of 117 Mt to 15 Mt in 2013. As Figure 19.22 shows, production has declined as well, illustrating that the decline in exports is not only the result of rapid growth in the iron ore needs of the domestic steel industry, but also stems from an inability to maintain previous production levels. This inability is partly the result of events in the state of Karnataka, where several mining operations that were operating without having obtained valid licenses were closed by court decisions. Closures of mines in Goa that were in breach of environmental regulations also played a role. There are, however, also more general institutional problems and policy failures underlying the decline in production.

First, the introduction of export taxes on iron ore in 2011 has turned some previous exporters from export towards the domestic market, where they can obtain higher prices. During the late 2014, the tax was 30 % for lumps and fines and 5 % for pellets. This has the result of eliminating the incentives for other producers to increase production, either by expanding output within the limits set by existing capacity or by adding new capacity. Recent decisions to raise royalty rates from 10 to 15 % have added to the miners' problems. Thus, deposits that could have been developed if the domestic market had been less well supplied, remain unexploited. Since Indian steel producers have had privileged access to lump ore and high quality fines, they have also had little incentive to develop pellets as a raw material for blast furnaces. Consequently, not only are stocks of fines that could be used for pellets lying unused (it is estimated that at least 100 million tonnes of such fines are in storage), but neither iron ore mines nor steel mills have any reason to develop pelletizing. The situation has deteriorated even further recently, with some steel companies importing iron ore, in spite of the fact that imports are more expensive and attract a 2.5 % tariff.

In addition, even if the tax on exports were to be abolished, the iron ore industry in India faces formidable obstacles to expansion. The transport infrastructure is of low quality and overland transports are expensive. Building new railways is time consuming and subject to multiple bureaucratic obstacles. Rail freight rates were to be raised by 6.5 % in 2014. The permitting procedure for new mines is cumbersome and subject to political capture, making new investment risky. Planned legislative changes will, if anything, make matters worse. Foreign investment is surrounded by so many crippling regulations that international companies have almost given up.

Figure 19.22 – Indian Iron Ore Production, Iron Ore Exports and Crude Steel Production, Mt, 2004-2014



Sources: SNL Metals & Mining, World Steel Association

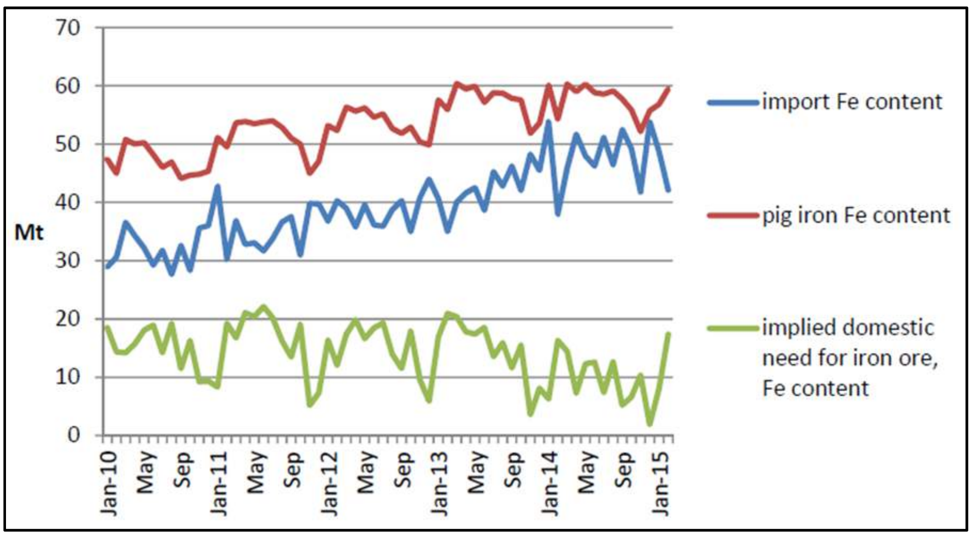
Against this background, it appears unlikely that India's iron ore exports will rebound, particularly in view of the expected rapid growth of India's steel production. Possibly, India might become a net importer of iron ore unless the regulatory problems are sorted out.

Chinese iron ore mines have assumed the role of swing producers and are likely to remain in this position. A large part of the Chinese iron ore industry had to shut down following the slower demand growth and lower prices in 2008. The mines used to be protected by high freight rates but as freight rates came down and production costs went up, only the unexpectedly strong recovery of the Chinese steel industry and the consequent high prices for iron ore have saved them. With the prospects for steel production in China now looking decidedly less positive, the time of reckoning has clearly come.

Chinese ore production fell from 344 Mt (gross run of mine production converted to comparable grades) in 2010 to 269 Mt in 2013. Thus, about 80 Mt of capacity had already disappeared at the beginning of 2014. Reports of further closures have surfaced in early 2015 and it remains to be seen how many mines stay closed after the cold months of January and February.

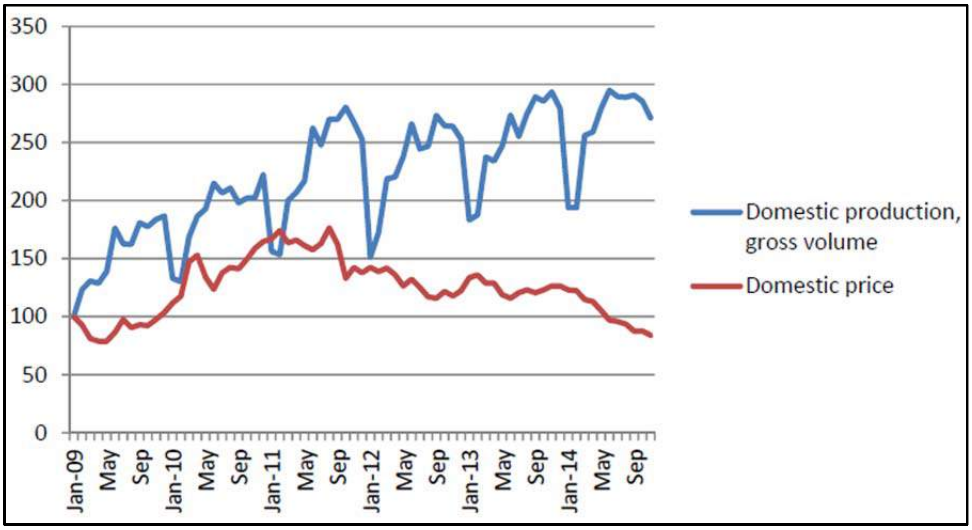
In Figure 19.23 we have calculated domestic Chinese iron ore production in terms of iron content as the difference between pig iron production and iron ore imports. As the figure shows, iron ore imports have risen both in absolute terms and as a share of supply, while domestic iron ore production has remained constant or declined somewhat. Figure 19.24 shows the development of gross run of mine production along with domestic prices. Clearly, grades are falling precipitously, which implies a loss of competiveness for the Chinese iron ore mines that are still producing.

Figure 19.23 – Chinese Iron Ore Imports, Pig Iron Production and Need for Domestic Ore (Pig Iron Production Minus Ore Imports), Fe Content, Mt/Month 2009-May 2014, Three-Monthly Moving Averages



Source: SNL calculations, based on World Steel Association and China Metallurgical Newsletter

Figure 19.24 – Iron Ore Prices (USD/Dry Tonne, 66 % Fe) and Production (Gross Tonnage) in China 2008 – November 2014 (January 2008 = 100)



Source: China Metallurgical Newsletter

There are at least four factors that have a larger impact on Chinese iron ore miners' costs than on those of their competitors:

- Wage inflation is faster in China than in most other countries; in recent years, annual nominal wage increases have been on the order of 15 %.
- The low grade of Chinese ore deposits makes their costs of exploitation disproportionately sensitive to increases in energy prices.
- The Chinese Yuan is likely to continue its appreciation against most major currencies, including the US dollar, while the Australian and Canadian dollars and the Brazilian real have already appreciated dramatically against the US dollar. Moreover, all of China's costs are Yuan denominated, while those of its competitors contain a significant dollar denominated element, in the form of, for instance, freight rates.
- Environmental mitigation investments that have already been carried out in other countries remain to be made in China and will weigh on production costs.

Finally, freight rates are expected to remain low for several years to come, which means that imported ore will be relatively competitive on the Chinese market. There is however suggestions from the Chinese authorities to lowering taxes on domestic iron ore production. How much difference this could do is debated as many argue that these tax cuts have already been administrated unofficially.

Accordingly, we believe that Chinese iron ore production will decline by a further 100-150 Mt to about 120-170 Mt in 2020. At the time of writing, in March 2015, iron ore prices have just fallen below 60 USD/dry tonne CFR China. In view of the very large additions to supply in the pipeline, prices are unlikely to rise more than marginally over the next couple of years. This means that the decline in Chinese production should take place during this year and next.

Other observers agree that Chinese iron ore production will fall substantially. Opinions vary concerning the magnitude of the fall. Most base their estimates on an assessment of how much of the Chinese capacity has costs over 80 USD per tonne, with numbers varying from 80 to 200 Mt. When spot prices hit 90-100 USD per tonne in late 2008, half of the Chinese iron ore industry operated at a loss, and was directly or indirectly subsidized. While some of that capacity has been replaced by new additions to capacity with higher productivity and lower costs, production costs in general have risen as already noted. Since trends in the cost of labour and domestically produced inputs are still rising, and since most observers agree that the Yuan is likely to appreciate further, our prediction of a 100-150 Mt decline may be somewhat conservative.

Even with the expected Chinese production decline, however, the next couple of years are likely to be characterized by lower prices than producers have become accustomed to over the past five (5) years. Hundred (100) USD per dry tonne is likely to become a ceiling rather than a floor for prices.

During the period 2015 to 2020, the market would be expected to rebalance at a higher price level. The reductions in capital expenditure by the large producers together with cancellations of projects by other producers will serve to eliminate the excess supply. However, a number of projects will be ready to come on stream at relatively short notice, so any price upturn is unlikely to be of more than short duration.

From 2020 to 2030, the need for capacity additions will remain and, given the experience of the complications associated with increasing capacity in Australia and Brazil at present, it is possible that these regions will find it difficult to grow production further. As a result, the 2020s may see the emergence of Africa as a major producing region. Many of the projects now being planned in Africa will not become fully operational until the early 2020s, but they will be followed by others. However the growth seen in the last couple of years in iron ore production from Africa have been stalled as most new producers are in administration or are struggling with financing.

19.5 The Market for Products from Lac Otelnuk

19.5.1 The Characteristics of the Lac Otelnuk Concentrate

As explained in the report by Lawrence Hooey of Swerea MEFOS, the Lac Otelnuk concentrate can be characterized as a high grade concentrate suitable for blast furnace or DR pellets. It has no drawbacks with respect to impurities.

19.5.2 The Chinese Market

The main market for Lac Otelnuk concentrate is likely to be in China, North America and the Middle East, roughly in that order.

Since we expect Chinese iron ore production to decline with large amounts of capacity closed over the next couple of years due to low prices, Chinese steel plants will need to look for new supplies. Much of the slack will clearly be taken up by Australian and Brazilian producers. However, most of the mines that close in China will be those mining low grade ores that have to be ground fine to produce pellets feed. Very little of the Australian capacity meets the requirements. Vale produces pellets feed and would probably be the main competitor for Lac Otelnuk on the Chinese market. Since the reduction in Chinese capacity is likely to be large, there should be room for a large part or all of Lac Otelnuk's production, however.

19.5.3 North America

In North America, Lac Otelnuk is less likely to be in a position to compete successfully on the market for blast furnace pellet feed. Most of the mining and pelletizing capacity is captive and enjoys important locational advantages. However, significant DRI capacity has been and will be added in North America, partly as a result of the dramatic fall in the price of natural gas. A new unit entered production in Louisiana at the end of 2013, and Midrex expects 5-6 Mt of DRI capacity to be added by the end of the decade (Midrex, 2013 World Direct Reduction Statistics). The new units will be in coastal locations which would be to Lac Otelnuk's advantage.

19.5.4 The MENA Region

Due to the abundance of cheap gas in the region, one of the most important sources of iron for steel in the MENA region is DRI. DRI production in the region has increased from 19.44 Mt in 2009 to 32.46 Mt in 2013. In 2013, 4.25 Mt of DRI capacity was added and a further 5.16 Mt will enter into operation this year. A number of additional plants are under construction and new projects are likely to be initiated as countries in the region see the opportunity to add value to their natural gas output.

The main exporters of iron ore to the MENA region are Brazil, accounting for 74 % of total imports, and Sweden, with 11 % (see Table 19.9).

Table 19.9 – Iron Ore Imports into the MENA Region in 2013 (kt)

Exporters	Importers							
	Bahrain	**Egypt**	**Libya**	**Oman**	**Qatar**	**Saudi Arabia**	**U.A.E.**	**Total**
Bahrain					1,300	1,250	200	**2,750**
Brazil	1,943	2,859	1,738	10,073	936	1,330	2,069	**20,948**
Canada		150			165			**315**
Chile	1,138							**1,138**
India					64			**64**
Iran	251							**251**
Norway	120							**120**
Sweden		1,132			1483	2,400	587	**5,602**
Total	**3,452**	**4,141**	**1,738**	**10,137**	**3,884**	**4,980**	**2,856**	**31,188**

Pellet plants have been built in Oman and Bahrain to cater for the DRI industry. The total capacity of these pellet plants is 20 Mt/y, 11 Mt/y in Bahrain and 9 Mt/y in Oman. Neither of the two countries has any iron ore mines, leaving them to import from the world market. The Oman pellet plant is owned and operated by Vale making Brazil the sole supplier. Most of the Bahrain plant's needs are also met by imports from Brazil.

The MENA market is in continual change given that it is the location of the vast majority of new DRI plant installations. For that reason it is an attractive market for Lac Otelnuk.

19.5.5 Conclusions

We believe that the Chinese market will be the main outlet for Lac Otelnuk concentrate, with some of it going to blast furnace pellets and some to DR pellets. North America and the Middle East are the other two possibly important destinations, where new DRI plants will provide good opportunities for high grade Lac Otelnuk concentrate. The European market will in all likelihood be of only marginal importance. As for prices, we believe that it is appropriate to assume that Lac Otelnuk concentrate will attract a premium.

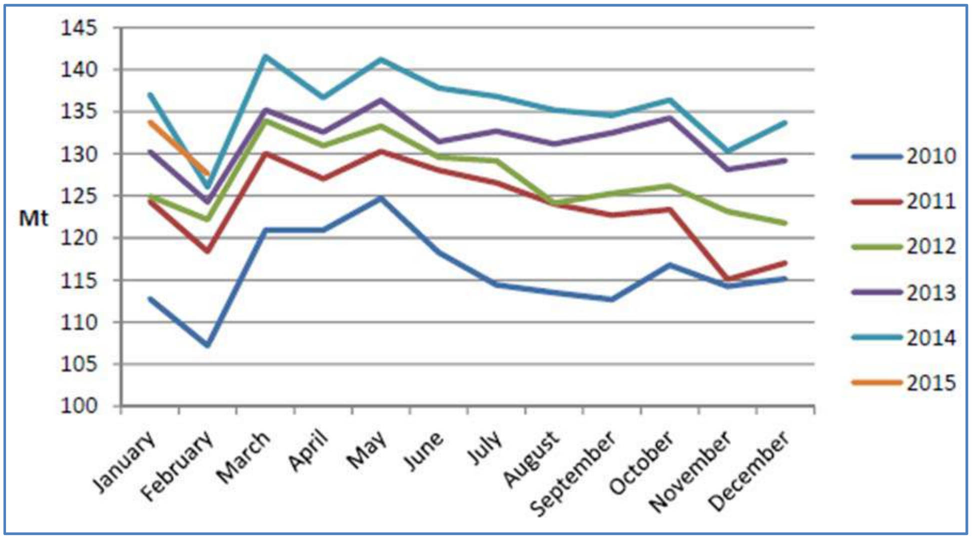
19.6 Iron Ore Price Forecast

19.6.1 Price Outlook for the Period Until 2050

Our intention in this forecast is to give a conservative long term view that is not overly influenced by the current situation. We give our forecast as a range: low/high.

Over the next few years, the world iron ore market will continue to be characterized by oversupply. As seen from Figure 19.25, despite the recent slowdown, world crude steel production is running around the same level or slightly above that of last year. However, the large iron ore companies have succeeded better than expected with bringing their latest capacity additions on stream and a supply overhang has developed in 2014, resulting in prices falling by more than 47 % in 2014 and continued to fall by 23 % from early January to early March 2015. Present price levels, around 55 USD/t, are however unlikely to be sustained over the longer term since this would render uneconomic not just a large part of Chinese ore production but also several high cost mines elsewhere. Moreover, with current prices new investments into the iron ore mining industry is highly doubtful which could put some stress to the long term iron ore supply and thus bring around another period of price increases. On the other hand, prices will probably not increase by much over the present level for the next few years as many investments and expansions are already planned and committed. Also with the current price decreases miners has managed to cut costs in their producing mines. The extent of these cost cuts to the long term price is debated but could lower the long term price of around 10 USD/t if these cost cuts are savings that can be sustained over the long term instead of cuts in investments and other necessary expenditures to keep the production stable over time. These potential cost cuts have not been considered in the price forecasts below.

Figure 19.25 – Monthly World Crude Steel Production, 2010 – January 2015 Mt



Source: World Steel Association

This means that this downturn is likely to be longer than the one in August/September 2012, when the TSI 62 % quotation bottomed out at around 86 USD/t, only to rise quickly back to 145 USD/t at the end of the year.

Iron Ore Price Forecast

The forecast uses the MBIOI (Metal Bulletin Iron Ore Index) index for 62 % Fe iron ore delivered in China as a basis. The reason for this is the dominance of the Chinese market and its role as a price setter. MBIOI is one of the more widely used of the various indexes and quotations. The forecast is expressed in real US dollars, using 2014 as the base year.

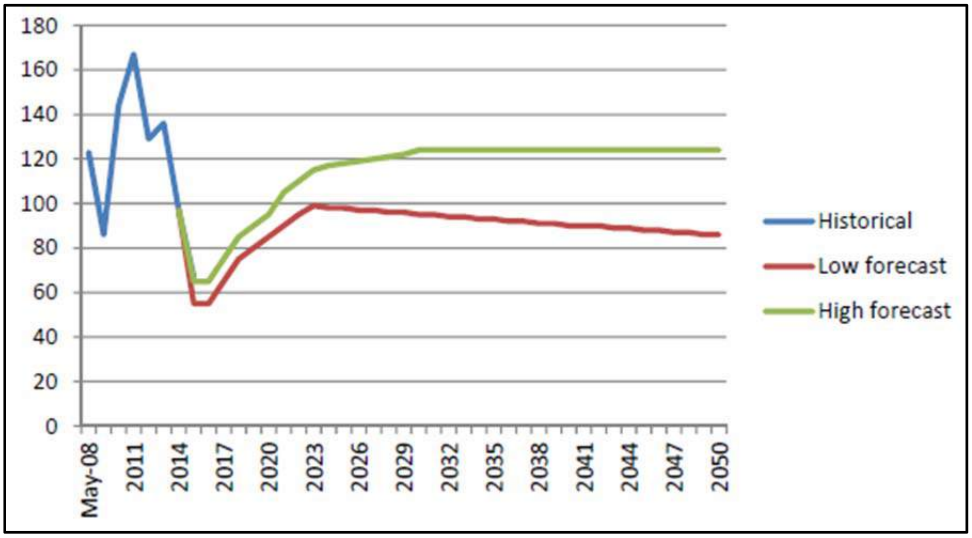
We believe that iron ore prices will evolve in accordance with the following four stage scenario:

1. Over the next three years, prices will be determined by the oversupply with a floor being set by the need to ensure that enough mines break even.
2. From about 2017 to 2020, the situation will resemble that of the 1980s and 1990s, when production increased in an orderly fashion and iron ore price movements were moderate. While price spikes may occur temporarily, particularly if producers encounter unexpected problems with capacity additions, prices will rise somewhat at the beginning of the period and then remain more or less constant.
3. From 2020 to 2030, prices will be set by costs in new projects and could be expected to decline gradually as a function of productivity improvements. However, increases in costs resulting from significantly higher energy prices and/or rising extraction costs due to the need to mine lower grade ore bodies could set a floor for this decline.

4. From 2030 to 2050, productivity improvements are expected to lead to continuing declines in prices, to some extent offset by rising extraction costs in new mines.

The forecast is illustrated in Figure 19.26 and Table 19.10.

Figure 19.26 – Prices for 62 % Fe Iron Ore Delivered in China, USD/Dry Tonne



Source: Metal Bulletin (historical prices), SNL forecast

Table 19.10 – Price Forecast, US$ per Dry Tonne, Basis MBIOI 62 %, CFR China

	Historical	Low Forecast	High Forecast
May-08	123		
2009	86		
2010	144		
2011	167		
2012	129		
2013	136		
2014	97	97	97
2015	67	55	65
2016		55	65
2017		65	75
2018		75	85
2019		80	90
2020		85	95
2021		90	105
2022		95	110
2023		99	115
2024		98	117
2025		98	118
2026		97	119
2027		97	120

	Historical	Low Forecast	High Forecast
2028		96	121
2029		96	122
2030		95	124
2031		95	124
2032		94	124
2033		94	124
2034		93	124
2035		93	124
2036		92	124
2037		92	124
2038		91	124
2039		91	124
2040		90	124
2041		90	124
2042		90	124
2043		89	124
2044		89	124
2045		88	124
2046		88	124
2047		87	124
2048		87	124
2049		86	124
2050		86	124

a) The Period 2014-2016

Over the next two years, prices will be determined by the oversupply with a floor being set by the need to ensure that sufficient mines break even. Supply will adapt gradually through the closure of high cost mines, particularly in China.

Using MBIOI as our benchmark, we believe that prices will recover very slightly from the present low and will reach an average between 55-65 USD late this year and in 2016. A floor for the price decline will be set by the cost in Chinese mines, where we believe that production capacity of 100-150 Mt has a cost of 85 USD/t or higher. This capacity will close down during the period.

b) The Period 2017-2020

We believe that this period will be characterized by a gentle price increase to 85-95 USD/dry tonne in 2020 as the supply overhang disappears and prices rise sufficiently to allow new investment. The reason for this is that a good part of the new mines will be located elsewhere than in Australia and Brazil and will have higher costs due to lower grades or large infrastructure investments. As a large portion of Chinese capacity closes, it will be gradually replaced in its role as swing producer by newer low grade mines in remote locations and therefore high costs. In our estimation, the cost curves for China and the rest of the world intersect at about 80 USD/t and this would consequently be the level at which the consolidation and shakeout of the Chinese iron ore mining industry is finally concluded.

c) The Period 2020-2030

For the period after 2020, prices must be high enough to both allow investment in new capacity to meet the growing demand and make it possible for most existing producers to survive. Since the major producers are unlikely to be able and/or willing to meet market needs by themselves, and since there are not enough other large low cost projects under way, much of that new capacity will be developed in what is now seen as marginal deposits and areas, from the point of ore characteristics or infrastructure costs.

In order to provide a conservative lower bound for the projected price range, we take as a starting point the 99 USD/dry tonne that we had already projected for 2023. Beyond that point, we assume that continued improvements in productivity will affect production costs and we postulate an annual price decline of 0.5 %. The high end of the range is set by the forced shift to lower grade, more remote deposits, which will require a higher price to enter into operation. We have chosen to reflect this by letting the higher limit rise at an annual rate of 1 %. We then arrive at a price span of 95-124 USD per dry tonne in 2030 (see Table 19.10).

d) The Period 2030-2050

We have no basis for projecting substantial changes in price trends during the final 20 years of the projection period. For the lower limit of the range, we assume that real prices continue declining by 0.5 % per year due to productivity improvements, and we freeze the upper limit at the 124 USD/dry tonne reached in 2030.

19.6.2 Shipping Costs

The development of freight rates is important since prices are equal for all in the main Chinese market, meaning that nearby suppliers enjoy an advantage. During the last decade, freight rates have varied dramatically, from lows of 5 USD/tonne to highs of 100 USD/tonne from Brazil to China. At present, they are influenced by a very large excess supply of shipping capacity, and although they have recovered from the very low levels reached in late 2008/early 2009, in 2014 they have again edged closer to those rates. We assume that there may be some upward movement as the world economy recovers and the excess supply of shipping capacity is eventually worked off, although that will take a few years, assuming that new orders do not increase significantly. However, we do not see a return to the extremely high freight rates of 2007-2008 and we expect rates to stay at levels not much above the present.

Accordingly, for the purpose of this report we have used as a starting point freight rates for Cape size vessels based on TC (Time Charter) rates of 25,000 USD per day and bunker fuel cost of 650 USD/t. Considering the fall in oil prices in late 2014 and early 2015 this might seem high, but considering the very long term forecast we feel it is a reasonable however slightly conservative price range. We have made the calculations for Alexandria, as a proxy for the MENA region, New Orleans, representing North America, and Qingdao, representing North China. We have carried out the calculations for all destinations. Our assumptions concerning shipping costs are shown in Table 19.11.

Table 19.11 – Shipping Costs (Vessel Capacity 150 000 DWT)

	Pointe Noire - Alexandria	Pointe Noire - New Orleans	Pointe Noire - Qingdao
Distance, nautical miles	4609	2700	11436
Speed, knots	13	13	13
Days for return trip	29.5	17.3	73.3
Time in loading port	1	1	1
Time in discharging port	4	4	4
Total return trip	34.5	22.3	78.3
T/C daily rate (US$)	25000	25000	25000
Total T/C (1000 US$)	863.6	557.7	1957.7
US$/tonne	5.76	3.72	13.05
Bunker costs (US$)	5.00	2.93	12.42
Port charges, US$/t	0.7	0.7	0.7
Total	**11.46**	**7.35**	**26.17**

19.6.3 Netback calculation for Lac Otelnuk

Our netback calculation for Lac Otelnuk is shown in Table 19.14. The shipping costs are those shown in the previous section.

While the basis for the netback calculation is obvious in the case of China as a destination, the MBIOI 62 % index being used as a basis for the forecast, a comparator has to be chosen for the other destinations. We have chosen to use Brazilian pellet feed shipped from Tubarao. By deducting the freight from Tubarao to Qingdao from the forecast value for MBIOI 62 % and adding the freight from Tubarao to Alexandria and New Orleans respectively, we obtain comparator values for these destinations. The calculations for shipping costs from Tubarao are shown in Table 19.12 and the calculations of the comparator price in Alexandria and Rotterdam are shown in Table 19.13.

Table 19.12 – Shipping Costs for Brazilian Pellet Feed from Tubarao

	Tubarao-Alexandria	Tubarao-Qingdao	Tubarao-New Orleans
Distance	5719	11092	4856
Speed	13	13	13
Days for return trip	36.7	71.1	31.1
Time in loading port	2	2	2
Time in discharging port	4	4	4
Total return trip	42.7	77.1	37.1
Total T/C (US$ 25000)	1066.5	1927.6	928.2
US$/tonne	5.93	10.71	5.16
Bunker costs	6.21	12.04	5.27
Port charges	0.7	0.7	0.7
Total	**12.83**	**23.44**	**11.13**

Table 19.13 – Calculation of Comparator Value in Alexandria and New Orleans

		2015	2020	2025	2030
Low, MBIOI 62 %	$/tonne CFR China	55	85	98	95
High, MBIOI 62 %	$/tonne CFR China	65	95	118	124
Freight costs	Tubarao-Alexandria, $/tonne wet	12.83	12.83	12.83	12.83
	Tubarao-New Orleans, $/tonne wet	11.13	11.13	11.13	11.13
	Tubarao-Qingdao, $/tonne wet	23.44	23.44	23.44	23.44
	Humidity	8 %	8 %	8 %	8 %
	Tubarao-Alexandria, $/tonne dry	13.95	13.95	13.95	13.95
	Tubarao-New Orleans, $/tonne dry	12.10	12.10	12.10	12.10
	Tubarao-Qingdao, $/tonne dry	25.48	25.48	25.48	25.48
Low price alternative	$/dry tonne CFR Alexandria	43.47	73.47	86.47	83.47
	$/dry tonne CFR New Orleans	41.62	71.62	84.62	81.62
High price alternative	$/dry tonne CFR Alexandria	53.47	83.47	106.47	112.47
	$/dry tonne CFR New Orleans	51.62	81.62	104.62	110.62

19.6.4 Price Discussion for Lac Otelnuk's Products

The netback calculation is carried out as follows:

1. The forecast price for TSI 62 % is used as a starting point.
2. The freight from the comparator, ore dry weight, is calculated for Alexandria and New Orleans to obtain comparator values.
3. The freight cost from Pointe Noire to the three (3) destinations is deducted from the TSI price in the case of Qingdao and from the comparator values for the two (2) other destinations.

4. The Fe grade premium is added to obtain the netback FOB Pointe Noire.
5. We have maintained freight rates at the same levels as in the forecast to 2030.

The price for any single iron ore product depends on many different factors of which the Fe content is the most important. However, there are a number of other elements that could result in a reduction of the price or in some cases a premium. The size distribution of the product is another factor determining the value of the product to the steel plants.

In addition to these general observations it is however necessary to understand that the value of a specific iron ore product also depends on the specifics of the particular blast furnace and sinter or pellet plant in which the ore will be used. And on top of that also which are the other ores being blended in the blast furnace burden. Finally any sale of iron ore is a negotiation between the buyer and seller and in the final analysis depending on how well the seller and buyer understands the metallurgy of the customer's blast furnace and the other feeds. A deleterious element for one steel plant might not be a problem for another depending on the blending of different iron ore products and different types of steel end products.

Given the complexity of the operating parameters of the blast furnace and the varying properties of the final steel product, it is difficult to forecast the exact effects of all the properties in any iron ore in advance. The Fe grade premium is based on historical values (see Section 19.3.5, particularly Figure 19.15, Figure 19.16 and Figure 19.18) and is set at 4 USD per Fe unit. This is somewhat higher than the present rate but since it is in line with historical values it is our long term estimate. Moreover, since the Lac Otelnuk concentrate is particularly free from impurities a premium of this size does not appear unjustified.

The following table calculates a theoretical price for the Lac Otelnuk products with the freight rates. Our approach is to try to make our price realistic but conservative.

Table 19.14 – Netback Calculation for Lac Otelnuk

	2015	2020	2025	2030	2040	2050
Low, MBIOI 62 %	55	85	98	95	90	86
High, MBIOI 62 %	65	95	118	124	124	124
Theoretical comparator values						
Low						
Alexandria	43.47	73.47	86.47	83.47	78.47	74.47
New Orleans	41.62	71.62	84.62	81.62	76.62	72.62
High						
Alexandria	53.47	83.47	106.47	112.47	112.47	112.47
New Orleans	51.62	81.62	104.62	110.62	110.62	110.62
Lac Otelnuk						
Freight Pointe Noire-Alexandria, $/tonne wet	11.46	11.46	11.46	11.46	11.46	11.46
Freight Pointe Noire-New Orleans, $/tonne wet	7.35	7.35	7.35	7.35	7.35	7.35
Freight Pointe Noire-Qingdao, $/tonne wet	26.17	26.17	26.17	26.17	26.17	26.17
Humidity	8 %	8 %	8 %	8 %	8 %	8 %
Freight Pointe Noire-Alexandria, $/tonne dry	12.45	12.45	12.45	12.45	12.45	12.45
Freight Pointe Noire-New Orleans, $/tonne dry	7.99	7.99	7.99	7.99	7.99	7.99
Freight Pointe Noire-Qingdao, $/tonne dry	28.45	28.45	28.45	28.45	28.45	28.45
Netback Lac Otelnuk						
Fe grade premium, $/dry tonne	26	26	26	26	26	26
Low price alternative, FOB Pointe Noire						
Pointe Noire-Alexandria	57.02	87.02	100.02	97.02	92.02	88.02
Pointe Noire-New Orleans	59.63	89.63	102.63	99.63	94.63	90.63
Pointe Noire-Qingdao	52.55	82.55	95.55	92.55	87.55	83.55
High price alternative, FOB Pointe Noire						
Pointe Noire-Alexandria	67.02	97.02	120.02	126.02	126.01	126.01
Pointe Noire-New Orleans	69.63	99.63	122.63	128.63	128.63	128.63
Pointe Noire-Qingdao	62.55	92.55	115.55	121.55	121.55	121.55

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section provides an overview of the environmental legislation and guidelines applicable to the Otelnuk Iron Ore Project, summarizes the Project permitting process including closure, identifies potential social and environmental impacts based on available environmental information, as well as permitting and social or community factors for the four major aspects of the project:

- Mine and concentrator, including the main access road;
- Product delivery system ("PDS");
- Power transmission line;
- Product dewatering and storage facilities at the Port of Sept-Îles.

20.1 Permitting

Environmental permitting for the Lac Otelnuk Iron Ore Project falls under both the Quebec provincial and Canadian federal regulatory frameworks. The Project will require Environmental authorizations, as well as other permit applications for its construction and operation, from both provincial and federal authorities.

20.1.1 Regulatory Framework

The following presents the key aspects of the two regulatory frameworks (provincial and federal) applicable to the Lac Otelnuk Iron Ore Project.

In Quebec, the Lac Otelnuk Iron Ore Project has components located in four regions that are subject to four specific environmental regimes:

- The Kativik regime: corresponds to the territory north of the 55th parallel, subject to the Kativik environmental regime for project approvals under the James Bay and Northern Quebec Agreement ("JBNQA") (Section 23) and under Chapter II of the Quebec Environmental Quality Act ("EQA");
- The James Bay Regime: corresponds to the territory south of the 55th parallel, subject to the James Bay environmental regime for project approval under the JBNQA (Section 22) and under Chapter II of the EQA;
- The Moinier Regime: corresponds to the Moinier region, subject to the Moinier environmental regime for project approval under the Quebec Regulation respecting the environmental impact assessment and review applicable to a part of the northeastern Quebec region as per the North Eastern Quebec Agreement ("NEQA");
- The South Quebec Regime: corresponds to the Southern Quebec territory, subject to the environmental regime for project authorization under Section 31 of the Quebec EQA.

At the federal level (Canada), the environmental approval of mining projects is governed by the Canadian Environmental Assessment Act, 2012 (CEAA 2012). This Act allows the environmental assessment process to focus on environmental aspects within federal jurisdiction, including:

- Fish and fish habitat;
- Species at risk;
- Other aquatic species;
- Migratory birds;
- Federal lands;
- Effects that impact Aboriginal peoples, such as their use of lands and resources for traditional purposes;
- Changes to the environment that are directly linked to or necessarily incidental to any federal decisions about a project.

Application of the Canadian Environmental Assessment Act ("CEAA") 2012 is independent of the four Quebec regulatory requirements for the four regions. Application of CEAA 2012 to mining projects is determined by the regulations under CEAA 2012, in particular the Regulations Designating Physical Activities which specify the "designated projects" that may require an environmental assessment by the federal agency. Under Paragraph 16 of this regulation, the mine and concentrator are subject to the CEAA 2012 as the mine will exceed 3 000 tpd production. Application of CEAA 2012 to other components of a mining project, aside from the mine and concentrator, depends on other triggers specified by the regulation and requires a case-by-case screening of the features of a given project against the triggers specified by the federal regulation. It is not anticipated that any of the four (4) project components will require a Panel Review under the CEAA 2012.

Each of the four above-mentioned environmental regimes for project approval has their own specific procedures and requirements for the scope and execution, when applicable, of Environmental Impact Assessments ("EIA"), and each one also has specific mechanisms for the evaluation of EIAs and for public hearings.

Table 20.1 presents the regimes that would be applicable for each of the specific component of the Lac Otelnuk Project.

Table 20.1 – Applicable EA Regime for Each Lac Otelnuk Project Component

Project Component	Applicable EA Regimes
Mine site and its infrastructure	• JBNQA – Section 23 • CEAA 2012
Product delivery system (PDS)	• JBNQA – Section 23 • NEQA • EQA
Transmission line	• JBNQA – Sections 22 • JBNQA – Sections 23 • NEQA

Port area	• EQA

Other Quebec provincial laws pertaining to the Lac Otelnuk Iron Mine Project include:

- Quebec Mining Act: Section 232 of the Quebec Mining Act governs the obligations and requirements for mine closure;
- Quebec Sustainable Development Act: This law does not create specific legal obligations for the mining project; however, it plays a role that cannot be ignored in the government review and approval process for the project.

Aside from the above, the Province of Quebec also has regulatory design requirements and guidelines applicable to the design of mining projects. These include, and are not limited to:

- Directive 019 regarding the mining industry, which specifies some of the requirements and/or criteria for the design of tailings management facilities (TMF), drainage networks at mine sites, water quality criteria for the discharge of water from TMFs, criteria for the management of overburden and waste rock, as well as other requirements for monitoring, reporting, etc.;
- Quebec Water Quality Criteria (2013) for the protection of surface waters;
- Dam safety regulation;
- Guidelines for specifications of culverts at water crossings;
- Regulations respecting land filling and incineration;
- Mine closure guidelines.

At the federal level (Canada), an amendment to Schedule 2 of the Metal Mining Effluent Regulations (MMER) may be required for draining water bodies or watercourses frequented by fish to create tailings disposal areas for the Lac Otelnuk mine site. An amendment to Schedule 2 of the MMER will not be granted unless it can be shown that there is no viable alternative to disposing the tailings in the water bodies or watercourses in question. A thorough evaluation of the alternatives will be conducted and will be included in the EIA. If required, a fish habitat compensation plan will need to be included in the EIA to provide a clear understanding of the effectiveness of the proposed compensation.

Aside from the regulatory requirements, another key item is the social acceptance of the mining project by First Nations communities. The project is located in territories where there are specific provisions under the JBNQA (Section 22 – James Bay Region, Section 23 - Kativik Region) and the NEQA to consult the Cree, Inuit and Naskapi as well as Innu communities in Sept-Îles.

20.1.2 Permitting Plan

Based on the Lac Otelnuk Iron Ore Project construction strategy, the four components will not all be built at the same time, with each project component being constructed following a specific construction strategy and schedule. Therefore, four (4) separate applications for environmental permitting approvals will be filed, one for each project component. This strategy has not been discussed with government authorities.

As environmental approvals are to be obtained under five (5) environmental regimes and procedures, for a given project component, each application will only be filed under the regimes and procedures that are applicable to the given component. In due course, a Project Notice for each project component will be filed with the appropriate authorities (Provincial Administrator of the JBQA, ministère du Développement durable, de l'Environnement et de la Lutte contre les changements climatiques, Canadian Environmental Assessment Agency).

The timelines for issuing EA guidelines vary according to the regulatory agency. Usually, the Government of Quebec is expected to issue generic guidelines within one (1) month of submitting the Project Description, but receipt of the guidelines for the components assessed under the JBNQA regime could take more time (several months).

Given the 365-day timeline for completion of EAs under CEAA, the guidelines from the Canadian Environmental Assessment Agency should be received within 60 days of the decision to require an EA (which would follow a regulatory 45-day period for the review of the Project Description) following receipt of a satisfactory Project Description. The mine site project component may require additional time (8 to 12 months) in the event the MMER is triggered.

20.2 Social and Community Requirements

The James Bay and Northern Quebec regions located between the 49th and 62nd parallels are more than 1 million km^2, and represent about two-thirds of Quebec's total land mass. Human presence in these regions dates back about 4,000 years with the Cree, Inuit, Naskapi, and Innus being the main First Nations communities in these areas.

Mining activities are ongoing in the study area, for example in Fermont (iron ore) and other extraction activities (sandpits, quarries, etc.) are also active. Numerous exploration activities (mining claims) are being conducted in the project area by various mining companies.

Hydroelectricity production is significant in the study area, from the North to South with the La Grande Complex, the Hart-Jaune hydroelectric power plant located on the Petit-Manicouagan Reservoir, the Sainte-Marguerite River Complex and the Toulnustouc River development.

Since the First Nation communities are an integral part of these territories, consultations and pro-active engagement are essential to support social acceptance of the proposed Otelnuk Iron ore Project, which, in turn, is essential to obtaining environmental approval. LOM is committed to a communication and consultations plan with these communities.

As the ESIA process progresses, consultation will be conducted according to a consultation plan and will provide for consultation sessions with the concerned communities and their representatives. Through these consultation sessions, concerns, expectations, territorial use and valued components will be defined. During these studies, additional consultation will be held as required to address concerns as they are identified. Before the completion of the studies, further consultations will present the preliminary findings in order to validate, the analysis of the baseline situation, the potential impacts identified and the proposed mitigation measures with communities and stakeholders.

LOM has planned to develop and eventually sign, memorandum of understandings (MOUs) and then Impact and Benefit Agreements (IBAs) with First Nations communities as the project advances.

20.2.1 Mine Site and its Infrastructure

a) Land Use

The proposed mine site and associated infrastructure is located in the Labrador trough iron range in Nunavik, Quebec, approximately 165 km northwest of the town of Schefferville, 225 km south of Kuujjuaq and 1 200 km from Montreal. The proposed mine site is located between the Caniapiscau River and the Swampy Bay River in the Caniapiscau watershed that flows north to Ungava Bay. The project is located in a region administered by the Kativik Regional Government ("KRG") administration and also within the territory under the JBNQA.

The closest inhabited areas to the mine site are:

- The Indian Reserve of Kawawachikamach, located about 155 km to the south-east and inhabited by the Naskapi Nation of Kawawachikamach;
- The town of Schefferville, located about 165 km to the south-east;
- The Indian Reserve of Matimekosh and Lac-John, located about 167 km to the south-east and inhabited by the Matimekush-Lac John Innu Nation;
- The Northern Village of Kuujjuaq, about 230 km to the north.

Aside from the Lac Otelnuk exploration work there has been no mining activity on the property or in the surrounding areas, therefore no mine workings, tailings impoundment areas, waste piles or other infrastructure are on or near the Property. The mine site is uninhabited and accessible only by air or snowmobile during the summer and winter, respectively. There are no protected areas within the proposed site. The Colline-Ondulées Provincial Park is located approximately 50 km to the southeast and there are two reserved lands for provincial parks that are within this 50 km radius: 1) Lac Cambrian territory located 35 km northwest, and 2) Canyon-Eaton territory located 25 km to the south. There is no intensive land use by the Kuujjuaq Inuit, Matimekush-Lac John Innus and the Kawawachikamach Naskapis on the proposed mine site based on currently available information.

b) Archeology

In 2012, Arkeos conducted an archaeological potential study of the mine site sector and identified 249 areas that could have been suitable for human activity in the past and may present some archaeological potential. Archeological inventories will be conducted on high (at least 3 areas identified) or medium (at least 18 areas identified) potential archeological sites identified before any of the planned development work begins.

The consultation plan for this project component will include information/consultation activities with Inuit, Naskapi and Innus communities respectively located in Kuujjuaq, Kawawachikamach Matimekosh and Lac-John as well as with Schefferville representatives.

c) Landscape

No landscape baseline study was conducted for this project component. It will be part of the EIA.

d) Consultation and Information

Adriana Resources Inc. signed a Letter of Intent ("LoI") in 2007 with Makivik Corporation ("Makivik"), the development corporation mandated to manage the heritage funds of the Inuit of Nunavik. The LoI opened communications with the Nayumivik Landholding Corporation of Kuujjuaq, the Northern Village of Kuujjuaq and the KRG. The Naskapi Nation of Kawawachikamach holds a seat on the Kativik Board. Nayumivik Land Holding Corporation is an affiliate of Makivik which holds title to the Inuit Lands. The general purpose of the LoI is to address issues that may be of interest or concern to all communities and stakeholders in the region affected by LOM's exploration and development work.

In 2012, LOM initiated a series of information and consultation sessions with representatives of the Inuit in Kuujjuaq, the Innus in Matimekosh-Lac John, and the Naskapis in Kawawachikamach. In the same year, information and consultation sessions were led by Golder in Kuujjuaq for institutional stakeholders and the local community.

e) Conclusion

WSP considers the social and economic baseline and consultation/information work conducted to date for the proposed mine site is adequate on which to develop and define the potential additional baseline studies and consultation/information activities that will be required for the EIA to meet the regulatory requirements.

20.2.2 Product Delivery System

a) Land Use

The proposed study area for product delivery system is an approximately 700 km long corridor extending from the mine site to the Port of Sept-Îles. This corridor is used for fishing and hunting by the First Nation communities (Cree, Naskapi, and possibly the Innu and Inuit). There are also numerous hunting routes used by the Naskapi within the proposed study area. There are also other mining and forestry activities, primarily located south of the 53rd parallel.

Sept-Îles, located on the north shore of the St. Laurence River and Fermont near the border with Newfoundland and Labrador, are the only communities of importance in the study area. There are numerous recreational camps located in the study area, especially along the Sainte-Marguerite River near Sept-Îles.

In terms of roads, Provincial road 389 links Fermont to Baie-Comeau and there exists a network of forestry roads that has been developed mainly in the southern portion of the study area.

A private railway, linking Port-Cartier to Fermont, is partially located in the study area. The QNS&LR Railway links Labrador City with Sept-Îles, however this railway passes through the study area for only a few kilometres near Sept-Îles.

The main airports in or near the study area are located in Sept-Îles, Fermont and Labrador City.

The PDS crosses three types of infrastructure: roads, rail way, and transmission power lines and their locations have been identified to a kilometre point (PK) along the proposed PDS route, where the PK 0 is the mine site location. Between PK 465 and PK 477, the PDS crosses Provincial Road 389, the Cartier Railway (CFC) and one Hydro-Québec 161 kV Transmission Power Line. From PK 742 to PK 749, the PDS crosses the Provincial Road 132, the access road to Pointe-Noire from Provincial Road 132 and numerous Hydro-Québec Transmission Power Lines. At the Port of Pointe-Noire, before reaching PK 754, the PDS may have to cross Regional roads and Medium voltage Power lines.

b) Archeology

Archeological sites (prehistoric, historic, modern, and contemporary) have been identified in the PDS corridor study area, primarily located along the watercourses that were natural transportation routes used by the First Nations communities. An archaeological potential study will be required as part of the EIA.

c) Landscape

No landscape baseline study was conducted for the PDS component and will be part of the EIA.

d) Consultation and Information

No consultation or information sessions have been conducted for the PDS component by LOM to date. The EIA consultation plan will include information/consultation activities with the Inuit (Kuujjuaq), Naskapi (Kawawachikamach) and Innus (Matimekosh-Lac-John, Takuaikan Uashat mak Mani-Utenam) communities and with Schefferville, Fermont, Labrador City/Wabush representatives.

e) Conclusion

WSP considers the social and economic baseline and consultation/information work is required for the EIA to meet the regulatory requirements.

20.2.3 Port Area and its Infratructure

a) Land Use

The Lac Otelnuk product dewatering, storage and loading facility is planned be located in a designated industrial area near the port of Sept-Îles on Pointe Noire. Recently, following the controversy with new development projects, especially mining projects, the local population of the area including Sept-Îles have increased their demands and requirements on consultation and commitments from proponents to gain social acceptability.

b) Archeology

No potential archeological site studies were conducted for this project component but archeological sites (prehistoric, historic, modern and contemporary) can be present in areas located close to the St. Lawrence River. An archeological potential site study will be conducted for this project component as part of the EIA.

c) Landscape

No landscape baseline study was conducted for this project component and will be part of the EIA.

d) Consultation and Information

No consultation or information sessions have been conducted for the product dewatering, storage and loading facility component by LOM, with the exception of meetings with key stakeholders (Sept-îles Port Authority, *Conseil régional des élus, Ville de Sept-îles, Ministère des Ressources naturelles*, Hydro-Quebec) involved in the site selection. The EIA consultation plan for this project component will, include information/consultation activities with the Innus from Takuaikan Uashat mak Mani-Utenam and with Sept-îles and surrounding regional representatives.

e) Conclusion

WSP considers the social and economic baseline and consultation/information work is required for the EIA to meet the regulatory requirements.

20.2.4 Power Transmission Line

a) Land Use

The proposed study area for the power transmission line is a 466 km long corridor to connect Hydro- Quebec's existing 735/315 kV substation at Tilly to a new 735/230 kV substation at Lac Otelnuk mine site. This area is used for fishing and hunting by the First Nation communities (Cree, Naskapi and possibly Inuit). At the time of producing the Hydro-Québec report for the 315 kV line Brisay-Nikamo-Tilly (Hydro-Québec, 1989), the study area between LG-4 and Brisay overlapped five Chisasibi Cree traplines. At the time, several other Cree families were also hunting and fishing in this region and are probably still do so today. Most camps used more or less regularly by Cree hunters and trappers are currently located along the access road from LG-4 to Caniapiscau. Camps are also found near Polaris, des Voeux and Hervé lakes, and close to the LA-2 Reservoir spillway. From Brisay to LOM mine site there are some hunting routes in the study area with most of them following the rivers. Within the study area, only one village (Keyano) is located near the Tilly substation at the west end of the study area.

Hydroelectric generation and transmission is the most important economic activity in the study area. There are four major hydroelectric power stations present: LG-4, Laforge 1 and 2 and Brisay, in addition to a major dam and spillway on the Caniapiscau River.

Since 1978, the area is served by a permanent road 275 km that connects LG-4 to Brisay. Westbound, this road connects to the Matagami – LG-2 road. Eastbound, the road joins the Caniapiscau reservoir spillway. A 34 km access road connects this road to the Laforge-1 generating station. A 7 km access road provides access to the Laforge-2 generating station. The roads on the east side of LG-4 are maintained during the summer but are closed in winter.

Airfields were constructed at LG-4, LA-1, LA-2 and Caniapiscau hydroelectric facilities during the construction period. With the exception of the LG-4 airstrip, the other airfields may not be operational but for the purpose of this assessment, they are considered as being active.

b) Archeology

Hundreds of archeological sites (prehistoric, historic, modern and contemporary) are known in the region (12 modern or contemporary sites are located along the Tilly-Nikamo line). The south-eastern part of the study area includes several large bodies of water that were the main transportation routes for First Nations people, and is an area of transition between various watersheds. This area may contain a large number of archeological sites from all eras. No potential archeological site studies were conducted in the transmission line corridor, these studies will be carried out as part of the EIA.

c) Landscape

The proposed study area corridor or the power transmission line is representative of the entire landscape of the plateau forming the centre of Quebec and has only a few exceptional sites including the Eaton Canyon on the Caniapiscau River. No landscape baseline study was conducted for this transmission line component and it will be part of the EIA.

d) Consultation and Information

No consultation or information sessions have been conducted for the Transmission lime component by LOM to date. The EIA consultation plan for this project component will, at a minimum, include information/consultation activities with Crees (Chisasibi, Mistissini) Inuit (Kuujjuaq), Naskapi (Kawawachikamach) and Innus (Matimekosh-Lac-John) organizations and with Schefferville representative organizations.

e) Conclusion

WSP considers the social and economic baseline and consultation/information work is required for the EIA to meet the regulatory requirements.

20.3 Environmental Studies

The four (4) project components are distributed in a large area in the province of Quebec. The following is a general description of the environment and the subsequent subsections summarize the known natural environment and the baseline studies completed to date. The environmental baseline is the necessary pre-requisite for the applications for environmental approvals, and provides the necessary input to support the EIAs. The list of baseline studies conducted to date is presented in the reference section of this report. Additional baseline work for each of the four project components will be completed as part of the EIA studies.

a) Physiography

The northern portion of the project study area is located in the Ungava Bay Basin natural region. This region is a high plateau inclined to the north. Continuing southwards, the study area crosses the North of Quebec Central Plateau natural region, a large hilly area with shallow lakes and peat bogs. The southern portion of the study area goes through the Central Laurentides natural region, characterized by a hydrographic network that is oriented north-south and that flows toward the St. Lawrence River.

b) Climate

From Sept-Îles to the mine site, the climate is sub-arctic, with average temperatures varying between -43 °C in February and 32 °C in July. Extreme total precipitation (rain and snow) varies between 8 mm in February to 114 mm in July. The maximum monthly average snowfall (from Schefferville and Kuujjuaq) is 57 cm.

c) Permafrost

Perennial frozen ground or permafrost is discontinuous and occurs in isolated areas. There was no permafrost patches identified in the Mine, Plant and TMF areas. The PDS corridor is characterized as having isolated patches of permafrost within the discontinuous permafrost zone, primarily associated with peaty wetlands, which insulate the subsurface. Overall, the continuity and the thickness of permafrost is increasing northward through Quebec-Labrador.

Permafrost in the vicinity of Schefferville, Labrador City, and Fermont ranges from less than 5 m to 50-100 m thick. A permafrost depth of about 120 m was observed in an area about 30 km north of Schefferville, and the ground temperature was reported as low as -2.8 °C at a 15 m depth. This area is located approximately 130 km south of the Lac Otelnuk mine site.

20.3.2 Mine Site and its Infrastructure

a) General Setting

The proposed mine site infrastructure for the 50 Mt/y iron ore concentrate will include the open pit, waste rock piles, ore stockpiles, water supply pond, concentrator, product delivery system, site access and haul roads, water distribution system, accommodations, power substations, fuel (diesel, gasoline, jet fuel) storage system, explosives storage and preparation plant, airport facility, and tailings management system and associated pipelines.

A subarctic forest climate characterizes the proposed mine and is part of the open boreal forest which is characterized by a coniferous forest, primarily black spruce stands on moss or lichens, combined with other species, including wetlands. The local climate is characterized by continental long and cold winters and short and mild summers. Located between the Caniapiscau River and the Swampy Bay River, the project site is situated in the Caniapiscau watershed where the water flows north to the Ungava Bay. Rivers and lakes are numerous on the project site, providing habitat for fish species. Wildlife and fish species are typical of the northern Quebec environment.

Golder Associates was retained by LOM to review the existing government reports, data bases and publications and to complete the various engineering and environmental studies in support of the EIA and Feasibility Study and for the mine site. In addition a baseline study was conducted by AECOM for the mine site access trail corridor from Schefferville to the proposed mine site. Table 20.2 summarizes the environmental baseline studies and fieldwork that has already been completed on the proposed mine site and site access trail. This environmental baseline information serves as the basis for preparing the EIA that will be required.

Table 20.2 – Summary of the Environmental Baseline Studies and Fieldwork Completed on the Proposed Mine Site and Site Access Trail

Environmental Sector	Baseline Field Work and Studies
Climate Data	• LOM has operated and maintained a meteorological station since July 2010 with Annual Climate reports prepared by Golder: o Annual reporting (years 2010, 2011, 2012, 2013) of recorded climatic data, Golder (2013); o Weather Report 2013, Golder (2013); o Weather Report 2011-2012, Golder (2013).
Physical Environment	• General description, Golder (2011); • Geomorphology - Field Report 2012, Golder (2013); • Baseline Study - Geomorphology and Soils - 2012 Fieldwork Report, Golder (2013).
Hydrogeology	• Ground Water Sampling at LOM site - Technical Memorandum - Fieldwork Result Compilation, Golder (2010); • Initial Hydrogeology Study, Golder (2012); • Complementary Hydrogeology Study – Summer 2012, Golder (2013).
Hydrology	• Hydrology Field Report - 2010 to 2012, Golder (2013).
Baseline Water Quality	• Water Quality Surface Water and Sediments Baseline – Fieldwork, Golder (2013).
Vegetation and Wetlands	• Vegetation and Wetlands - Baseline Study, Golder (2013).
Bathymetry	• Technical Memorandum - Release of Bathymetry Data, Golder (2013).
Fish Habitat	• Fish Habitat Baseline Study – Field Report 2011 and 2012, Golder (2013); • Fish Inventory Baseline Study – Field Report, Golder (2013).

Environmental Sector	Baseline Field Work and Studies
Wildlife	• Baseline Study – "Inventaire de la Sauvagine" - Field Report 2011, Golder (2013); • Baseline Study – Beavers - Field Report 2011, Golder (2013); • Baseline Report – Upland Breeding Birds, Amphibian and Semi- Aquatic Mammals Surveys Golder (2013); • Baseline Report – Aerial Survey of Ungulates, Golder (2013); • Baseline Study – "Avifaune Nicheuse" (French), Golder (2013).
Mine access trail corridor	• Baseline Study and Field Work Report, AECOM (2014).
Project Components Specifics	**Baseline Field Work and Studies**
Product delivery system	• SNC (2014). Field investigation report and mapping of superficial deposit in the PDS corridor
Power transmission line	• Preliminary cartography of vegetation and water courses in the power transmission line corridor; presented in Desfor (2014) report on access strategy for the 735 kV power transmission line
Product dewatering, storage and reclaiming facility	• The product dewatering, storage and reclaiming facility is to be located in a known designated industrial area near the port of Sept-Iles. All necessary baseline conditions are available for this area.
Main access road to mine site	• AECOM (2014): Baseline study and field work report

The following, is a summary of the environmental baseline studies conducted as of January 2015 and available public information review for the proposed mine site and associated infrastructure:

- A meteorological station has been in operation at the Lac Otelnuk exploration site since July 2010.
- The watercourses within the proposed mine site are located in the Caniapiscau River watershed which flows north up into the Ungava Bay. The project site is located at the edge of two sub-watersheds of which one flows through a series of lakes (Alpha Lake, Delta Lake, Lace Lake, du Gouffre Lake) then into the Caniapiscau River. The other flows into the Swampy Bay River, upstream of Otelnuk Lake and the Hautes Chutes then into the Caniapiscau River approximately 100 km upstream.
- The proposed mining infrastructure is located in a sporadic discontinuous permafrost area (surface cover between 10 and 50 %) according to the Canada Atlas of Natural Resources Canada (1993). However, no permafrost was observed during preliminary summer field work conducted in 2012 and 2013.

- The proposed mine site is located in the spruce-lichen bioclimatic domain within the boreal taiga subzone, which extends between the 52nd and 55th parallels (Saucier et al., 2003) in Quebec and is characterized by low density forest stands.
- No observations of plant species at risk were recorded within a 20 km radius from the project site. Two occurrences of special status plant species was observed during the summer of 2012 and 2013 rare plant surveys; the rock sedge (Carex petricosa var. Misandroides) and Nahanni oak fern (Gymnocarpium jessoense subsp. parvulum), two species likely to be designated at risk or vulnerable under the provincial Act Respecting Species at Risk or Vulnerable Species.
- According to public information, observation of a golden eagle (Aquila chrysaetos) was recorded near the project site. In addition, observations of the peregrine falcon (Falco peregrinus), the golden eagle, the harlequin duck (Histrionicus histrionicus), the bald eagle (Haliaeetus leucocephalus), and the short-eared owl (Asio flammeus) were recorded near the project site. If suitable habitats are present in the project site, these species could be present. Field investigations conducted in 2011, 2012 and 2013 for birds and six bird species at risk were identified within the project site: the rusty blackbird (Euphagus carolinus), the harlequin duck, the bald eagle, the golden eagle, the peregrine falcon, and the barrow's goldeneye (Bucephala islandica).
- No observations or records of mammals, amphibians or fish species at risk were recorded at the project site either in the database or during the field surveys conducted to date.

b) Protected Areas and Reserved Lands

There are no protected areas that are directly included in the project site. The Collines-Ondulées Provincial Park, located approximately 50 km southeast of the project site is the only legally protected area present within a 50 km radius of the project site.

In addition, there are two reserved lands for potential provincial parks that are located in this radius:

- The Lac-Cambrian territory, approximately 35 km northwest of the project site;
- The Canyon-Eaton territory, approximately 25 km south of the project site.

c) Access Trail

LOM commissioned AECOM to conduct and validate a mine site access trail study in July 2014. Ground truthing of a 180 km by approximately 4 m wide access trail corridor starting in Schefferville and running to the mine site revealed that a total of 26 % (47,792 m) of the area was forested, including commercially valuable stands, with 57 % (103,223 m) being open areas (wetlands, soil, sparsely treed areas, shrub lands and burnt areas) and with 79 stream crossings less than 5 m wide. Currently a 4 m wide access trail is being considered to minimize the impacts on the natural environment and fauna.

d) Conclusion

WSP considers the baseline environmental work conducted to date at the proposed mine site adequate on which to develop and define the potential additional environmental baseline studies that could be required for the EIA to meet the regulatory requirements.

20.3.3 Product Delivery System

a) General Setting

The proposed Lac Otelnuk mine site will be linked with the Port of Sept-Îles by a 755 km long PDS consisting of pipelines and pipeline bench in the northern part. The western limit of the PDS corridor follows an imaginary line linking the Caniapiscau River, the eastern borders of the Caniapiscau and Manic 5 Reservoirs and finally the Sept-Îles area. The eastern limit of the PDS corridor is defined by the Quebec and Newfoundland and Labrador border and the Moisie River. The proposed corridor represents an area of 43,700 km^2 of more or less pristine area, with the following protected and reserved lands:

- Protected area for the Canyon-Eaton national park along the Caniapiscau River near the proposed mine site (northern part of the proposed corridor);
- Proposed Biodiversity Reserve of Lac Gensart (approximately halfway of the proposed corridor);
- Proposed Aquatic Reserve of the Moisie River (adjacent to the Lac Gensart proposed reserve). This proposed reserve is centred on the Moisie River one of the most valued salmon rivers in Quebec. This protected area extends south-south-east on 328 km until the Sept-Îles area;
- Uapishka Biodiversity Reserve (near the Manic 5 Reservoir);
- Proposed Biodiversity Reserve of the Monts Groulx;
- Proposed Biodiversity Reserve of Lac Pasteur (in the southern part of the proposed corridor, in the Fauna Reserve of Port-Cartier-Sept-Îles).

These areas are currently protected by legislation. Approximately ten areas for the protection of wood caribou habitat are located between 50 km and 160 km north of Sept-Îles, and within the PDS proposed corridor. One zone of controlled exploitation of the fauna, called ZEC Matimek, extends from the Baie de Sept-Îles northward for 87 km. This zone is centred on the Sainte-Marguerite River, with an objective to manage the exploitation of the hunting and fishing activities on this territory.

b) Geology

Southwest of the Labrador Trough, the study area is characterized by tonalite and gneiss. It is covered by a dense network of orthogonal fractures. Deep glacial deposits, mainly Rogen moraines and drumlins are predominant. Fluvioglacial deposits (sand and gravel) are also present. The southern half of the study area goes through the Greenville Geologic Province. The bedrock is dominated by gneiss.

c) Hydrography

The hydrographic network, generally oriented north-south is well developed within the study area.

The Caniapiscau River originates in the Caniapiscau Reservoir and flows generally northward. This is the same for most of the watercourses in the LOM Project area. The density of lakes is high. At the latitude of the Caniapiscau Reservoir, the land is relatively flat. This area corresponds to the drainage divide area between watercourses flowing northward, westward, and southward. The border with Newfoundland and Labrador in this area corresponds to the watershed of the watercourses flowing eastward. The density of watercourses is high and is characterized by small lakes associated with the presence of Rogen moraine. South of the 53rd parallel, the drainage system is oriented north-south, taking advantage of major fractures in the bedrock. The rivers flow towards the St. Lawrence Estuary.

d) Vegetation and Wetlands

The northern portion of the study area is part of the Hudsonian Vegetation Region. This is the domain of the subarctic forest characterized by: balsam fir (*abies balsamea*), tamarack (*larix laricina*), white spruce (*picea glauca*), black spruce (*picea mariana*), paper birch (*betula papyrifera*), balsam poplar (*populus balsamifera*), and trembling poplar (*populus tremuloides*). In this area, trees can be found in sheltered areas and on thick deposits near lakes and rivers. Lichens and shrubs occupy the most exposed areas. Around the 53rd parallel, hills are forested except for the summits. To the south of the Hudsonian Vegetation Region, the forest is continuous and black spruce is dominant. Progressing southwards, the vegetation community transforms to that of the Laurentian Vegetation Region (Ref: Marie-Victorin, 1995) which is characterized by a mix of fir, spruce, and deciduous trees in the southern portion. Peat bogs are largely present in the entire study area.

e) Fish and Fish Habitat

There are two salmon rivers in the southern portion of the study area: the Au Rocher River near Port-Cartier and the Moisie River east of Sept-Îles.

f) Terrestrial Fauna

Caribou and moose are found in the area, but in low densities in the northern half, given the scarcity of deciduous trees. Woodland caribou, a protected species, is present in the southern half of the study area.

g) Geomorphology

A geomorphological analysis of the landforms was conducted in the selected corridor (4-5 km wide) to provide the type and quality of surficial deposits along the proposed PDS alignment and the distribution of granular sources within the corridor, in order to optimize the PDS alignment and to plan the construction activities.

In the absence of a detailed map of the surficial deposits, a geomorphological analysis based on the morphogenetic interpretation of the landforms from aerial photos was done for the entire corridor. Classes of surface materials and their thicknesses can be distinguished by mapping to identify terrains that can be favourable areas for the optimization of the passage of the PDS, the locations of intermediate pumping stations, the access road inside the same right of way and the borrow pits of granular materials, that will help building the pipe bench required for the construction of the PDS.

A field investigation confirmed the geomorphological mapping regarding the extent and depth of peat land, the thickness of the surficial deposits and the bedrock outcrops, the potential sources of granular materials, the water body crossings and the circumventions, and the surficial forms associated with cold climate conditions along the proposed PDS alignment.

h) Corridor Selection

A field investigation to map the superficial deposits in the proposed PDS corridor (SNC-Lavalin, 2014) was conducted to establish a corridor that considered the technical and environmental constraints and to confirm the individual watercourses to be crossed by the proposed PDS.

More than 30 simulations were completed by SNC-Lavalin to adjust the model and test various weightings of the technical, biological, and social constraining components and attractiveness components. These simulations helped identify the preferred preliminary alignments for the various segments of the corridor. Comparative analysis was then conducted for the most important segments, where the simulations identified possible options.

After more than 30 simulations that took into account the technical, biological, and social constraining components and attractiveness components, the final PDS corridor is approximately 4 km wide and represents the shortest path. The main characteristics of the corridor centre line include (Phases I and II):

- Total length of approximately 754.8 km;
- 7 % of the corridor is sloped >12 %;
- 174 watercourses of widths < 50 m, and 71 of widths >50 m;
- Located in the catchment basin of a major salmon river for 14 km;
- Crosses the wood caribou habitat protection zone for 18.7 km;
- It impacts the Matimek ZEC for 61.9 km;
- 26 % of its length is less than 25 km from an existing road.

i) Conclusion

WSP considers the corridor selection adequate on which to develop and define the additional environmental baseline studies for the PDS that will be required to meet the regulatory requirements for an EIA.

20.3.4 Power Transmission Line

a) General Setting

The Lac Otelnuk Iron Ore Project requires an estimated power supply of 1,049 MW for operation at full capacity. The proposed mining facilities will be linked by a 735 kV transmission line between the Hydro- Quebec's existing 735/315 kV substation at Tilly and a new 735/230 kV substation at Lac Otelnuk Iron Ore Project. The total length of the transmission line is approximately 466 km with 1,229 towers. The new substation transfers the power to two (2) new 230/34 kV substations, one for Phase 1 and the other for Phase 2, by means of two (2) 230 kV transmission lines with double circuits and approximately one (1) km long. The 735 kV overhead transmission line is approximately 466 km long and includes one (1) single circuit, one (1) overhead shield wire, and one (1) optical ground wire. LOM conducted a corridor selection exercise to establish a corridor that considered the technical and environmental constraints.

The proposed corridor for this project component extends from the Tilly substation near La Grande-4 (LG-4) hydropower plant on the west side to the LOM substation (distribution station) at the Lac Otelnuk Mine on the east side. From Tilly to Brisay, the project corridor northern limit is defined by the LG-4, Laforge-1 (LA-1), and Laforge-2 (LA-2). The southern limit is determined by a south-west axis, which includes the road from LG-4 to Caniapiscau. This southern limit was set in order to allow the development of a potential corridor near the LG-4 to the Caniapiscau road. The west area that extends from Tilly to the Brisay power plant has an existing 315 kV transmission line and an access road. The proposed corridor is bounded by the Caniapiscau Reservoir and encompasses the Caniapiscau River which flows northeast in this area before turning north-northwest near the proposed mine site. From Brisay to the mine site, the transmission line will be constructed in a more or less pristine area.

SNC-Lavalin reviewed government reports, databases, publications in order to prepare the basis for an EIA at the mine site area, including a preliminary mapping of the vegetation and watercourses in the power transmission line corridor presented (Desfor, 2014). No environmental baseline field work has been conducted for the power transmission line.

b) Hydrography

The hydrographic network is well developed within the proposed corridor. Major rivers such as La Grande River and Caniapiscau River have their headwaters in the proposed corridor. The density of lakes is important and the area is characterized by large reservoirs created as part of the James Bay hydroelectric development: LG-4, Caniapiscau, Laforge-1 and 2.

c) Vegetation

Wooded heath, black spruce and lichen make up the most common plant communities in the region. Alpine or arctic heath, which is restricted to the highest peaks, is characterized by black spruce which rarely exceeds three metres in height and cover less than 20 % of the surface.

d) Terrestrial and Semi-Aquatic Fauna

Within the proposed corridor, habitats suitable for terrestrial and semi-aquatic fauna are scarce and not very productive, which explains why fauna population densities are uniformly low for the entire area. This observation dates from the late 1980s, so the situation may have since changed. The proposed corridor has good potential for use by caribou. Moose are found in the area but in low densities given the scarcity of deciduous trees.

e) Protected Areas and Reserved Land

Two protected areas are partially located in the proposed corridor: the proposed Lake Sérigny Biodiversity Reserve and the land mass reserved for the creation of the Canyon-Eaton national park.

f) Geomorphology

Various aerial photographs taken between 1948 and 1987 (depending on the location) were used for the geomorphological interpretation of the proposed corridor. No field investigations were conducted to validate the photo interpretation.

Results of the geomorphological photo interpretation illustrates that the proposed transmission line's right-of-way will primarily cross surficial deposits consisting of till (77.2 %) as well as sand and gravel (8.5+1.6=10.1 %). Power line towers will be placed in locations with slightly more favourable deposits (i.e. featuring a higher percentage of till). A typical composition would include till (approximately 82.1 %), as well as sand and gravel (approximately 8.5+1.7=10.2 %).

g) Transmission Line Corridor Selection

A geomorphological analysis of the proposed transmission line corridor was conducted to support the siting of the towers and the overall engineering design. The approach was to avoid crossing over wherever possible and to minimize crossing-over spans elsewhere. Many lakes will be crossed and there are two major river crossings: one at the outflow of the Caniapiscau reservoir at the Brisay power plant, and the other at the Caniapiscau River about 8.5 km from the mine site. Substantial crossing spans are required at these locations. The current selected alignment crosses over an existing Hydro-Quebec 315 kV transmission line at approximately 5.6 km from the Tilly substation. In addition, since the final configuration of the transmission line coming out of the Tilly substation is unknown, another crossover may be required just outside the substation. Finally, the 735 kV transmission line passes between three existing telecommunication towers near the Brisay power plant. The current alignment provides approximately 250 m distance between the conductor and the telecommunication tower masts to avoid interference.

There are numerous risk-of-frost zones as well as high-risk-of-frost zones along the proposed transmission line corridor. The approach has been to avoid these zones, however in the vicinity of the mine site, the currently selected alignment crosses a high-risk-of-frost zone for about 4 km in order to reduce the transmission line length.

h) Conclusion

WSP considers the corridor selection adequate on which to develop and define the additional environmental baseline studies that will be required for the EIA to meet the regulatory requirements for the proposed Transmission line.

20.3.5 Port Area and its Infrastructure

a) General Setting

The Lac Otelnuk proposed product dewatering, storage and reclaiming facility is planned to be located in a designated industrial area near the port of Sept-Îles on Pointe Noire. LOM conducted a site selection study to locate the best technical and environmental alternative. This facility will include the following:

- Slurry receiving terminal station;
- Agitated slurry tank storage;
- Filtration dewatering using ceramic disk filters;
- Polishing and emergency discharge ponds;
- 3.2 Mt covered storage capacity.

Five (5) potential locations were evaluated using the following criteria:

- Area to accommodate full project capacity on one site;
- Favourable soil bearing capacities;
- Minimized earthworks and rock excavation;
- Avoidance of environmentally sensitive areas;
- Proximity to export wharf facilities;
- Slurry transport system length and head;
- Maximize distance from residential areas;
- Avoidance of utility corridors (power lines, highway, railway);
- Shortest distance to the effluent discharge point.

The site that best met LOM's criteria is located on private land and cannot be used by LOM until LOM acquires the land or the rights to use the land under a lease. For this reason it has been discarded for the purpose of the feasibility study. There are two other sites that meet all of LOM's criteria with the exception of one site requiring mitigation measures, both sites are located on Crown land and are available for industrial development. The selected site is located atop a rock outcrop and will require a significant amount of blasting and fill to establish the required infrastructure; however, it is relatively close to the existing wharf transfer tower. The proposed treated water discharge point is an intermittent creek near the dewatering plant that discharges to the St-Lawrence River on the west side of Pointe-Noire.

b) Conclusion

WSP considers that there are some baseline conditions of the terrestrial environment and air quality on which to develop and define the additional environmental baseline studies that will be required for the EIA to meet the regulatory requirements. Since the selected site is located on a rock outcrop, the environmental baseline studies will focus on the human, aquatic and marine environments.

20.4 Anticipated Environmental and Social Issues

The proposed Lac Otelnuk iron ore Project has four major components, with an expected project life of at least 30 years with infrastructure covering a significant area of both industrialized and pristine habitat. In the absence of the individual project component EIAs, the following is a summary of both the anticipated positive and negative global impacts in no particular order that can be identified from the currently available information. These impacts can be reduced or eliminated during the engineering phase of the project and the implementation of mitigation measures.

a) Potential Positive Impacts

- Increased job opportunities for the labour market through the hiring of large amount of employees;
- and awarding of construction and operations contracts;
- Contribution to the GDP of the various governments;
- Contribution to government revenues through royalties and taxes;
- Indirect and induced economic spinoffs locally, regionally and nationally;
- Provision of contracts, training and jobs to the local and regional Aboriginal groups;
- Numerous and important benefits through the IBAs with the First Nations communities;
- Transmission line infrastructure for future development of the region;
- Increased access to new areas for hunting fishing, trapping and gathering.

b) Potential Negative Impacts

- Vehicle accidents along the access trail;
- Loss of wetland habitats;
- Visual, noise and light impacts at both the mine site and port infrastructure;
- Increased Greenhouse gas emissions;
- Perceived negative public perception regarding the potential for spills at the mine site, along the PDS and at the Port Facilities;
- Negative public perception of potential impacts on the Atlantic salmon population by fuel and product spills;

- Draining and infilling of lakes for tailings impoundment at the mine site, resulting in the loss of fish and fish habitat;
- Transformation of sizeable areas of land for mining operations, transmission line and PDS, resulting in impacts on the natural, social and visual environments;
- Reduced enjoyment of hunting, fishing, trapping and gathering;
- Significant freshwater discharge from the port infrastructure into a marine environment;
- The crossing of numerous watercourses and water bodies for the construction of the product delivery system and transmission line;
- Intrusion of the product delivery system and transmission corridors into certain protected areas (either projected or existing);
- Potential disturbance of protected species;
- Negative public perception of the inter-basin transfer of a large volumes of water through the product delivery system and the annual discharge of ≈12 Mm^3 of treated water meeting discharge criteria into the St. Lawrence River;
- The perception of the Project's contribution to cumulative impacts, particularly in the Sept-Îles area (e.g., water quality; noise, light, air quality; visual environment; increased cost of living);
- Presence of a large work force in both the Schefferville and Sept-Îles regions that is predominantly male and from the outside;
- Increased pressure on the infrastructure and services in the Schefferville and Sept-Îles areas during the construction and operational phases.

c) Conclusion

Even if the main infrastructure of the project is significant in size, that it has a potentially long lifespan (from 30 to more than 100 years), that most of the project area is a pristine environment since no project or infrastructure is present and the natural environment is highly concentrated with watercourses and water bodies, the information available at this stage of the project development indicates that the benefits of this project exceed its disadvantages as none of the anticipated negative impacts should be considered significant once the appropriate mitigation and compensation measures are implemented. The project will also include monitoring, follow-up activities and adaptive management over its life span.

However, both desktop studies and initial and additional field investigations will be required to complete the baseline information for all the project components (mine site and infrastructure, PDS, Transmission Line, Port Area). In addition LOM will have to develop and implement a communication plan that will include all the affected stakeholders so as to manage these stakeholder expectations and to fulfill the public consultation requirements of the EIAs. Before each one of the project component descriptions can be submitted to the regulatory authorities, public consultations will be required. Some sections of the EIA may only be finalized once the EA guidelines have been issued by the governmental authorities.

20.5 Tailings, Waste and Water Management

By design, the Lac Otelnuk Iron Ore Project already integrates the engineering measures and/or equipment required to meet the applicable technical regulatory requirements for the tailings management facility, waste rock piles, site drainage, effluents, air emissions, waste disposal, etc. The following summarizes the proposed designs of these facilities.

The construction and operation of the proposed project will be carried out under comprehensive environmental management plans (EMP), developed as part of the scope of the EIAs.

20.5.1 Mine Site and its Infrastructure

a) Tailings

The proposed Lac Otelnuk Tailings Management Facility (TMF) will be located immediately to the northwest of the proposed mine site infrastructure and has a projected area of 45 to 50 km^2 to hold the 3,675 Mt (dry basis) of tailings expected over the 30-year life of the mine, as well as to store seasonal runoff and other water sources to provide the required process water to the concentrator. The central area of the proposed TMF includes a 5.1 km^2 lake with a 22.5 km perimeter as well as an average depth of 18 m and a maximum depth of 70 m. The TMF's location was selected to meet the following criteria: to provide the volume capacity required for tailings storage and water volume required to the concentrator under a 1:25 dry year, accessibility for equipment (construction, operation and closure), shortest distance and elevation from the concentrator for tailings transportation and availability of construction materials.

Representative samples of tailings and waste rock are considered to be non-acid generating with low metal leaching potential and meet the "low risk" classification of the Quebec Directive 019 regulations based on the following studies:

- Interim Report on Static Testing of Waste Rock and Tailings for the Lac Otelnuk Iron Ore Project, Golder (2014);
- Environmental Characterization of waste products from the Lac Otelnuk Deposit (SGS (2014).

The key design features of the TMF will include the water pond, the West Dam, lower perimeter dams including the North dam, Saddle dam and seven separator dams and the tailings pipelines that will be located on the south bank of the TMF.

Tailings deposition at 60 % solids content will start in the natural lake in the centre of the proposed TMF facility and continue for 3 years with deposition moving to the northwest. Floating barges equipped with pumps will provide the water volumes required for the concentrator. After the 3rd year, a water pond will be dedicated to water recycling. This water pond will progressively move to the north in the western end of the TMF as tailings are deposited. Mine water from the open pit, as well as surface water drainage from the waste rock piles, low grade ore piles and from the concentrator site, will all be directed to the TMF. Table 20.3 summarizes the expected tailings deposition (dry basis) over the 30-year expected mine life.

Table 20.3 – Expected Tailings Deposition (dry basis) over the 30-Year Expected Mine Life

Year	Cumulative Tailings Deposition (MT)
1	27.7
2	97.1
3	180.3
4	263.5
5	346.8
7	430
8	624.2
9	762.9
10	901.6
11-15	1,595
16-20	2,289
21-25	2,982
26-30	3,675

The final footprint of the tailings will be 45 to 50 km^2 with a maximum depth of 160 metres at the bottom of the original TMF pond.

Water will be discharged from the TMF for the first 18 years of operation via the West Dam spillway to a monitoring pond located immediately downstream of the West Dam. In Year 19, the TMF discharge will be located at the North Dam spillway as the TMF expands and the water pond moves to the north. The estimated annual TMF discharges are as follows:

- Year 4.5 West Dam spillway 42.3 Mm^3;
- Year 15 West Dam spillway 24.4 Mm^3;
- Year 30 North Dam spillway 22.4 Mm^3.

The TMF water discharge is expected to meet all Quebec Directive 019 and Canadian MMER discharge limits applicable to the Lac Otelnuk mine.

The TMF closure plan is summarized in a separate section below.

b) Waste Rock Management

The proposed mine will have three waste rock piles, one located on the west side of the deposit and two located inside the pit area once the pit floor has been developed. The three waste rock piles will have a capacity of 14, 21 and 21 Mm^3 respectively. The rock piles were designed with a 25º side slope based on a 17.3 m wide berm for each 20 m lift. As stated above, the waste rock is non-acid generating with low metal leaching potential, and the surface waste rock piles have water collection ditches that will be directed to the TMF for reuse in the process. Water from the waste rock piles in the pit will be collected and directed to the pit sumps to be pumped to the TMF.

c) Solid Waste Management

The project design includes sanitary landfills for solid waste disposal at the mine site, in compliance with the Quebec regulations respecting land filling and incineration.

d) Used Oil Management

The project design includes an incinerator for used oil disposal, in compliance with applicable provincial regulations.

e) Surface Water Management

A geochemical characterization study was performed on the waste rock, low-grade ore, ore, tailings and overburden materials (Golder, 2014) showed that the ore does not cause any leaching of metals into the water. All site surface water including water from the concentrator site, ore pads and waste rock piles will be collected and directed to the TMF for use in the process.

f) Domestic and Sanitary Wastewater Management

The design of the permanent accommodation complex includes wastewater treatment units that meet applicable provincial standards.

20.5.2 Port Area and its Infrastructure

The proposed Port Facility and its water management system will be located at Pointe Noire near Sept-Îles. The PDS will have a continuous dewatering system that will include a water management pond with a 38,000 m^3 effective volume and a surface area of 16,000 m^3. This facility will handle surface run off from rain and snowmelt and batch water from the process. During day to day operations the hydraulic retention time will be 24 hours for phase one (25 Mt/y) and 14 hours for phase two (50 Mt/y). The overflow rate will range from 0.1 to 0.16 $m^3/h/m^2$ for Phase 1 and 2 respectively. During the batching process the water management pond will have approximately 7 hours retention time and the discharge rate will increase to 0.25 $m^3/h/m^2$. The size of the pond will provide sufficient time to settle out solid particles.

An emergency concentrate handling pond (89,000 m^3) capable of handling the concentrate volume of one pipeline after settling, will be built upstream of the water management pond and the downstream dyke will be constructed of granular material that will allow it to function as a filter dam.

Some of the filtrate produced from the dewatering process will be re-used as process water at the port. The excess water will be discharged to the St-Lawrence River on the west side of Pointe Noire via an intermittent stream. This effluent meet the discharge criteria set out in the Quebec Directive 019. The geochemical characterization study (Golder, 2014) conducted on the waste rock, low-grade ore, ore, tailings and overburden materials indicated that the ore is non-acid generating and does not leach any metals.

Total suspended solids (TSS) concentration in the filtrate would be an issue of concern. A filtrate sample using magnetite concentrate was filtered with a ceramic leaf tester (slurry dewatering equipment) was revealed that there were no elements of concern. TSS concentration in the filtrate sample was below the discharge criteria to the environment.

20.6 Closure Plan

This section presents the key closure concepts applicable to the Lac Otelnuk Iron Ore Project. Closure concepts were developed in respect with the various government guidelines and regulations namely *the Quebec Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements (Ministère des Ressources naturelles du Québec,* 1997) is herein referred to as "the Guidelines".

The closure concepts for the mine were developed based on the requirements in section 3 of the Guidelines, pursuant to article 232 of the Quebec Mining Act. These closure concepts will also later serve as the basis for preparing a detailed closure plan for the mine site and process plant, as mandated by article 232 of the Quebec Mining Act.

The closure concepts include but are not limited to:

- Mine site and process plant;
- Main access road (185 km) between Schefferville and the mine site;
- TMF and waste rock piles;
- Water management pond;
- 735 kV power transmission line (450 km) from Hydro-Quebec's La Grande power generation station to the LOM mine site;
- PDS from the mine site to the Port of Sept-Îles;
- Product dewatering, storage, and reclaiming facility located at the Port of Sept-Îles.

Areas of potential soil contamination will be identified, characterized, and dealt with, as applicable, in compliance with the Quebec Policy for Contaminated Soils and Rehabilitation. Such areas include fuel storage, fuel distribution, and mechanical maintenance areas.

20.6.1 Mine Site and its Infrastructure

The mine site and process plant closure concepts include the recycling of materials, such as structural steel, and the reuse or sale of mobile equipment, mining equipment, and process equipment as applicable.

a) Mine site and process plant (concentrator)

All buildings and all support structures on the surface will be dismantled. This includes all process equipment, conveyors, HVAC equipment, tanks, above ground piping, electrical equipment and other surface equipment. Criteria for the decontamination of equipment and other materials will apply as required before disposal. All uncontaminated materials that could not be sold or recycled will be disposed of in authorized waste disposal facility. Concrete footings and walls are to be demolished down to the original ground level with the concrete slabs remaining in place then covered. These areas will be re-graded to ensure public safety.

Underground infrastructure at the mine site, such as underground services, underground pipes, other pipes such as water supply, air, and others as applicable, will be left in place.

The airstrip and its associated facilities at the mine site will be left in place.

The closure concepts for the open pit include:

- The walls around the open pit will be inspected and stabilized if required. The open pit will gradually fill with groundwater, precipitation and natural run-off. Geochemistry tests have shown that the rock on the pit walls is not acid generating, and therefore is not expected result in long term water quality issues. Water quality will be monitored as part of the post closure monitoring program;
- All haul roads to the mine pit will be closed using road blocks, such as rock fill berms. Safety warning signs will be posted;
- A 2 m high embankment will be provided around the perimeter of the open pit.

In the subarctic conditions of the mine site, artificial re-vegetation is not technically feasible, nor adapted to sub-arctic conditions; natural tundra type of vegetation will be allowed to re-establish over time.

b) Tailings Management Facility

Similar to the mine site and process plant, the closure concepts for the Tailings Management Facility (TMF) and the waste rock piles were developed based on the requirements of the Guidelines. These closure concepts will be used to develop the detailed closure plan for the TMF and the waste rock piles.

The closure concepts for the TMF include but are not limited to the following:

- The surface of the tailings beach will be provided with an erosion protection cover, to be constructed using material from overburden and waste rock piles;
- The geochemistry test program showed that the tailings material is non-acid generating, not leachable for heavy metals, and is classified as "low risk" under Quebec provincial criteria;
- The tailings dams have upstream and downstream rockfill slopes therefore no upgrades of the perimeter dams for the TMF closure are required;
- Based on the stability assessment of the final West Dam, no closure upgrades are required The downstream rockfill slope of the dam will be designed to satisfy stability requirements for long-term mine closure. The perimeter dams have been designed to meet closure requirements;
- The spillway at the North Dam and the spillway channel to Stream D, both located in bedrock are upgraded for closure;

- The TMF discharge water is expected to meet the applicable discharge limits during operation and after closure;
- After closure, the TMF discharge at the closure spillway in the North Dam will be monitored during closure;
- After closure, the monitoring pond dam will be retained to monitor water discharge at this location and once the discharge consistently meets all required criteria, this dam will be breached for final closure;
- There is a relatively large natural watershed area southwest of the tailings footprint where the run-off from this watershed area flows northeast towards the TMF. At closure, this runoff will be diverted across the tailings surface by riprap-lined lateral channels to discharge to the riprap-lined main channel;
- The exact locations of the lateral channels will be defined in the detailed closure plan. Three lateral channels will cross the complete width of the tailings area;
- Near the Saddle Dam, the small local watersheds will be diverted to a stream, within the drainage patterns along the final access road.

c) Waste Rock Piles

The closure concept for the waste rock piles will follow the requirements of the Guidelines with respect to slope stability and with respect to the run-off water from the stockpiles:

- The design of the waste rock pile will have the slopes meeting the safety factors required for long term stability;
- No cover for the waste rock piles and no water collection are planned for closure. The waste rock is both non-acid generating and low leachability is classified as "low risk".

d) Water Management Pond

For closure the water supply dam will be breached and the natural inflows from the pond to the hydrographic system downstream of the TMF will be re-established.

e) Main Access Road

The main access road between Schefferville and the mine site and the secondary roads on the mine site will be left in place after closure with no planned after closure maintenance.

20.6.2 Product Delivery System

The closure concepts for the PDS includes but are not limited to the following:

- The two (2) pipelines are considered as underground support infrastructure which will remain in place at closure;
- Floating bridges located on the northern sector of the PDS, along with the culverts and pillars together with pipes associated with the pipelines at water crossings will all be removed;

- In the northern portion of the PDS, the pipe bench will be left in place for progressive natural re-vegetation by native vegetation species;
- Borrow areas for the construction of the bench in the northern portion of the PDS will have been rehabilitated before closure in compliance with the Quebec provincial Regulation respecting pits and quarries;
- Pumping stations will be dismantled and removed from the site and sent to recycling facilities.

A comprehensive closure plan will be prepared for the PDS as this phase of the project advances.

20.6.3 Power Transmission Line

The 735 kV power interconnection transmission line (450 km) from Hydro-Québec La Grande power generation station to the mine site is not considered as part of the a mining facility and therefore not subject to mine closure regulations. This 735 kV power interconnection line can be considered as necessary infrastructure to support, on a sustainable basis, the long-term development of this part of the Northern Quebec territory.

20.6.4 Port Area and its Infrastructure

The closure concept for the port area ned associated infrastructure will include the removal of the dewatering process equipment while maintaining the rest of the facility, including the process buildings, administrative and services buildings, associated services (electrical substation, water supply, wastewater treatment, emergency pond, parking, etc.). The reasons for this are as follows:

- The product dewatering, storage, and reclaiming facility is located next to the Port of Sept-Îles and may be considered as a standard industrial facility and therefore not subject to dismantling after closure of the mine;
- The facility is located near the Port of Sept-Îles and near the City of Sept-Îles, which has infrastructure and services readily available for supporting industrial operations. The facility could be sold and reused for similar or other industrial operations;
- Once the dewatering process equipment is removed, the rest of the dewatering, storage, and reclaiming facility could be sold to another industrial operator to take advantage of having their industrial operations next to the Port of Sept-Îles.

A comprehensive closure concept will be prepared for the Port Area and infrastructure as this phase of the project advances.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Costs Estimate

The iron ore deposit has defined resources of over 20 billion tonnes with potential in excess of 30 billion tonnes. The project has a potential to produce 50 million tonnes of concentrate based on a 30-year mine life. The concentrate will produce high grade 68.5 % Fe pellet feed. The Project includes the development of an open pit mine, ore processing plant, tailings management facility, concentrate delivery system to Sept-Îles, 735 kV transmission line and port facilities.

The Lac Otelnuk Project will be developed in two distinct construction phases:

- Phase 1 sized for 30 Mt/y (3 process trains);
- Phase 2 sized for additional 20 Mt/y (2 process trains);
- Each phase will be comprised of 10 Mt/y process train.

The estimate is aligned with the proposed schedule, which is based on five years of construction. This timeline excludes early work for Phase 1 (52 months from first concrete to mechanical completion of process train #3). Production capacity of approximately 30 Mt/y will be achieved in operating year 3. Phase 2 will require four years of construction (40 months from first concrete to mechanical completion of line #5) with full production capacity of approximately 50 Mt/y being achieved in operating year 9.

Emphasis has been placed on low cost country sourcing of equipment and bulk commodities, pre-assembly, modularization and logistics as a means of reducing construction requirements at site and lower overall CAPEX.

All major permanent equipment pricing is based on technically and commercially evaluated budgetary quotes from multiple vendors. Contractor quotes were received for major portions of scope such as site civil work, site access road and pipeline installation. Measured against other classifications, this type of estimate is considered a Class 3 estimate as defined in American Association of Cost Engineers ("AACE") International Recommended Practice No. 47R-11. As such, AACE provides a broad range for accuracy within each estimate class. The level of engineering during the Feasibility Study targeted a CAPEX accuracy of ±15 %.

Table 21.1 shows the organization and division of responsibilities at the major area level with respect to Engineering, Procurement, Material Take-offs ("MTO") and Estimating.

Table 21.2 provides a broad overview of the construction duration by phase and production ramp-up by process train.

Table 21.3 provides a summary of the project Capital Costs by major area and by construction phase in USD x 1,000.

Table 21.1 – Division of Responsibility

MAJOR AREA	RESPONSIBLE ENTITY			
DIRECT	**Engineering**	**Procurement**	**MTO's**	**Estimating**
Mine	SNC-Lavalin/Met-Chem	Met-Chem	Met-Chem	Met-Chem
Run of Mine	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin
Process	SNC-Lavalin/NETC	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin
Infrastructure	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin
Transmission Line	SNC-Lavalin-T&D	SNC-Lavalin-T&D	SNC-Lavalin-T&D	SNC-Lavalin-T&D
PDS: pipelines & pumping stations	Ausenco-PSI	Ausenco-PSI	Ausenco-PSI	SNC-Lavalin
PDS: routing, bench, culverts, bridges & civil	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin
Port Area	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin	SNC-Lavalin
Port Terminal Marine Work	LOM/Sept-Îles Port Authority			
INDIRECT	**Engineering**	**Procurement**	**MTO's**	**Estimating**
Owner's Costs				LOM
EPCM Services				SNC-Lavalin
Construction Field Indirects				SNC-Lavalin
Construction Camp				SNC-Lavalin
Freight				SNC-Lavalin
Contingency				SNC-Lavalin
Escalation				Excluded

Table 21.2 – Construction Duration by Phase & Production Ramp-up by Process Train (Mt/y)

Calendar Year *	2017	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Implementation Phase	**Early Works**	**PHASE 1**							**PHASE 2**						
Construction Year		**1**	**2**	**3**	**4**	**5**	**6**		**1**	**2**	**3**	**4**	**5**		
Operating Year							1	2	3	4	5	6	7	8	9
Train 1							5	10	10	10	10	10	10	10	10
Train 2							5	10	10	10	10	10	10	10	10
Train 3								5	10	10	10	10	10	10	10
Train 4														5	10
Train 5														5	10
Target Production Mt/y							**10**	**25**	**30**	**30**	**30**	**30**	**30**	**40**	**50**
# of Crushers In Operation							2	3	3	3	3	3	3	5	5

Table 21.3 – Capital Cost Estimate Summary for all Major Areas and for Phase 1 and 2 (USD x 1,000)

Area	Description	Total		Phase 1 Total	Phase 2 Total
1000	Mine	221,527		210,870	10,657
2000	Run of mine	482,358		274,798	207,560
3000	Process	3,325,389		1,998,180	1,327,209
4000	Tailings management facilities	404,026		263,271	140,755
5000	Infrastructure & utilities	632,959		494,641	138,318
5310	Main substation (735 / 230 kV)	207,405		139,577	67,828
5360	Power transmission line	690,610		690,610	0
7000	Product delivery system	4,395,896		2,621,436	1,774,460
8000	Port Area	702,842		520,371	182,471
	Subtotal Direct Costs	**11,063,010**		**7,213,750**	**3,849,260**
	Construction field indirects	323,700		202,700	121,000
	Construction Camp and catering	287,800		228,700	59,100
	Travel fly in fly out - contractor	91,860		58,250	33,610
	Freight	225,750		142,210	83,540
	Vendor reps, spare parts, first fills + heavy lift cranes	173,010		105,390	67,620
	EPCM services	750,000		500,000	250,000
	Contingency	700,000		458,000	242,000
	Subtotal Indirect Costs	**2,552,120**		**1,695,250**	**856,870**
	Owner's costs (2.5% of direct costs)	276,000		180,000	96,000
	HQ - extension of Tilly & LeMoyne substation, Line to port	63,200		63,200	
	Power to PDS (pumping stations 2)	201,388		201,388	
	Power to PDS (pumping stations 3)	30,646		30,646	
	Total CAPEX (excluding escalation and risk)	**14,186,400**		**9,384,200**	**4,802,100**
	Escalation	excluded		excluded	excluded
	Risk	excluded		excluded	excluded

21.1.1 Basis of Estimate – Scope

It is assumed that the mine will be operated 365 days per year with iron ore feed production to produce about 50 Mt/y of magnetite iron concentrate at full plant capacity.

The estimate scope of work consists of the following major areas:

- 1000 Mine
- 2000 ROM crushing

- 3000 Process
- 4000 Tailings management facilities
- 5000 Infrastructure and utilities
- 5360 Power transmission line
- 7000 Product delivery system
- 8000 Port Area
- 9000 Project indirect costs

21.1.2 Estimate Structure

a) Work Breakdown Structure

The project Work Breakdown Structure is structured by area, subarea and sector (4-digit code). Engineering deliverables have been developed to the area and subarea levels.

b) Unit of Measure

The estimate uses SI (International System of Units) with the exception of piping, where the nominal diameter is expressed in inches.

21.1.3 Base Currency and Exchange Rates

The estimate base currency is US dollars. The estimate has been developed using quoted currencies and converted to US dollars using the currency exchange rates as of December 1, 2014, shown in Table 21.4 below. Fluctuations to the nominated exchange rates are excluded from the estimate.

Table 21.4 – Currency Exchange Rates

Currency Code	Currency Name	Dec. 1, 2014	% Content
USD	US dollar	1	13.78
CAD	Canadian dollar	0.8756	79.28
EUR	Euro	1.24647	1.58
CNY	Yuan renminbi	0.16294	5.35
AUD	Australian dollar	0.84998	0.01

21.1.4 Basis of Estimate - Labour

a) Labour Sourcing

The estimate is based on 100 % of the construction craft sourced from the province of Quebec. Peak construction levels at the mine, transmission line, PDS and port areas are within the levels of skilled craft availability. If levels are surpassed, craft originating from other provinces are bound by wage rates set forth in the Quebec construction industry collective agreement.

b) Labour Rates

Installation labour costs are based on a sixty-hour work week and a rotation of 3 weeks at site followed by 7 days off (travel included) for construction at the mine and process plant (northern site) and 735 kV transmission line.

Construction at the dewatering, storage and reclaiming facilities near the port of Sept-Îles (southern site) is based on a forty-hour work week.

Construction of the product delivery system pipeline is based on a seventy-hour work week with a rotation of 3 weeks at site followed by 7 days off (travel included). Wage rates for crafts have been established based on the Quebec construction industry labour agreement of hourly labour rates for industrial projects in accordance with the Industrial sector as of August 31, 2014.

Crew wage rates have been established for each commodity based on a craft mix comprised of foreman, journeymen, apprentices and general labour across all construction trades.

Contractor indirect costs are expressed as a percentage of the weighted average crew rate for each commodity. Hourly equipment costs include the material portion (depreciation, interest, cost of repair and maintenance, insurances permits and taxes) and operating portion (fuel, lubricants and filters). Sources for rates include the yearly Quebec Government publication entitled "Taux de Location de Machinerie Lourde" used by the ministry of Transport for civil contracts related to public works, roads and highways, contractor pricing and or pricing received by crane suppliers. The cost of the operator is excluded from the hourly operating cost and included in the crew mix.

All lifting equipment greater than 90 t is excluded from the crew rates and calculated separately in the indirect costs under the "Heavy Lifting Equipment" area.

Room and board allowances are not applicable to crew rates at the northern site since 100 % of construction personnel will be lodged in the construction camp. The southern site near Sept-Îles will require a much smaller workforce. It is assumed that 80 % of the workforce will be non-local. For these workers room and board allowances have been built into the crew rates. It is anticipated that port work will peak at over 400 construction personnel.

Travel costs for contractor manual and non-manual staff is excluded from the crew rates and analyzed separately in the indirect costs. Contractor’s overhead, fixed fees and profit has been applied as percentage of all elements to arrive at an "all-in" crew rate.

Table 21.5 provides a summary of "all-in" crew rates used in the estimate.

Table 21.5 – Hourly Labour "All-in" Crew Rates

Crew Code	Crew Description	Hourly Rate North CAD	Hourly Rate South CAD
41A	Light Civil Work - Temporary Fences / Selective Demolition	$168	$144
41B	Hand Excavation Underground Utilities / Landscaping	$126	$111
41C	Rock Excavation / Blasting + Hauling	$242	$201
41D	Piling + Shoring	$238	$197
41E	Heavy Civil Work = Excavation / Backfill + Compaction	$241	$200
41F	Excavation / Backfill + Compaction / Hauling + Heavy Demolition	$214	$179
41G	Light Civil Work = Excavation / Backfill + Compaction	$196	$165
42A	Concrete Work = Formworks + Reinforcement + Concrete	$131	$116
43A	Structural Work = Unload + Shake out / Erect + Plumb	$168	$145
44A	Heavy Architectural Work = Metallic Roofing / Cladding with Cranes	$162	$140
44B	Light Architectural Work = Gypsum Board / Flooring / Painting	$121	$108
44C	Architectural Work = Masonry / Roofing / Cladding (no cranes)	$133	$117
45A	Heavy Mechanical Work	$150	$130
45B	Light Mechanical Work	$136	$121
45C	Field Erected Tanks, Bins and Silos Work	$160	$138
45I	Mechanical Insulation Work	$122	$110
46A	Piping	$147	$128
46B	Light Piping Works (Building Services - Fire Protection)	$131	$116
46I	Piping Insulation Work	$122	$110
47A	Electrical	$130	$116
48A	Automation	$126	$112
46P	Major Pipeline	$251	

c) Labour Unit Hours and Productivity

Direct field labour is the skilled and unskilled labour required to install the permanent plant equipment and bulk materials at site. Direct field installation man-hours have been developed using estimated unit man-hours for each commodity multiplied by the quantity.

Installation hours have been adjusted to reflect local conditions including: effects of extended overtime, weather and wind conditions, labour availability, expected turnover, site congestion etc.

SNC-Lavalin developed installation crew rates and unit installation hours in advance of contractor budget pricing received during the study. Contractor pricing has served as the basis for comparison to SNC-Lavalin estimated labour costs only and the estimated installation hours and costs have not been influenced by contractor pricing. For all C-packages where pricing was requested, SNC-Lavalin estimated costs fell within the range of bids received.

21.1.5 Basis of Estimate – Quantity and Pricing Development

a) Quantity Development Overview

The estimate quantities provided by engineering were based on key engineering deliverables that were reviewed jointly between the estimators and discipline engineers during the scope and quantity reviews.

All quantities generated for the estimate exclude contingencies of any kind. Allowances for elements such as platforms, machine guarding, etc. have been identified separately. Allowance for design growth with respect to quantities and or pricing have been established according to the general guidelines described in the following Table 21.6.

Table 21.6 – Allowances for Design Growth and Losses

Item	Allowance
Concrete pouring losses	5 % applied to the concrete supply price
Concrete MTO's	5 % design growth applied to all MTO's
Steel MTO's	7 % for connections; 5 % allowance on Chinese steel for thicker plates for built-up sections for strength to account for differences in inertia between rolled sections
Mechanical and electrical equipment	No design growth on quantity or capacity
Piping	5 % applied for fabrication losses
Wire and cable	Cut & waste allowance of 10 % applied to material pricing

b) Major Quantity Summary

Quantities for the main process areas were generated by engineering based primarily on Smart Plant SP3D modelled quantities for concrete, steel and process piping which was modelled from 4" in diameter and up. The use of modelling for the concentrator (area 3000) was extensive. Quantities for the PDS (pipelines and pumping stations) were generated by Ausenco-PSI based on detailed lists and drawings. Quantities for the PDS bench, trench, culverts, bridges and other PDS civil work were generated by SNC-Lavalin.

The port area buildings were not modeled but quantities for the major buildings were generated by engineering based on preliminary design and mechanical layout drawings.

Table 21.7 provides a summary of quantities by major areas.

Table 21.7 – Summary of Quantities by Major Area

Cost Element	Unit	Mine	ROM Crushing	Process	Tailings	Infrastructure & Utilities	Product Delivery System	Port Terminal	Total Quantity
Earth machine excavation mass	m³	14,655	378,700	375,900	59,500	2,026,900	825,000	144,100	3,824,755
Rock excavation	m³	0	433,000	1,123,000	43,400	425,100	30,800	142,000	2,197,300
Detailed rock excavation	m³	2,111	18,739	678,111	45,928	17,518	12,359	87,408	862,174
Mass common backfill	m³	22,700	903,600	229,300	116,500	1,310,900	92,500	681,200	3,356,700
Granular backfill	m³	42,660	2,902,500	161,500	95,100	888,880	2,498,200	166,800	6,755,640
Structural granular backfill	m³	700	0	1,023,100	0	25,400	44,000	167,000	1,260,200
Pipe piles	m	0	0	0	0	0	800	18,513	19,313
Cast in-place concrete (excludes lean)	m³	3,213	6,226	211,795	16,438	21,986	41,448	48,849	349,955
Precast concrete	m³	137	6,464	19,304	1,487	6,305	321	2,907	36,924
Steel	T	1,138	5,496	99,928	4,138	12,826	10,856	29,868	164,250
Insulated metal roof cladding	m²	5,570	6,544	127,562	10,507	9,557	60,278	125,210	345,228
Un-insulated metal roof cladding	m²	40	1,440	28,098	0	473	0	5,540	35,591
Membrane roof	m²	930	0	59,556	0	37,513	0	0	97,999
Insulated steel wall siding	m²	5,171	11,538	214,921	39,026	66,247	56,531	42,301	435,735
Un-insulated steel wall siding	m²	72	0	105,549	0	1,778	0	10,335	117,734
Piping (excluding PDS pipelines)	**m**	**0**	**2,566**	**107,346**	**54,136**	**117,965**	**11,412**	**19,147**	**312,572**
Small Bore (4" and less)	m	0	2,273	48,200	1	84,114	1,172	8,070	143,830
Medium Bore (6" to 12")	m	0	293	25,779	3,188	28,679	3,739	8,572	70,249
Medium Bore (14" to 24")	m	0	0	17,232	3,253	3,490	1,104	1,762	26,842
Large Bore (26" to 48")	m	0	0	12,054	46,202	1,683	5,396	743	66,078
Large Bore (>48")	m	0	0	4,082	1,492	0	0	0	5,574
PDS Pipelines	**km**	**0**	**0**	**0**	**0**	**0**	**1,509**	**0**	**1,509**
735 kV transmission line	**km**	**0**	**0**	**0**	**0**	**466**	**0**	**0**	**466**
Electrical									
Installed Motor Rating	MW	36	49	985	50	68	150	50	1,387
MV Insulated Power Cables (1 KV to 50 KV)	m	50	12,220	144,740	4,905	6,555	8,285	10,815	187,570
LV Insulated Power Cables (<= 1 KV)	m	150	11,340	269,415	15,565	19,790	41,985	37,110	395,355
Control cables	m	0	17,365	232,015	14,175	34,760	1,350	35,845	335,510
Grounding wires & cables	m	0	0	0	0	28,000	0	0	28,000
Cable trays & accessories	m	125	3,305	96,283	2,053	5,524	2,050	6,270	109,340

c) Pricing Development Overview

Pricing for process equipment and major bulk material packages is based primarily on budgetary quotes from multiple bidders. Formal requests were issued to bidders based on general specifications and data sheets. Formal bids received have been technically and commercially reviewed. Final recommendations for formal bids have been issued to Estimating after review by Lac Otelnuk.

To the extent possible packages maximized pre-assembly of equipment. Overland conveyor pricing is based on pre-assembled sections.

The following Table 21.8 describes the percentage of pricing sources for mechanical equipment.

Table 21.8 – Pricing Sourcing for Mechanical Equipment

Pricing Source	**Supply Value USD**	**% of Value**
Budget Quote +/-10 % (technical and commercial review)	1,452,581,033	81.5 %
Informal Quote +/-20 % (data sheet and technical review)	124,859,210	7.0 %
Estimated (+/- 25 %)	204,935,366	11.5 %
TOTAL	**1,782,375,608**	**100 %**

21.1.6 Indirect Costs

a) Construction Field Indirect Costs

Costs include all temporary roads, fencing and temporary buildings, lay down areas, material handling and warehousing, construction services (electrical energy production, fuel supply, temporary potable and waste water services, surveying, security, medical, scaffolding, janitorial, concrete testing, craft training, etc) and allowances for construction & safety signs, plant sheds, barricades, guard rails, etc.

Costs associated with security and safety induction for construction personnel are also included.

b) Construction Camp and Travel

The main construction camp for the Process Plant areas will be constructed in Phase 1 and will be used to support Phase 2 construction. Camps for the PDS and transmission lines are included in their respective direct costs, as these will be contractor supplied.

Based on the duration and estimated hours for Phase 1 construction, the camp at the process plant area has been sized for 1,950 beds. Phase 2 construction will peak at 1,450. It is assumed that the Phase 1 camp will be purchased and be used for Phase 2 construction.

The Phase 1 camp is priced at $50,000 CAD per bed which is aligned with recent project data and recent informal pricing from camp suppliers.

Camp catering for Phase 1 is based on 11 M direct and 16.7 M total process plant site hours and 1.95 M camp person-days at $60 CAD per day exclusive of diesel fuel. It is estimated that 37 M litres of fuel will be consumed for camp operation related to Phase 1 construction, with a peak demand of 12 MW.

Phase 2 process plant area construction is based on 6.3 M direct and 9.6 M total process plant site hours and 1.12 M camp person-days at $60 CAD per day.

Travel costs (fly in fly out) for contractor personnel are based on 83 k trips for Phase 1 and 48 k trips for Phase 2 construction at $800 CAD per round trip. These numbers exclude the PDS and transmission line, which have these costs in the direct costs of the estimate.

Work at the port will peak at roughly 450 direct craft.

There is no allowance for a camp at the port. Room and board has been built into the crew rates for construction at the port.

c) Freight Costs

Cost for freight, including land transportation and handling, marine transportation and on site handling and transportation was evaluated by package and major bulk commodity based on weights and shipping volumes. SNC-Lavalin procurement obtained quotes for marine freight for both container and break bulk cargo for various ports of exit. These rates were applied to the shipping volumes provided to develop marine costs. Likewise quotes were obtained for the transport of cargo by rail from Sept-Îles to Schefferville. Finally, trucking rates for the portion from Schefferville to the site are based on $2,160 per round trip.

The net value of freight represents 5.7 % of supply excluding the transmission line and main sub-station whose freight costs are part of the direct costs. This percentage is explained by the fact that many of the purchase package pricing and the PDS mainline pipe pricing is delivered FOB port of Sept-Îles.

d) Vendor Site Representation

Vendors were asked to provide per diem rates for construction and pre-commissioning support as well as an estimate of days required for each effort. An estimate was prepared by SLI based on historical data in unison with the vendor per diem rates to establish vendor costs per package including costs for air travel and allowances for subsistence.

The net value represents 1.9 % of the value of mechanical equipment.

e) Heavy Lift Cranes

The hourly labour crew rates allow for cranes up to 90 t capacity. Heavy lift cranes will be required to erect steel at the process plant buildings and for lifts that are beyond the realm of a 90 t capacity due to either reach or weight of equipment and or assemblies. The estimate allows for ten (10) 400 T cranes for Phase 1 construction and seven (7) 400 T cranes for Phase 2.

f) Spare Parts

Spare parts have been calculated by package for Commissioning, 1-year Operating and Capital Spare Parts based on information obtained in the budgetary quotes.

The net value represents 3.6 % of the value of mechanical equipment.

g) First Fills

This account covers the initial fills of grinding media for the Sag and Ball mills. The balance of first fills such as lubricants, reagents and fuel are included in the OPEX estimate or part of the Owner's Costs allowance.

h) EPCM Services

EPCM services are calculated based on a percentage of direct costs. The percentages vary by major area, according to the design effort, schedule duration and effort required to support construction. The costs cover the detailed execution phase up to mechanical completion and pre-operational verification (POV). Costs associated with this study and all other previous studies are excluded from this amount, as are costs associated with potential intermediate phases related to value engineering and/or basic engineering prior to the commencement of detailed design.

The following percentages were applied by major area to generate the EPCM costs:

- Mine, run-of-mine, process plant, tailings and infrastructure – 9 %;
- 735 kV transmission Line –5 %;
- 735 kV/230 kV Main Substation – 10 %;
- Product Delivery System – 4 %;
- Port Area – 10 %.

i) Taxes & Duties

All taxes and duties are excluded from the estimate.

j) Contingency

The contingency evaluation was structured by discipline, followed by package and/or area and limited to direct costs and indirect costs excluding contingency, owner's costs, escalation and risk.

A probabilistic analysis was performed using Monte Carlo simulation. This approach provides the level of contingency as a function of probability of under-run and also provides the level of confidence or probability that the estimate falls with the target precision.

Contingency at P_{80} yields 12 % contingency which is SNC-Lavalin's recommendation at the FS stage.

LOM have instructed SNC-Lavalin to set contingency at P_{50}, which represents an equal chance (50 % of probability based on the Monte Carlo simulations) that the project

CAPEX will be higher or lower than the current estimate. The P_{50} yields 5.4% contingency.

The simulation results show that approximately 90 % of the simulation results fall within the target accuracy of +/- 15 %.

21.1.7 Owners Costs

Owner's Costs were established by LOM based 2.5 % of direct costs. SNC-Lavalin deems that the costs of the following list of exclusions are covered in owner's costs:

- Owner’s contingency;
- Owner’s team;
- Land acquisition and rights of way;
- Environmental permitting;
- Environmental monitoring, water analysis, etc.;
- Deferred or sunk costs (spent study costs);
- Capitalized interest;
- Cost of working capital;
- Owner's project office (other than space provided by the EPCM contractor’s site construction office) including rent, communications, furniture and equipment, and office supplies;
- Owner’s contribution to the potential wharf construction;
- Public relations;
- Owner’s travel, legal and other corporate office charges to the Project;
- Taxes and duties;
- Owner’s consultants (legal, environmental, etc.);
- Cost associated with start-up and commissioning;
- Modifications after start-up;
- Relationships with government authorities;
- Project insurance including comprehensive general liability and insurance for construction equipment and tools, builder’s all-risk insurance;
- Performance bond premiums;
- Closure cost bond premiums;
- Allowance for upgrade of any off-site facilities;
- Removal and disposal of hazardous materials;
- Training of plant operating personnel.

21.1.8 Escalation

All costs were developed in present pricing valid to the estimate base date. All forward escalation from the estimate base date through to mechanical completion is excluded from SNC-Lavalin's estimate.

21.1.9 Qualifications and Exclusions

The following list defines the qualifications and exclusions of the SNC-Lavalin estimate:

- The estimate is expressed in fourth quarter 2014 USD;
- Forward escalation is excluded;
- CAPEX and schedule risk are excluded;
- Mitigation costs associated with risks identified in the risk register are excluded;
- Fluctuations to nominated currency exchange rates are excluded;
- Allowance for industrial dispute or lost time arising from industrial actions is excluded;
- Project financing and interest during construction is excluded;
- Acceleration or deceleration of the project schedule is excluded;
- Plant operating costs are produced by SNC-Lavalin but are excluded from the CAPEX and not addressed in the basis of estimate. There will be an independent document that describes the methodology and basis for the OPEX estimate;
- The cost of working capital is excluded;
- Plant mobile equipment is considered included in the fixed operation costs;
- Owner's costs are provided by LOM based on an allowance derived as 2.5 % of direct costs.

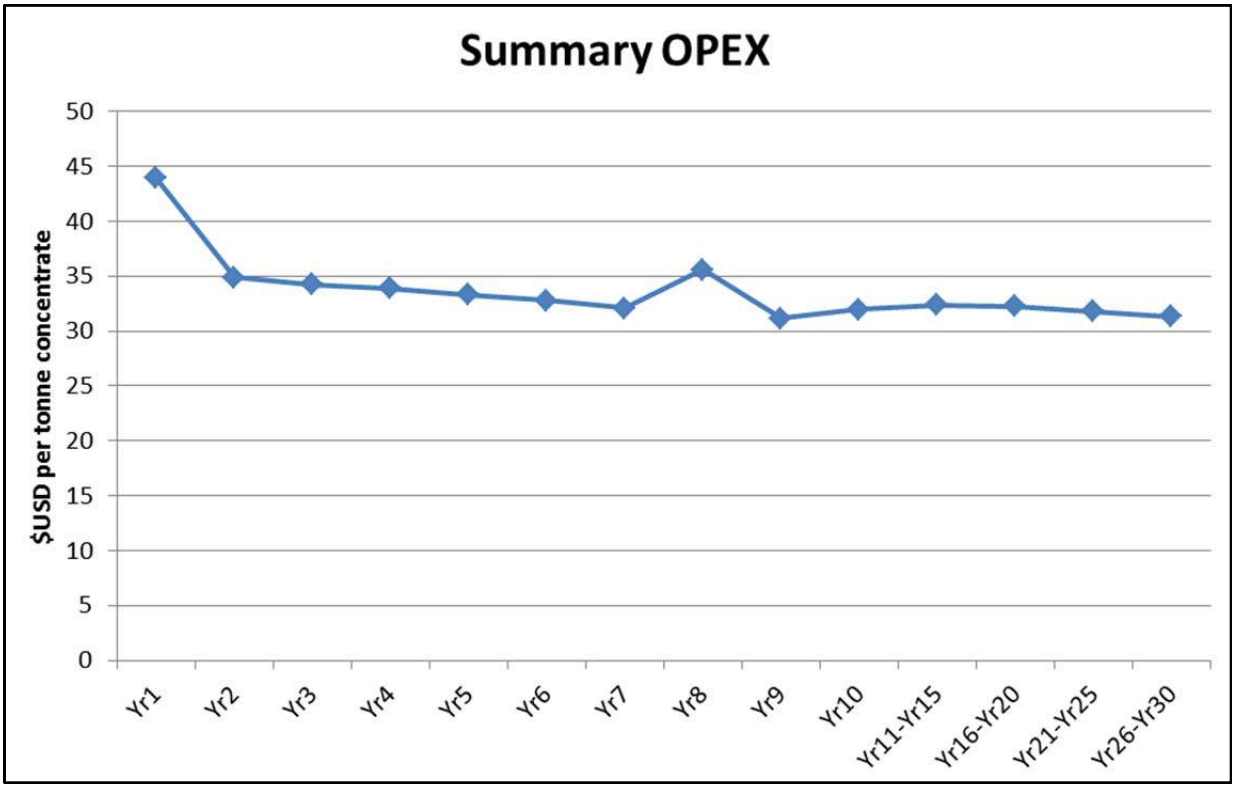
21.2 Operations Costs Estimate

This section presents the annual operating cost estimates (OPEX) for the Lac Otelnuk Iron Ore Project. The level of engineering during the Feasibility Study targeted an OPEX accuracy of ±15 %.

21.2.1 Operating Costs Summary

The OPEX is a function of the iron ore concentrate production and has been developed for each production year up to the 30-year mine life. Figure 21.1 shows the estimated calculated cost per tonne of concentrate over the life of mine. Year 1 is when the plant starts to produce concentrate and Year 3 is the first year that the plant achieves its nominal Phase I production of 30 Mt/y. Year 8 is when Phase 2 ramp-up begins and Year 9 is the first year the plant achieves nominally 50 Mt/y of concentrate.

Figure 21.1 – Operating Unit Cost over Life of Mine



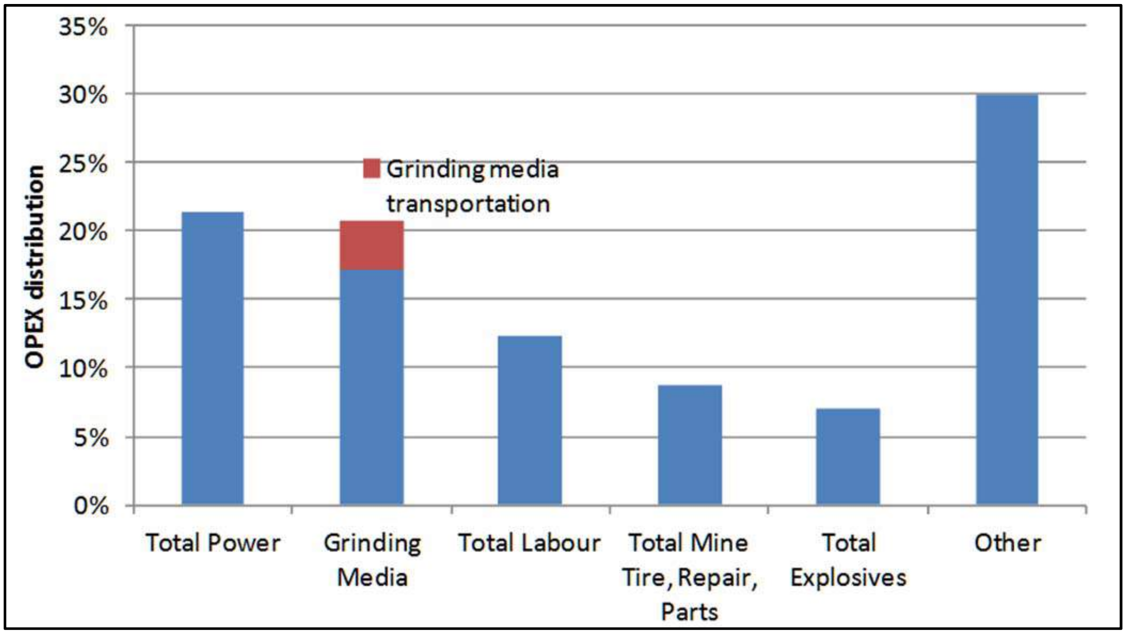
The OPEX breakdown per area for year 3 (first year at nominally 30 Mt/y of iron ore concentrate) and year 9 (first year at nominally 50 Mt/y of iron ore concentrate) is shown in Table 21.9.

Table 21.9 – OPEX Breakdown per Area for Phase 1 and 2

Area	Phase 1, Year 3 (30 Mt/y)		Phase 2, Year 9 (50 Mt/y)	
	USD	USD per tonne concentrate	USD	USD per tonne of concentrate
1000 – Mine	372,589,384	12.40	506,810,637	10.14
2000 – ROM	22,687,640	0.76	39,310,438	0.79
3000 – Concentrator	455,323,798	15.16	754,774,984	15.10
4000 - TMF	29,105,890	0.97	40,455,527	0.81
5000 – Infrastructure and utilities	79,752,741	2.66	103,595,354	2.07
7000 – Product delivery system	43,984,853	1.46	71,608,409	1.43
8000 – Port Area	24,138,432	0.80	38,886,823	0.78
Total	**1,027,582,737**	**34.21**	**1,555,442,171**	**31.12**

The major OPEX items over the life of mine are identified in the Figure 21.2 below.

Figure 21.2 – Major OPEX Items over Life of Mine



Note: Total Labour – The labour for the construction of the Tailing Management Facility up to Year 2 falls under CAPEX. The OPEX labour for tailings dam starts at year 3, when the dam construction activities fall under sustaining capital.

21.2.2 Basis of Estimate

The OPEX is based on the following and reflects these key assumptions:

- The OPEX battery limit starts at the mine and includes PDS, Power Supply, Main Access Road, Port Area (Product Dewatering, Storage, Reclaiming and the ship loading of concentrate) for a life of mine of 30 years, developed in two phases;
- The model has been built up with prices predominately in Canadian dollars (CAD) and do not include sales tax. The final OPEX estimate is converted to USD;
- The exchange rates used are dated December 1, 2014, and are as follows:
  - USD 0.88 = CAD 1.00
  - CNY (China) 5.37 = CAD 1.00
- The price of energy is based on Hydro-Québec's "L" rate, which is the rate applicable to industrial activity with a demand of more than 5 MW. The rate is comprised of a fixed component (peak demand) and a variable component (average consumption). Hydro-Quebec performed simulations for the mine site, the PDS pumping stations, and the Port Area. The calculated combined fixed and variable average varies from 0.046 to 0.0495 $/kWh;
- The overall fuel cost of 1.316 $/l is based on an email quote from Harnois Groupe Petrolier (Saint-Thomas, Quebec, J0K 3L0) received August 22, 2014. Low-temperature clear diesel fuel is available in Sept-Îles. The price per litre includes a fuel cost of 0.907 $/l, fuel tax of 0.2038 $/l, and mark-up of 0.018 $/l. The price does not

include GST and PST taxes. From Sept-Îles, the fuel will be transported to Schefferville by rail (0.100 $/l) using 45,000-litre trailers and then trucked (by a contractor) to the mine site (0.050 $/1). The cost of trucking the fuel from Schefferville to the mine site includes the return trip to Schefferville. The empty fuel trailer is then returned to Sept-Îles by rail (0.038 $/l);

- Fuel consumption is based on an hourly rate for all mining equipment. For all other areas of the Project, an allowance of either 50 or 100 l/day for fuel consumption per mobile equipment is included;
- The manpower required for the operation is segregated by areas. Labour costs are based on typical salaries in northern Quebec and are verified with the Canadian Mine Salaries, Wages, and Benefits 2014 edition (Salzer, 2014);
- The electrical consumption for the crushing plant, process plant, tailings management facility, infrastructure & utility were developed using the electrical load list document. The electrical consumption is estimated based on the nominal power requirements and a utilization factor for nominally 50 Mt/y of concentrate. All production years prior to 50 Mt/y are prorated;
- Allowances are given for safety consumables ($500) and general consumables ($500) on a per employee per year basis.
- For most processing equipment, annual equipment maintenance (4 %) and annual overhaul costs (2 %) is based on the mechanical equipment cost. Exceptions are gyratory crushers, SAG mills, ball mills, screens, and LIMS drums, for which the maintenance and annual overhaul costs are itemized and included separately;
- Operating supplies are estimated to be 4 % of the Labour cost;
- Laboratory supplies have an allowance of $50,000;
- Building maintenance (architectural) is estimated to be 1 % of the building CAPEX;
- Allowances are given for consultants, community relationship, recruitment, and communication;
- Training is based on 1 % of all direct labour cost (salary + fringe);
- The mobile equipment requirements have been compiled. Major mining mobile equipment costs are based on budget quotes with financing. For all other mobile equipment, an allowance is included based on the mobile equipment list developed for the feasibility study;
- Head office labour is included in the manpower requirements under the Port Area;
- The ship loading of the concentrate will be done with the deep water-port facilities owned and operated by the Sept-Îles Port Authority. The fee for using the facilities is on a per tonne basis. This rates needs to be negotiated by LOM. As per the MOU signed between the Sept-Îles Port Authorities and LOM dated December 16, 2014, $0.40 / t conc. is applied. It is to be noted that for the purpose of the feasibility study, the purchase and installation of the ship loader is by LOM.

21.2.3 Mining Operating Costs

The mine operating cost estimate breakdown for year 3 (first year at nominally 30 Mt/y of iron ore concentrate) and year 9 (first year at nominally 50 Mt/y of iron ore concentrate) is shown in Table 21.10.

Table 21.10 – Estimated Mine Operating Cost Breakdown for Phase 1 and 2

Cost	Phase 1, Year 3 (30 Mt/y) USD	Phase 2, Year 9 (50 Mt/y) USD
Variable		
Fuel	54,071,980	92,198,679
Tires	17,240,799	30,301,330
Repair / parts	53,130,913	90,208,306
Electricity	2,769,395	4,855,698
Explosives	61,866,024	102,315,460
Financing	72,027,919	45,021,710
Other (clearing, overburden contract, dispatch, etc.)	24,340,711	24,839,185
Sustaining Capital	22,326,168	16,509,680
Fixed		
Labour (salary with fringe benefits only)	64,708,072	95,892,206
Building maintenance (% of capital cost of architectural building)	107,404	160,284
Cost per tonne mined	**2.7**	**2.3**
Cost per tonne ore processed	**3.6**	**2.7**
Cost per tonne concentrate	**12.4**	**10.1**

a) Financing

Financing is applicable to the major mining equipment (haul truck, cable shovel, etc.) and support equipment (dozers, grader, water-sand truck, etc.). Based on the budget quote from Caterpillar, the financing is a 25 % down payment, followed by a monthly fee over 60 months with a 6 % interest, after which the equipment is owned by the mine. The down payment is capital cost and the monthly payments which include both principal and interest are operating costs.

b) Trucks

Haul trucks are purchased as required (following the production ramp-up plan), and fall under sustaining capital beyond year 2 of production.

c) Mine Equipment Parts

All mine equipment parts cost includes transportation to site. Major suppliers assume the presence of their own on-site shop.

21.2.4 Run of Mine Operating Costs

The Run of Mine operating cost estimate breakdown for year 3 (first year at nominally 30 Mt/y of iron ore concentrate) and year 9 (first year at nominally 50 Mt/y of iron ore concentrate) is shown in Table 21.11.

Table 21.11 – Estimated ROM Operating Cost Breakdown for Phase 1 and 2

Cost	**Phase 1, Year 3 (30 Mtpa) USD**	**Phase 2, Year 9 (50 Mtpa) USD**
Variable		
Electricity	5,721,152	9,521,260
Gyratory spares	8,668,406	15,957,760
Fixed		
Building maintenance (% of capital cost of architectural building)	194,332	383,484
Equipment maintenance	2,825,218	4,860,115
General consumables (camp items, soaps, toilet paper, etc.)	21,890	35,900
Safety consumables (hard hat, safety boots, etc.)	21,890	35,900
Labour (salary with fringe benefits only)	5,033,414	8,188,480
Operating supplies (lubricants, test chemicals, custodial supplies, etc.)	201,337	327,539
Cost per tonne concentrate	**0.76**	**0.79**

21.2.5 Concentrator Operating Costs

The concentrator operating cost estimate breakdown for year 3 (first year at nominally 30 Mt/y of iron ore concentrate) and year 9 (first year at nominally 50 Mt/y of iron ore concentrate) is shown in Table 21.12.

Table 21.12 – Estimated Concentrator Operating Cost Breakdown for Phase 1 and 2

Cost	Phase 1, Year 3 (30 Mtpa) USD	Phase 2, Year 9 (50 Mtpa) USD
Variable		
Electricity	155,530,579	258,837,206
Grinding media	165,828,356	275,974,978
Grinding media transportation	34,597,050	57,577,126
Liners / mantles / screens / LIMS drum parts	45,418,614	83,611,606
Fixed		
Labour (salary with fringe benefits only)	37,931,795	52,704,974
Building maintenance (% of capital cost of architectural building)	5,992,818	9,988,679
Equipment maintenance	6,412,892	10,882,107
General consumables (camp items, soaps, toilet paper, etc.)	159,359	222,402
Safety consumables (hard hat, safety boots, etc.)	159,359	222,402
Operating supplies (lubricants, test chemicals, custodial supplies, etc.)	1,517,272	2,108,199
Laboratory supplies allowance	43,780	43,780
Fuel mobile equipment	1,598,832	2,524,471
Cost per tonne concentrate	**15.2**	**15.1**

21.2.6 Tailings Management Facilities Operating Costs

The TMF operating cost estimate breakdown for year 3 (first year at nominally 30 Mt/y of iron ore concentrate) and year 9 (first year at nominally 50 Mt/y of iron ore concentrate) is shown in Table 21.13.

Table 21.13 – Estimated TMF Operating Cost Breakdown for Phase 1 and 2

Cost	**Phase 1, Year 3 (30 Mt/y) USD**	**Phase 2, Year 9 (50 Mt/y) USD**
Variable		
Electricity	3,602,207	5,994,867
Flocculent (tailings)	8,806,547	16,845,296
Tailings sustaining capital	14,055,225	13,137,492
Fixed		
Labour (salary with fringe benefits only)	2,006,522	2,742,026
Building maintenance (% of capital cost of architectural building)	175,180	300,750
Equipment maintenance	364,188	1,304,401
General consumables (camp items, soaps, toilet paper, etc.)	7,880	10,507
Safety consumables (hard hat, safety boots, etc.)	7,880	10,507
Operating supplies (lubricants, test chemicals, custodial supplies, etc.)	80,261	109,681
Cost per tonne concentrate	**0.97**	**0.81**

21.2.7 Infrastructure and Utilities Operating Costs

The operating cost estimate breakdown for general, administration, infrastructure, and utilities for year 3 (first year at nominally 30 Mt/y of iron ore concentrate) and year 9 (first year at nominally 50 Mt/y of iron ore concentrate) is shown in Table 21.14.

Table 21.14 – Infrastructure and Utilities Operating Cost Breakdown for Phase 1 and 2

Cost	Phase 1, Year 3 (30 Mt/y) USD	Phase 2, Year 9 (50 Mt/y) USD
Variable		
Electricity	11,442,304	19,042,519
Fixed		
Building maintenance (% of capital cost of architectural building)	1,065,055	1,347,893
Equipment maintenance	1,254,507	1,518,870
General consumables (camp items, soaps, toilet paper, etc.)	73,550	83,182
Safety consumables (hard hat, safety boots, etc.)	73,550	83,182
Labour (salary with fringe benefits only)	17,143,441	19,293,555
Administrative operating supplies	685,738	771,742
Fly-in / fly-out	11,487,121	16,454,525
Catering	16,075,330	22,198,205
Other (recruitment, consultants, communication, etc..)	8,073,943	8,073,943
Road maintenance	1,444,740	1,444,740
Transmission line maintenance	3,550,000	3,550,000
Cost per tonne concentrate	**2.66**	**2.07**

21.2.8 Product Delivery System Operating Costs

Pumps stations 1 and 2 will have electricity supplied from the mine site. Pump station 3 will be connected to the Fermont grid. The terminal will have power supplied from the Port Cartier grid. The electricity cost is for all three pumping stations and the terminal.

The PDS operating cost estimate breakdown for year 3 (first year at nominally 30 Mt/y of iron ore concentrate) and year 9 (first year at nominally 50 Mt/y of iron ore concentrate) is shown in Table 21.15.

Table 21.15 – Estimated PDS Operating Cost Breakdown for Phases 1 and 2

Cost	Phase 1, Year 3 (30 Mt/y) USD	Phase 2, Year 9 (50 Mt/y) USD
Variable		
Electricity	22,263,345	37,051,119
Fixed		
Pumping Stations building maintenance (% of capital cost of architectural building)	938,651	1,627,339
Equipment maintenance	5,779,729	5,779,729
General consumables (camp items, soaps, toilet paper, etc.)	43,780	78,804
Safety consumables (hard hat, safety boots, etc.),	43,780	78,804
Labour (salary with fringe benefits only)	9,609,817	17,359,854
PS2 drive-in / drive-out	94,351	188,701
Catering	89,060	178,120
Administrative operating supplies	384,393	694,394
Fuel mobile equipment	715,267	757,341
Road maintenance	1444740	1444740
Transmission line maintenance	1205000	1205000
Outside engineering	1313400	1313400
Cost per tonne concentrate	**1.46**	**1.43**

The PDS operating costs include the pumping station terminal near the port.

21.2.9 Port Facilities Operating Costs

From the PDS terminal where the pipeline is depressurized, the slurry is sent to the dewatering plant at the Port. Water from the dewatering plant is discharged into the St-Lawrence River. The dewatered concentrate is sent to the stockpile area and then loaded onto a conveyor system for transport to the wharf located approximately 1.3 km away.

The port area cost estimate breakdown for year 3 (first year at nominally 30 Mt/y of iron ore concentrate) and year 9 (first year at nominally 50 Mt/y of iron ore concentrate) is shown in Table 21.16.

Table 21.16 – Estimated Port Area Facilities Operating Cost Breakdown for Phase 1 and 2

Cost	Phase 1, Year 3 (30 Mt/y) USD	Phase 2, Year 9 (50 Mt/y) USD
Variable		
Electricity	4,975,836	8,280,889
Fees for the use of the Port of Sept-Îles wharf	10,518,895	17,505,762
Fixed		
Building maintenance (% of capital cost of architectural building)	2,069,845	2,373,073
Equipment maintenance	5,752,730	9,553,616
General consumables (camp items, soaps, toilet paper, etc.)	70,048	84,933
Safety consumables (hard hat, safety boots, etc.)	70,048	84,933
Labour (salary with fringe benefits only)	134,842	134,842
Administrative operating supplies	5,394	5,394
Fuel mobile equipment	420,745	757,341
Transmission line maintenance	50,000	50,000
Cost per tonne concentrate	**0.80**	**0.78**

21.2.10 Exclusions

The following list defines the exclusions of the OPEX estimate:

- Corporate overhead (only labour is included in the estimate);
- All cost incurred after ship loading concentrate (i.e. ocean freight, ship unloading, etc.);
- Depreciation and amortization;
- Risk analysis and contingencies;
- Exploration;
- Interest charges on CAPEX;
- Cost escalation;
- Permitting costs and land acquisition;
- Customs and duties.

21.2.11 Operations and Manpower

The organizational structure is designed to support the successful operation of the Lac Otelnuk Iron Ore Project, while producing cost-effective, quality iron ore concentrate.

The organizational structure provides for appropriate ownership and accountability of all areas of the operation. It also provides for the long-term training as required to support a structured transition to an increasingly sustainable workforce and leadership over time.

a) Operating Philosophy

The primary project objective is to provide a safe and high performance working environment that creates values for LOM's shareholders and the local communities. It is critical to the success of the Project to perform effectively in the commissioning and ramp-up phases, and then to further improve through lessons learned during operations.

The mine and the ore processing plant operations are based on a schedule of 24 hours a day, 365 days a year. The utilization factor is 70 % for the primary crushers and 90 % for the balance of the Lac Otelnuk Iron Ore Project operations. This allows for putting operation lines out of service for either planned maintenance or unplanned events. During this condition, production on the remaining operating streams will be optimized to minimize the production disruption.

b) Documents

Useable, controlled, and accessible documentation will be developed during the project execution phase, including equipment documents, design documentation, operating procedures, maintenance procedures, and work practices. This documentation will be instrumental in maintaining safety, production performance, and consistent practices throughout the operations phase.

c) Recruitment

The recruitment process will help identify prospective qualified employees who demonstrate a level of safety awareness. Further induction and training programs will be geared towards promoting a culture where safety is critical to the successful operation of the Project.

d) Language of Operations

French and English will both be used as the official languages of the operation. The language employed for various roles and activities is shown in Table 21.17.

Table 21.17 – Language of the Operation

Personnel	Language
Dealings with the Government of Quebec	French
Dealings with the Government of Canada	English and French
Company senior managers	English and/or French
Supervisory staff, initially dual language	French and English
Tradesmen	Predominantly French
Operators	Predominantly French
Labourers	Predominantly French
Contractors	Predominantly French
Vendors	French and English

e) Workforce Estimate

Table 21.18 below provides a summary of the total estimated workforce by phase as well as the number of active employees on site at any given time.

Table 21.18 – Estimated Workforce Summary

Total Estimated Workforce	**Phase 1 (30 Mtpa)**	**Phase 2 (50 Mtpa)**
Mine	324	566
Mine Engineering Geology	28	38
Field Maintenance	92	146
Shop Maintenance	218	252
Services Plant, Airport Access Road	64	64
Total Process Plant Operation	432	614
G&A Infrastructure - Mine	104	126
Slurry Transport - Mine	28	56
Intermediate Pump Station PS2	16	32
Intermediate Pump Station PS3	16	32
Port - Dewatering and Material Handling	83	117
Port - Product Delivery System	40	60
G&A Montreal	43	43
G&A Sept-Îles	34	34
TOTAL	**1522**	**2180**
Estimated Workforce Active at Any Given Time	**Phase 1 (30 Mtpa)**	**Phase 2 (50 Mtpa)**
Active employees, contractors, and visitors at mine site	758	1057
Active employees at PS2	8	16
Active employees at PS3	8	16
Total employees at Sept-Îles	34	34
Total employees at Pointe-Noire	123	177
Total Employees in Montreal	43	43
TOTAL	**974**	**1,343**

22.0 ECONOMIC ANALYSIS

22.1 Summary

Based on the current assumptions, discounted cash flow modeling of the project yields an internal rate of return ("IRR") estimate of **13.0 %** after income taxes and before any withholding tax, in real dollars (i.e. without taking any inflation into account). A summary of the results is presented in the following Table 22.1:

Table 22.1 – Base Case Nominal 30-Year Mine Life (Actual Analysis on 29.7 Years) (100 % Production to China)

	Before Taxes	After Taxes
Project IRR*	15.8 %	13.0 %
NPV @ 6 %*	$ 17,457 million	$ 9,647 million
NPV @ 8 %*	$ 10,388 million	$ 5,240 million
NPV @ 10 %*	$ 5,906 million	$ 2,440 million
Payback Period**	7.0 years	7.3 years

* Based on Free Cash Flow to Equity.
** Calculated from start of commercial production and based on Free Cash flow to Equity.
All monetary values in USD, unless otherwise stated.

The primary project variables utilized in the financial and economic analysis of this project are outlined below:

- Estimated Construction Cost (Phase 1): USD 9.38 B
- Estimated Construction Cost (Phase 2): USD 4.80 B
- Estimated Operation Period: 29.7 years (base case)
- Future iron ore concentrate prices were evaluated by SNL Metals & Mining Ltd. for the Lac Otelnuk Project in real USD. Please refer to the Market Study Section 19.0 of this report.
- Financing Sources: 100 % Equity. Equity is drawn to finance all costs until 8 months after the start of Phase 1 commercial production.

The financial analysis was performed using estimates of Capital Expenditures ("CAPEX") and Operating Expenditures ("OPEX") previously described in this report. Since the financial analysis is based on a cash flow estimate, it should be expected that actual financial results will vary from these predictions.

Details of the assumptions on which the analysis is based and results of the sensitivity analysis are provided in the following sections.

22.2 Methodology

The analyses were performed using estimates of CAPEX, OPEX, sustaining capital, a construction schedule, a production schedule, and estimates of future iron ore prices provided as mentioned in previous sections of this report.

By feeding estimates and assumptions into a financial model constructed in Microsoft Excel, cash flow calculations were developed. The Internal Rate of Returns (IRR) were calculated according to the discounted cash flow methodology, and sensitivity analyses were undertaken. The timing of the cash flows was assumed to take place at the end of each year. The financial model covers approximately two construction periods, (Phase 1 and 2), totalling 13.1 years, and 29.7 years of production starting in year 7. All capital and operating cost estimates are expressed in constant 2014 dollars, i.e. no inflation is taken into account and all the results are calculated on a real dollar basis.

Note: For the purposes of the financial analysis, actual calendar years are shown in the graphs and tables.

To be consistent with the previous sections of this report the dates should be interpreted as per following Table 22.2:

Table 22.2 – Calendar Date Cross-References

Year	Calendar Year
Year -6	May 2016 – April 2017
Year -5	May 2017 – April 2018
Year -4	May 2018 – April 2019
Year -3	May 2019 - April 2020
Year -2	May 2020 - April 2021
Year -1	May 2021 – April 2022
Year 1	May 2022 – April 2023
Year 10	May 2031 – April 2032
Year 20	May 2041 – April 2042
Year 30	May 2051 – April 2052

22.3 Summary of General Input Data

The financial analysis was carried out using the following assumptions for the base case:

- All amounts are expressed in real 2014 US dollars;
- IRRs are estimated using the discounted cash flow methodology;
- The total project life is approximately 36 years, i.e. 29.7 years of operation (from 2022 to 2051) overlapping the engineering and construction period of Phase 2 covering about 9 years (from Q1 2016 to end of 2024);
- Estimates of the applicable provincial and federal taxes in Canada were considered in the calculation of the IRR (after tax). These estimates are based on external consultant recommendations;

- The original claim holder obtains a royalty of 1.25 % of all consideration received from sale of iron ore concentrates from the Lac Otelnuk project;
- The original claim holder obtains an advance royalty of CAD 450,000 from the Lac Otelnuk project payable each November until the Commencement of Commercial Production;
- The analysis was performed on the basis that the project will be financed with equity only;
- No escalation was considered over the life of the project;
- Payables (construction costs and sustaining capital costs) are deemed to be paid 1-month after they are incurred. Receivables and Operating costs are deemed to have been received and paid when they are incurred;
- Closure costs of USD 520 M was modelled by leaving USD 520 M in LOM at the end of the operation period to cover the expenses related to mine closure over a 24 month period. The bond for the closure cost is part of the owner's cost as shown in Section 21 secured at the beginning of the project;
- Two market scenarios are considered for future distribution of the iron ore product produced. The base case market scenario is defined as 100 % of the product being shipped to China; while in the alternative scenario, 80 % of the product is shipped to China and 20 % to Europe
- US 28.45 $/DMT of shipping costs to China and US 12.45 $/DMT of shipping costs to Europe;
- The Life of Mine ("LoM") is assumed to be 29.7 years in the base case analysis;
- Iron ore concentrate grade of 68.5 % and a moisture content of 8 %.
- Equity distributions start in 2024 when the mine is producing 30 Mt/y.

22.3.1 Revenues

All revenues have been based on prices provided in the 'Market Study Lac Otelnuk Project' prepared by SNL Metals & Mining Ltd. for Lac Otelnuk Mining Ltd. on March 24th, 2015. The report provides estimated and forecasted Lac Otelnuk concentrate quality and freight premiums in both European and Asian basins. As the study forecasts prices up until 2050, prices are assumed to remain the same beyond 2050. The base case scenario assumes that 100 % of the product is shipped to China based on the average of the Low and High price alternatives of the product. The alternative scenario discussed later in this section considers that 80% of the iron ore concentrate is shipped to China and 20% to Europe based on the average of the Low and High price alternative of the product as well.

Table 22.3 provides the yearly iron ore prices that were used in the preparation of the financial model and Table 22.4 provides the calculations for the Netback selling price.

Table 22.3 – Iron Ore Price – Shipment to China and Europe

Iron Ore Prices to China

Year	High	Low	Average
2016	62.55	52.55	57.55
2017	72.55	62.55	67.55
2018	82.55	72.55	77.55
2019	87.55	77.55	82.55
2020	92.55	82.55	87.55
2021	102.55	87.55	95.05
2022	107.55	92.55	100.05
2023	112.55	96.55	104.55
2024	114.55	95.55	105.05
2025	115.55	95.55	105.55
2026	116.55	94.55	105.55
2027	117.55	94.55	106.05
2028	118.55	93.55	106.05
2029	119.55	93.55	106.55
2030	121.55	92.55	107.05
2031	121.55	92.55	107.05
2032	121.55	91.55	106.55
2033	121.55	91.55	106.55
2034	121.55	90.55	106.05
2035	121.55	90.55	106.05
2036	121.55	89.55	105.55
2037	121.55	89.55	105.55
2038	121.55	88.55	105.05
2039	121.55	88.55	105.05
2040	121.55	87.55	104.55
2041	121.55	87.55	104.55
2042	121.55	87.55	104.55
2043	121.55	86.55	104.05
2044	121.55	86.55	104.05
2045	121.55	85.55	103.55
2046	121.55	85.55	103.55
2047	121.55	84.55	103.05
2048	121.55	84.55	103.05
2049	121.55	83.55	102.55
2050	121.55	83.55	102.55
2051	121.55	83.55	102.55
2052	121.55	83.55	102.55
2053	121.55	83.55	102.55
2054	121.55	83.55	102.55
2055	121.55	83.55	102.55
2056	121.55	83.55	102.55
2057	121.55	83.55	102.55

Iron Ore Prices to Europe

Year	High	Low	Average
2016	67.02	57.02	62.02
2017	77.02	67.02	72.02
2018	87.02	77.02	82.02
2019	92.02	82.02	87.02
2020	97.02	87.02	92.02
2021	107.02	92.02	99.52
2022	112.02	97.02	104.52
2023	117.02	101.02	109.02
2024	119.02	100.02	109.52
2025	120.02	100.02	110.02
2026	121.02	99.02	110.02
2027	122.02	99.02	110.52
2028	123.02	98.02	110.52
2029	124.02	98.02	111.02
2030	126.02	97.02	111.52
2031	126.02	97.02	111.52
2032	126.02	96.02	111.02
2033	126.02	96.02	111.02
2034	126.02	95.02	110.52
2035	126.02	95.02	110.52
2036	126.02	94.02	110.02
2037	126.02	94.02	110.02
2038	126.02	93.02	109.52
2039	126.02	93.02	109.52
2040	126.02	92.02	109.02
2041	126.02	92.02	109.02
2042	126.02	92.02	109.02
2043	126.02	91.02	108.52
2044	126.02	91.02	108.52
2045	126.02	90.02	108.02
2046	126.02	90.02	108.02
2047	126.02	89.02	107.52
2048	126.02	89.02	107.52
2049	126.02	88.02	107.02
2050	126.02	88.02	107.02
2051	126.01	88.02	107.02
2052	126.01	88.02	107.02
2053	126.01	88.02	107.02
2054	126.01	88.02	107.02
2055	126.01	88.02	107.02
2056	126.01	88.02	107.02
2057	126.01	88.02	107.02

Table 22.4 – Netback Calculation as Provided in the 'Market Study Lac Otelnuk Project

	2015	2020	2025	2030	2040	2050
Low, MBIOI 62 %	55	85	98	95	90	86
High, MBIOI 62 %	65	95	118	124	124	124
Theoretical comparator values						
Low						
Alexandria	43.47	73.47	86.47	83.47	78.47	74.47
New Orleans	41.62	71.62	84.62	81.62	76.62	72.62
High						
Alexandria	53.47	83.47	106.47	112.47	112.47	112.47
New Orleans	51.62	81.62	104.62	110.62	110.62	110.62
Lac Otelnuk						
Freight Pointe Noire-Alexandria, $/t wet	11.46	11.46	11.46	11.46	11.46	11.46
Freight Pointe Noire-New Orleans, $/t wet	7.35	7.35	7.35	7.35	7.35	7.35
Freight Pointe Noire-Qingdao, $/t wet	26.17	26.17	26.17	26.17	26.17	26.17
Humidity	8 %	8 %	8 %	8 %	8 %	8 %
Freight Pointe Noire-Alexandria, $/t dry	12.45	12.45	12.45	12.45	12.45	12.45
Freight Pointe Noire-New Orleans, $/t dry	7.99	7.99	7.99	7.99	7.99	7.99
Freight Pointe Noire-Qingdao, $/t dry	28.45	28.45	28.45	28.45	28.45	28.45
Netback Lac Otelnuk						
Fe grade premium, $/dry tonne	26	26	26	26	26	26
Low price alternative, FOB Pointe Noire						
Pointe Noire-Alexandria	57.02	87.02	100.02	97.02	92.02	88.02
Pointe Noire-New Orleans	59.63	89.63	102.63	99.63	94.63	90.63
Pointe Noire-Qingdao	52.55	82.55	95.55	92.55	87.55	83.55
High price alternative, FOB Pointe Noire						
Pointe Noire-Alexandria	67.02	97.02	120.02	126.02	126.01	126.01
Pointe Noire-New Orleans	69.63	99.63	122.63	128.63	128.63	128.63
Pointe Noire-Qingdao	62.55	92.55	115.55	121.55	121.55	121.55

22.3.2 Operating Expenditures

The breakdown of the major operating costs for selected years of operation in the base case LoM is as per following Table 22.5:

Table 22.5 – Operating Costs for Selected Years (In Constant 2014 USD)

Production Year	**2**	**12**	**22**
Mine Operating Expenditure ($)	317,266,866	579,948,692	539,322,813
ROM Operating Expenditure ($)	19,793,700	39,634,801	38,850,608
Concentrator Operating Expenditure ($)	384,733,964	756,536,826	752,584,816
TMF Operating Expenditure	12,802,979	43,162,535	46,487,884
G&A and Infrastructures Operating Expenditure ($)	76,631,889	89,922,785	103,441,630
Port Operating Expenditure ($)	21,547,715	38,722,466	38,708,970
Slurry Transport Expenditure ($)	40,267,457	70,189,346	70,343,257
TOTAL ($)	**873,044,570**	**1,618,117,451**	**1,589,739,978**
Cost per Tonne of Concentrate ($/t)	**34.00**	**32.37**	**31.79**

Closure costs are estimated at US 520 M$. The payment of these expenses was modelled by leaving US 520 M$ in Project Co. at the end of the operation period to cover the expenses over a 24-month period.

Sustaining CAPEX totals US 1.68 B$ over the LoM.

22.3.3 Capital Expenditures

The total project cost is 14.558 B in constant US dollars of which 14.186 B is related to total construction costs, and 0.372 B of additional costs funded by equity during the first 8 months of operation (0.352 B of operational costs, 0.010 B of Sustaining Capital and 0.010 B of royalties). The construction cost is based on an Engineering-Procurement-Construction Management (EPCM) execution strategy, not a fixed price contract.

Working Capital:

> The project draws on equity to cover all expenses until the first 8 months from the start of commercial production. The revenues of the first 8 months are used to build up working capital (cash) and are not used to pay any expenses. After this initial period of 8 months, the project becomes a self-funded project and no longer relies on equity to fund any costs.

Table 22.6 below demonstrates the sources and uses of funds until the Project becomes a self-funded project.

Table 22.6 – Sources and Uses of Funds until Project Becomes Self-Funding

TIMING								
End of period	Date	2016/12/31	2017/12/31	2018/12/31	2019/12/31	2020/12/31	2021/12/31	2022/12/31
PROJECT SOURCES AND USES								
Sources								
Revenue	US$	0	0	0	0	0	0	1,002,846,008
Construction Loan	US$	0	0	0	0	0	0	0
Long Term Refinance Debt 1	US$	0	0	0	0	0	0	0
Long Term Refinance Debt 2	US$	0	0	0	0	0	0	0
Equity	US$	155,336,438	921,356,322	1,982,674,131	2,487,181,862	2,230,022,017	1,265,693,956	680,696,586
Total Sources	US$	155,336,438	921,356,322	1,982,674,131	2,487,181,862	2,230,022,017	1,265,693,956	1,683,542,594
Uses								
Construction Cost - Phase 1	US$	154,940,438	920,960,322	1,982,278,131	2,486,785,862	2,229,626,017	1,230,431,363	345,406,155
Construction Cost - Phase 2	US$	0	0	0	0	0	0	0
Operating Costs	US$	0	0	0	0	0	34,866,593	317,828,103
Sustaining Capital	US$	0	0	0	0	0	0	10,003,995
Royalties	US$	396,000	396,000	396,000	396,000	396,000	396,000	7,458,333
Net Sales Tax	US$	0	0	0	0	0	0	0
Taxes	US$	0	0	0	0	0	0	13,611,178
Construction Loan - Principal Repayment	US$	0	0	0	0	0	0	0
Construction Loan - Interest	US$	0	0	0	0	0	0	0
Long Term Refinance Debt 1 - Principal Repayment	US$	0	0	0	0	0	0	0
Long Term Refinance Debt 1 - Interest	US$	0	0	0	0	0	0	0
Long Term Refinance Debt 2 - Principal Repayment	US$	0	0	0	0	0	0	0
Long Term Refinance Debt 2 - Interest	US$	0	0	0	0	0	0	0
Financial Advisor Fee	US$	0	0	0	0	0	0	0
Arrangement fee (Upfront fee)	US$	0	0	0	0	0	0	0
Commitment fee	US$	0	0	0	0	0	0	0
Reserves and Mine Closure Account	US$	0	0	0	0	0	0	0
Shareholder Distributions	US$	0	0	0	0	0	0	0
Cash	US$	0	0	0	0	0	0	989,234,830
	US$	155,336,438	921,356,322	1,982,674,131	2,487,181,862	2,230,022,017	1,265,693,956	1,683,542,594

22.3.4 Taxes

The financial model reflects the impact of Canadian and Quebec fiscal regimes and applicable taxes as listed below.

- Corporate income taxes
  - A 15 % federal tax rate and 11.9 % provincial tax rate is applied to taxable income;
  - Tax depreciation on asset class 41.2 – applied on a 25 % declining-balance basis starting in the year incurred;
  - Tax depreciation on asset class 41b – applied on a 25 % declining-balance basis starting in the year incurred;
  - Tax depreciation on Canadian Development Expenses (CDE) – applied on a 30 % declining-balance basis starting in the year incurred;
  - Commitment fees and upfront fees are depreciated over 5 years starting in the year incurred;
  - Loss carry-forward – applied against taxable income, but unused losses are lost after 20 years from the time they are recognised.
- Mining taxes
  - 16 % tax rate applicable to the portion of the operator's annual profit, for the fiscal year, attributable to the first segment of profit margin, i.e. 35 % or, if it is lesser, the operator's profit margin for the fiscal year;
  - 22 % tax rate applicable to the portion of the operator's annual profit, for the fiscal year, attributable to the second segment of profit margin, if any, i.e. 15 % or, if it is lesser, the amount by which the operator's profit margin for the fiscal year exceeds 35 %;
  - 28 % tax rate applicable to the portion of the operator's annual profit, for the fiscal year, attributable to the third segment of profit margin, if any , i.e. the amount by which the operator's profit margin for the fiscal year exceeds 50 %;
  - Tax depreciation on asset class 4 – applied on a 30 % declining-balance basis starting in the year of positive taxable income;
  - Tax depreciation on asset class 4A – applied on a 30 % declining-balance basis starting in the year of positive taxable income;
  - Minimum mining tax: a 1 % tax rate to the first CAD 80 million of the 'operator's output value at the mine shaft head' and a 4 % tax rate to the amount corresponding to the excess of the operator's output value at the mine shaft head, over an amount of CAD 80 million;
  - No Loss carry-forward permitted;
  - Additional Allowance for a mine situated in Northern Quebec: CAD 5.0 M limit available for a 36 month period following the start of commercial production;

- An annual processing allowance equal to 8% of the cumulative cost of processing assets, limited in any year to the greater of 75% of the income before the processing allowance or 10% of the output value at the mine shaft head before the deduction of the processing allowance has been included in the calculations.
- Processing Allowance as well as Federal and Mining taxes have been modelled as per the tax memo dated October 29, 2014 from Deloitte.

- Unused depreciation balances are used in the last operating year of the mine.

22.4 Financing

Analyses were carried out on the basis that 100 % of the project costs until the project becomes self-funded be financed by equity.

The following Table 22.7 presents a summary of the different sources of funds:

Table 22.7 – Sources of Financing (In Constant USD)

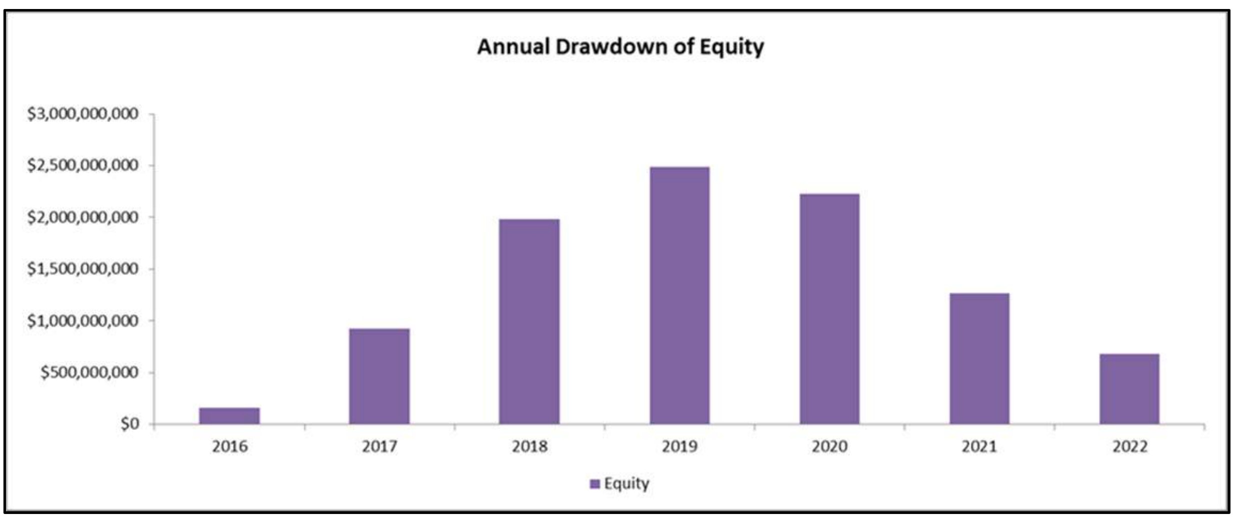
Sources of Capital	Amount Invested
Senior Debt	0.00 billion
Equity	9.72 billion
Total	**9.72 billion**

Financing assumptions were provided by Lac Otelnuk Mining Ltd. as detailed below:

- Gearing: 100 % equity;
- Interest income on Cash Balances: 1.0 %.

The projected draw schedule during the construction period is shown on Figure 22.1.

Figure 22.1 – Projected Draw Schedule Until Project Becomes Self-Funding



22.5 Results

Table 22.8 below provides a summary of the results of the economic and financial analysis for the base case (29.7 years operation, 100 % production to China):

Table 22.8 – Base Case Nominal 30-Year Mine Life (Actual Analysis on 29.7 Years) (100% Production to China)

	Before Taxes	After Taxes
Project IRR*	15.8 %	13.0 %
NPV @ 6 %*	$ 17,457 million	$ 9,647 million
NPV @ 8 %*	$ 10,388 million	$ 5,240 million
NPV @ 10 %*	$ 5,906 million	$ 2,440 million
Payback Period**	7.0 years	7.3 years

* Based on Free Cash Flow to Equity.
** Calculated from start of commercial production and based on Free Cash flow to Equity.
All monetary values in USD, unless otherwise stated.

22.5.1 Overall Cash Flows

Table 22.9 below presents the revenue and disbursements of the project for all years (in constant dollars). Note that these are based on calendar years, not operating years.

Table 22.9 – Project Cash Flows for all Years (in 2014 USD)

TIMING											
End of period	Date		2016/12/31	2017/12/31	2018/12/31	2019/12/31	2020/12/31	2021/12/31	2022/12/31	2023/12/31	2024/12/31
CASH WATERFALL											
Revenue											
Iron Ore to China	US$		0	0	0	0	0	0	1,000,500,000	2,613,750,000	3,151,500,000
Iron Ore to Europe	US$		0	0	0	0	0	0	0	0	0
Interest Income	US$		0	0	0	0	0	0	2,346,008	17,671,228	37,662,399
Taxes											
Quebec Mining Tax	US$		0	0	0	0	0	0	(13,611,178)	(13,881,805)	(20,546,985)
Federal Income Tax	US$		0	0	0	0	0	0	0	0	0
Quebec Income Tax	US$		0	0	0	0	0	0	0	0	0
Operating Costs											
Operating Costs	US$		0	0	0	0	0	(34,866,593)	(317,828,103)	(728,491,409)	(976,070,014)
Sustaining Capital Costs											
Sustaining Capital Costs	US$		0	0	0	0	0	0	(10,003,995)	(24,217,676)	(42,289,824)
Royalties											
Advance royalties	US$		(396,000)	(396,000)	(396,000)	(396,000)	(396,000)	(396,000)	0	0	0
Royalty on Revenues (FOB Sept-Iles)	US$		0	0	0	0	0	0	(7,458,333)	(22,907,475)	(28,696,692)
Sources of Funding											
Equity	US$		155,336,438	921,356,322	1,982,674,131	2,487,181,862	2,230,022,017	1,265,693,956	680,696,586	0	0
Project Capital Cost											
Construction Cost - Phase 1	US$		(154,940,438)	(920,960,322)	(1,982,278,131)	(2,486,785,862)	(2,229,626,017)	(1,230,431,363)	(345,406,155)	(25,617,633)	(8,019,489)
Construction Cost - Phase 2	US$		0	0	0	0	0	0	0	(10,597,393)	(160,821,039)
Reserve Movement											
Debt Service Reserve Account	US$		0	0	0	0	0	0	0	0	0
Closure Account	US$		0	0	0	0	0	0	0	0	0
Shareholder Distributions											
Shareholder Dividends	US$		0	0	0	0	0	0	0	0	(4,534,560,628)
Change in Cash Account											
Opening Balance	US$		0	0	0	0	0	0	0	989,234,830	2,794,942,665
Change in Cash	US$		0	0	0	0	0	0	989,234,830	1,805,707,836	(2,581,842,272)
Closing Balance	US$	0	0	0	0	0	0	0	989,234,830	2,794,942,665	213,100,393

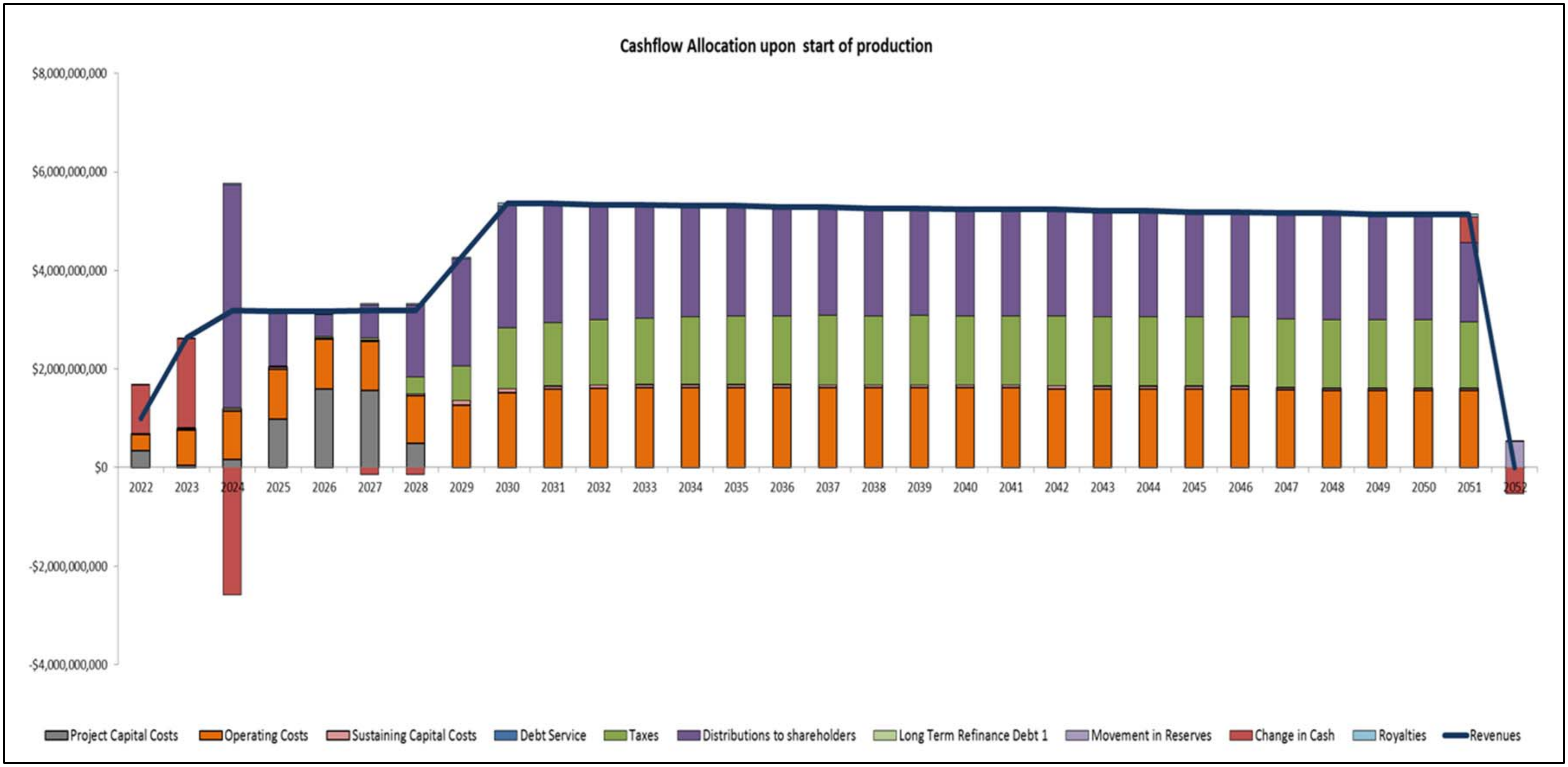
TIMING											
End of period	Date		2025/12/31	2026/12/31	2027/12/31	2028/12/31	2029/12/31	2030/12/31	2031/12/31	2032/12/31	2033/12/31
CASH WATERFALL											
Revenue											
Iron Ore to China	US$		3,166,500,000	3,166,500,000	3,181,500,000	3,181,500,000	4,262,000,000	5,352,500,000	5,352,500,000	5,327,500,000	5,327,500,000
Iron Ore to Europe	US$		0	0	0	0	0	0	0	0	0
Interest Income	US$		4,484,210	3,765,130	3,668,356	4,130,338	5,679,832	7,744,165	7,626,298	7,520,233	7,507,876
Taxes											
Quebec Mining Tax	US$		(22,718,364)	(21,561,020)	(21,466,021)	(24,038,163)	(124,210,582)	(374,945,943)	(400,994,167)	(419,223,902)	(437,577,376)
Federal Income Tax	US$		0	0	(12,188,835)	(173,952,217)	(324,452,500)	(478,951,729)	(490,310,250)	(499,310,071)	(510,649,151)
Quebec Income Tax	US$		0	0	(9,669,809)	(138,002,092)	(257,398,984)	(379,968,372)	(388,979,465)	(396,119,323)	(405,114,993)
Operating Costs											
Operating Costs	US$		(1,020,377,511)	(1,005,188,449)	(989,624,204)	(969,470,029)	(1,269,581,297)	(1,511,446,146)	(1,584,634,782)	(1,611,821,996)	(1,618,117,451)
Sustaining Capital Costs											
Sustaining Capital Costs	US$		(42,211,874)	(32,121,254)	(30,417,877)	(31,689,867)	(91,266,521)	(92,597,503)	(70,959,015)	(68,415,258)	(62,756,213)
Royalties											
Advance royalties	US$		0	0	0	0	0	0	0	0	0
Royalty on Revenues (FOB Sept-Iles)	US$		(28,896,875)	(28,912,500)	(29,084,375)	(29,100,000)	(37,407,292)	(49,098,958)	(49,125,000)	(48,838,542)	(48,812,500)
Sources of Funding											
Equity	US$		0	0	0	0	0	0	0	0	0
Project Capital Cost											
Construction Cost - Phase 1	US$		(169,620)	0	0	0	0	0	0	0	0
Construction Cost - Phase 2	US$		(975,908,979)	(1,596,193,524)	(1,563,748,011)	(494,395,683)	(464,250)	0	0	0	0
Reserve Movement											
Debt Service Reserve Account	US$		0	0	0	0	0	0	0	0	0
Closure Account	US$		0	0	0	0	0	0	0	0	0
Shareholder Distributions											
Shareholder Dividends	US$		(1,051,801,576)	(443,972,136)	(669,143,278)	(1,468,660,037)	(2,163,362,657)	(2,473,235,514)	(2,375,123,620)	(2,291,291,142)	(2,251,980,193)
Change in Cash Account											
Opening Balance	US$		213,100,393	241,999,805	284,316,053	144,142,000	464,250	0	0	0	0
Change in Cash	US$		28,899,412	42,316,248	(140,174,053)	(143,677,750)	(464,250)	(0)	0	(0)	(0)
Closing Balance	US$	0	241,999,805	284,316,053	144,142,000	464,250	0	0	0	0	(0)

TIMING											
End of period	Date		2034/12/31	2035/12/31	2036/12/31	2037/12/31	2038/12/31	2039/12/31	2040/12/31	2041/12/31	2042/12/31
CASH WATERFALL											
Revenue											
Iron Ore to China	US$		5,302,500,000	5,302,500,000	5,277,500,000	5,277,500,000	5,252,500,000	5,252,500,000	5,227,500,000	5,227,500,000	5,227,500,000
Iron Ore to Europe	US$		0	0	0	0	0	0	0	0	0
Interest Income	US$		7,456,251	7,456,359	7,412,862	7,407,000	7,356,920	7,357,028	7,313,422	7,305,510	7,333,592
Taxes											
Quebec Mining Tax	US$		(446,417,914)	(456,040,666)	(457,870,342)	(459,971,296)	(457,058,553)	(459,368,976)	(456,080,022)	(457,212,129)	(462,062,467)
Federal Income Tax	US$		(516,191,279)	(523,124,560)	(524,620,936)	(526,705,897)	(524,963,965)	(527,087,135)	(524,976,277)	(526,170,400)	(529,847,029)
Quebec Income Tax	US$		(409,511,748)	(415,012,151)	(416,199,276)	(417,853,345)	(416,471,412)	(418,155,794)	(416,481,180)	(417,428,517)	(420,345,310)
Operating Costs											
Operating Costs	US$		(1,618,117,451)	(1,618,117,451)	(1,618,117,451)	(1,615,230,786)	(1,613,787,454)	(1,613,787,454)	(1,613,787,454)	(1,613,787,454)	(1,597,755,804)
Sustaining Capital Costs											
Sustaining Capital Costs	US$		(62,756,213)	(62,756,213)	(62,756,213)	(64,246,563)	(65,311,098)	(65,311,098)	(65,311,098)	(65,311,098)	(64,640,666)
Royalties											
Advance royalties	US$		0	0	0	0	0	0	0	0	0
Royalty on Revenues (FOB Sept-Iles)	US$		(48,526,042)	(48,500,000)	(48,213,542)	(48,187,500)	(47,901,042)	(47,875,000)	(47,588,542)	(47,562,500)	(47,562,500)
Sources of Funding											
Equity	US$		0	0	0	0	0	0	0	0	0
Project Capital Cost											
Construction Cost - Phase 1	US$		0	0	0	0	0	0	0	0	0
Construction Cost - Phase 2	US$		0	0	0	0	0	0	0	0	0
Reserve Movement											
Debt Service Reserve Account	US$		0	0	0	0	0	0	0	0	0
Closure Account	US$		0	0	0	0	0	0	0	0	0
Shareholder Distributions											
Shareholder Dividends	US$		(2,208,435,606)	(2,186,405,318)	(2,157,135,103)	(2,152,711,613)	(2,134,363,397)	(2,128,271,571)	(2,110,588,850)	(2,107,333,412)	(2,112,619,817)
Change in Cash Account											
Opening Balance	US$		(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)
Change in Cash	US$		0	0	0	0	(0)	(0)	0	0	0
Closing Balance	US$	0	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)

TIMING												
End of period	Date		2043/12/31	2044/12/31	2045/12/31	2046/12/31	2047/12/31	2048/12/31	2049/12/31	2050/12/31	2051/12/31	2052/12/31
CASH WATERFALL												
Revenue												
Iron Ore to China	US$		5,202,500,000	5,202,500,000	5,177,500,000	5,177,500,000	5,152,500,000	5,152,500,000	5,127,500,000	5,127,500,000	5,127,500,000	0
Iron Ore to Europe	US$		0	0	0	0	0	0	0	0	0	0
Interest Income	US$		7,306,466	7,314,594	7,254,948	7,255,055	7,247,109	7,295,299	7,235,675	7,235,782	9,901,912	448,242
Taxes												
Quebec Mining Tax	US$		(459,739,881)	(460,128,194)	(455,493,762)	(455,684,036)	(451,863,127)	(452,432,435)	(447,591,449)	(447,637,134)	(447,669,113)	0
Federal Income Tax	US$		(528,231,207)	(528,766,318)	(525,464,508)	(525,765,482)	(523,152,807)	(524,131,645)	(521,026,965)	(521,475,795)	(494,150,534)	0
Quebec Income Tax	US$		(419,063,424)	(419,487,946)	(416,868,510)	(417,107,282)	(415,034,560)	(415,811,105)	(413,348,059)	(413,704,131)	(392,026,090)	0
Operating Costs												
Operating Costs	US$		(1,589,739,978)	(1,589,739,978)	(1,589,739,978)	(1,589,739,978)	(1,574,475,901)	(1,566,843,862)	(1,566,843,862)	(1,566,843,862)	(1,566,843,862)	0
Sustaining Capital Costs												
Sustaining Capital Costs	US$		(64,161,785)	(64,161,785)	(64,161,785)	(64,161,785)	(54,103,416)	(46,918,866)	(46,918,866)	(46,918,866)	(46,918,866)	(3,909,906)
Royalties												
Advance royalties	US$		0	0	0	0	0	0	0	0	0	0
Royalty on Revenues (FOB Sept-Iles)	US$		(47,276,042)	(47,250,000)	(46,963,542)	(46,937,500)	(46,651,042)	(46,625,000)	(46,338,542)	(46,312,500)	(46,312,500)	(3,859,375)
Sources of Funding												
Equity	US$		0	0	0	0	0	0	0	0	0	0
Project Capital Cost												
Construction Cost - Phase 1	US$		0	0	0	0	0	0	0	0	0	0
Construction Cost - Phase 2	US$		0	0	0	0	0	0	0	0	0	0
Reserve Movement												
Debt Service Reserve Account	US$		0	0	0	0	0	0	0	0	0	0
Closure Account	US$		0	0	0	0	0	0	0	0	0	(520,000,000)
Shareholder Distributions												
Shareholder Dividends	US$		(2,101,594,149)	(2,100,280,373)	(2,086,062,862)	(2,085,358,992)	(2,094,466,257)	(2,107,032,385)	(2,092,667,932)	(2,091,843,494)	(1,615,711,666)	(448,242)
Change in Cash Account												
Opening Balance	US$		(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	527,769,281
Change in Cash	US$		(0)	(0)	(0)	(0)	0	0	0	0	527,769,281	(527,769,281)
Closing Balance	US$	0	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	527,769,281	0

Figure 22.2 presents the annual cash flows and revenues over the 29.7-year operation period.

Figure 22.2 – Estimated (pro-forma) Cash Flows during Operation

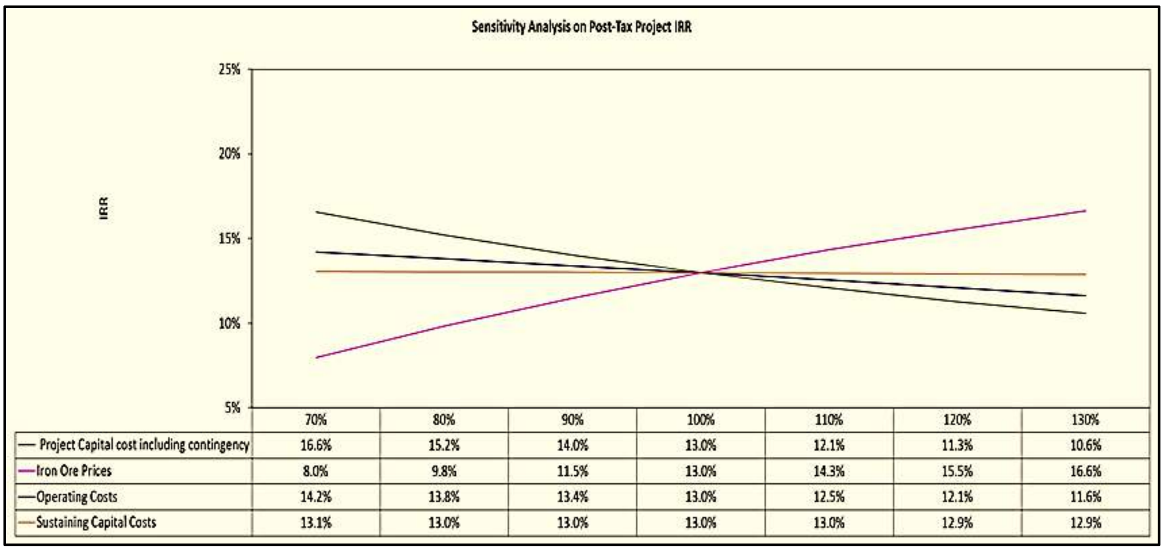


22.6 Sensitivity Analysis

The sensitivity of the after-tax Project IRR and NPV to changes in key variables is shown in Figure 22.3 and Figure 22.4 respectively. The after-tax Project IRR and NPV appear to be most sensitive to changes in the estimates of the iron ore prices and least sensitive to changes in sustaining capital costs. The discount rate used for the NPV sensitivities is 8 %. The selected input parameters used in the sensitivity analysis are:

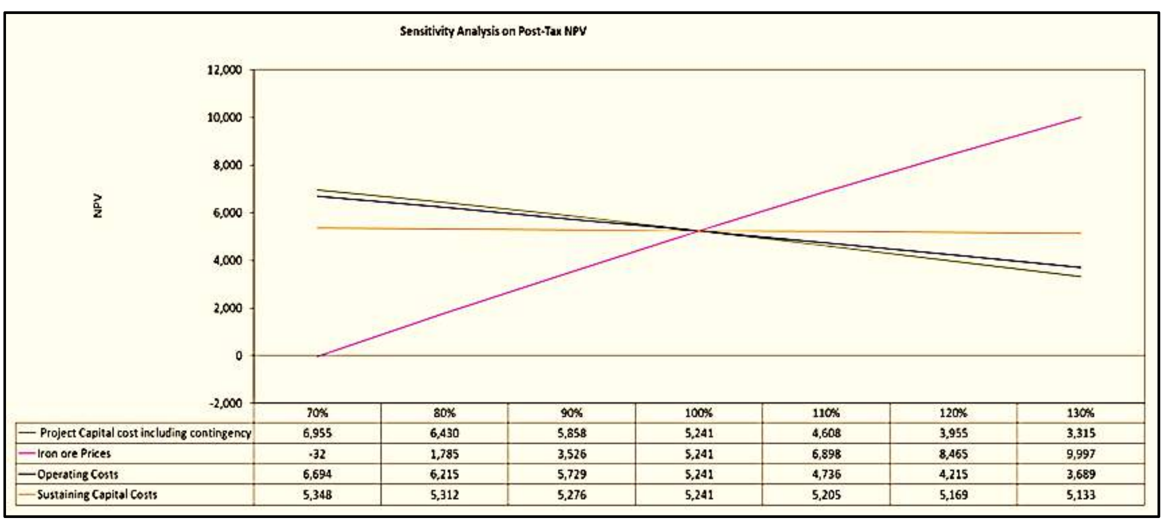
- Project Capital Costs
- Iron Ore Prices
- Operating Costs
- Sustaining Capital Costs

Figure 22.3 – Sensitivity of Project IRR (After Tax) to Key Parameters (Base Case)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	16.6%	15.2%	14.0%	13.0%	12.1%	11.3%	10.6%
Iron Ore Prices	8.0%	9.8%	11.5%	13.0%	14.3%	15.5%	16.6%
Operating Costs	14.2%	13.8%	13.4%	13.0%	12.5%	12.1%	11.6%
Sustaining Capital Costs	13.1%	13.0%	13.0%	13.0%	13.0%	12.9%	12.9%

Figure 22.4 – Sensitivity of Project NPV (After Tax) to Key Parameters (Base Case)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	6,955	6,430	5,858	5,241	4,608	3,955	3,315
Iron ore Prices	-32	1,785	3,526	5,241	6,898	8,465	9,997
Operating Costs	6,694	6,215	5,729	5,241	4,736	4,215	3,689
Sustaining Capital Costs	5,348	5,312	5,276	5,241	5,205	5,169	5,133

22.7 Additional Scenarios

22.7.1 Alternative Scenario 1

The first alternative scenario analysed assumed 20 % of the production was shipped to Europe and the balance to China (29.7 years operation, 20 % of product to Europe and 80 % to China). The project is fully financed with equity. A summary of the results and the sensitivity of the after-tax Project IRR and NPV to changes in key variables are presented in Table 22.10, Figure 22.5 and Figure 22.6. The small improvement in project economics can be attributed to the higher iron ore prices in Europe. Similar to the base case, the after-tax Project IRR and NPV appear to be most sensitive to changes in the estimates of the iron ore prices and least sensitive to changes in sustaining capital costs. The discount rate used for the NPV sensitivities is 8%. The complete market study including prices for the European and Chinese markets can be found in Section 19.0 of this report.

Table 22.10 – Summary of Financial Indicators - Scenario 1 (Production 80 % China – 20 % Europe, 100 % Equity)

	Before Taxes	After Taxes
Project IRR*	15.9 %	13.1 %
NPV @ 6%*	$ 17,783 million	$ 9,839 million
NPV @ 8%*	$ 10,618 million	$ 5,378 million
NPV @ 10%*	$ 6,072 million	$ 2,542 million
Payback Period**	6.9 yrs	7.2 yrs

* Based on Free Cash Flow to Equity.
** Calculated from start of commercial production and based on Free Cash flow to Equity.
All monetary values in USD, unless otherwise stated.

Figure 22.5 – Sensitivity of Project IRR (After Tax) – Scenario 1 (Production 80 % China / 20 % Europe, 100 % Equity)

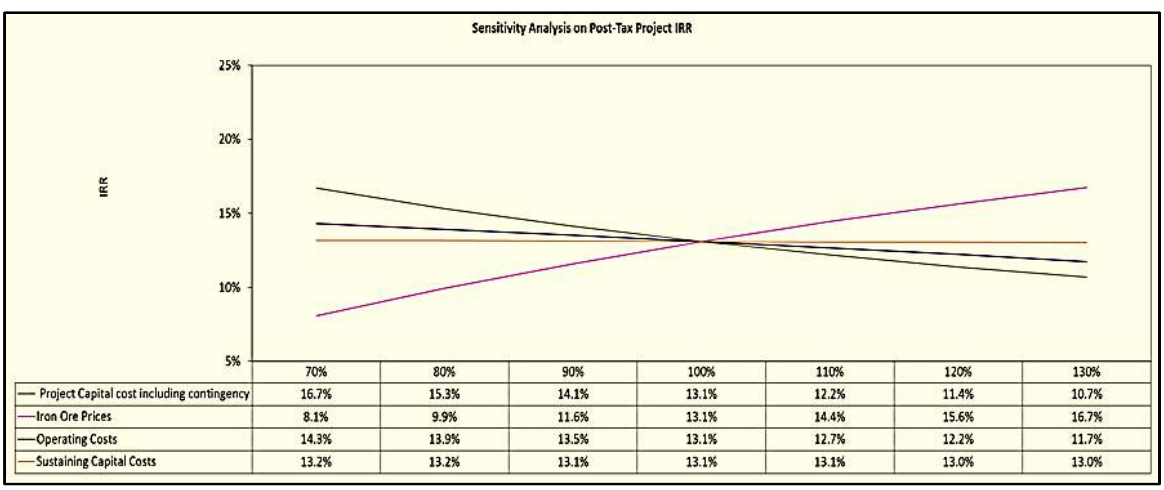
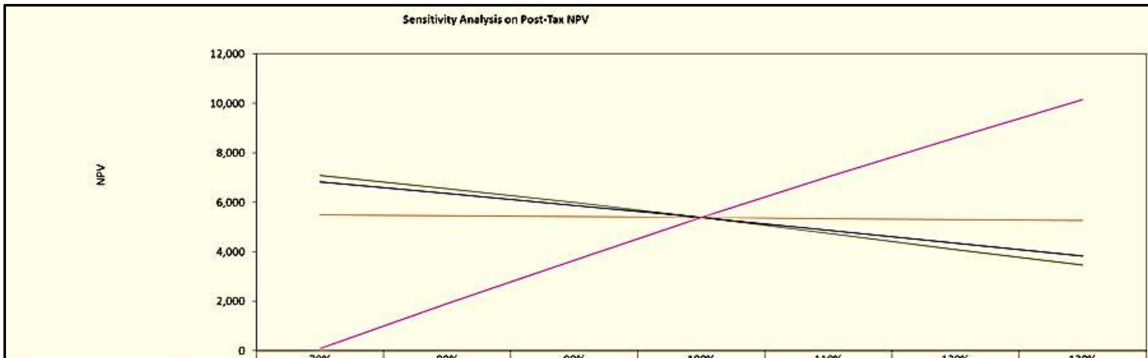


Figure 22.6 – Sensitivity of Project NPV (After Tax) to Key Parameters Alternative 1 – (80 % to China - 20 % to Europe, 100 % Equity)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	7,079	6,556	5,992	5,378	4,747	4,095	3,457
Iron ore Prices	79	1,898	3,653	5,378	7,034	8,614	10,159
Operating Costs	6,826	6,349	5,864	5,378	4,869	4,355	3,831
Sustaining Capital Costs	5,486	5,450	5,414	5,378	5,342	5,307	5,271

22.7.2 Alternative Scenario 2

A second alternative scenario was analysed where the project is financed with 70 % debt. In this case, the mine operates for 29.7 years with 100 % of the production shipped to China. Financing assumptions were provided by Lac Otelnuk Mining Ltd. as detailed below:

- Facility amount or gearing: 70 % debt and 30 % equity. The debt-equity gearing is calculated at 8 months from the start of commercial production. Debt and Equity are drawn to finance all costs up until 8 months after the start of commercial production;
- Interest rate (payable semi-annually) – Fixed annual rate of 5.5 % for the life of the loan + 2.0 % spread while the project continues to draw on the facility, after which the spread drops to 0.5 %;
- Repayment period starting in year 8 (from start of construction), with semi-annual principal repayment;
- Upfront fees: 0 %;
- Commitment fees: 0.5 %;
- Interest income on Debt Service Reserve Account: 2.0 %;
- Interest income on Cash Balances: 1.0 %;
- No cash sweep;
- Debt Service Reserve and any other reserves – 6 months interest and principal repayments;
- Min DSCR calculated: 1.52x. The DSCR is the ratio of the cash flow available for debt service to the debt service (principal repayment and interest) payable over the same period. Would the calculated DSCR fall below 1.0x, the borrower would be in default;

- Average DSCR calculated: 2.07x;
- The equity is injected parri-passu to the debt;
- Equity distributions start in 2026 when the mine is producing 30 Mt/y.

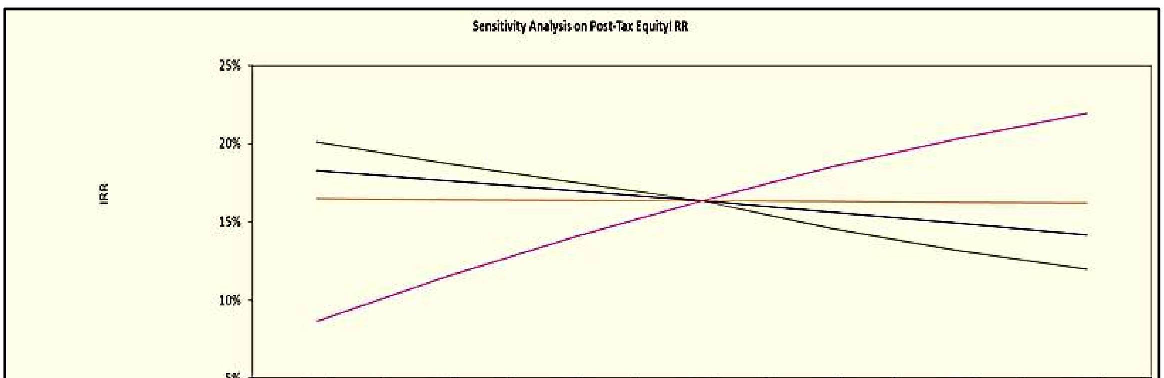
A summary of the results and the sensitivity of the after-tax equity IRR and NPV to changes in key variables are presented in the Table 22.11 and Figure 22.7 and Figure 22.8 respectively. The discount rate used for the NPV sensitivities is 8 %. The after tax Equity IRR and NPV appear to be most sensitive to changes in the estimates of the iron ore prices and least sensitive to changes in sustaining capital costs.

Table 22.11 – Summary of Financial Indicators -Scenario 2
(Production 100 % China, 30 % Equity / 70 % Debt)

	Before Taxes	After Taxes
Equity IRR*	19.7 %	16.4 %
NPV*	$ 10,174 million	$ 5,519 million
Payback Period*	7.2 yrs	7.5 yrs

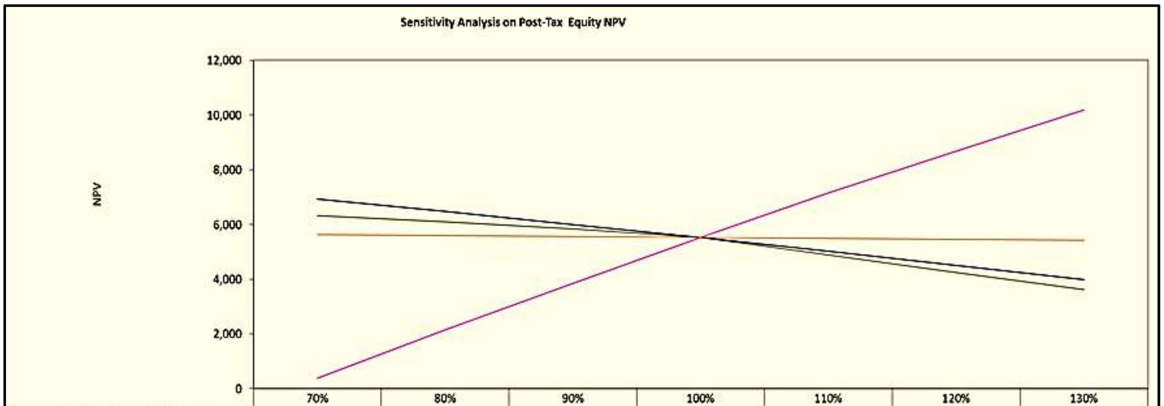
* Based on dividends to shareholders
** Calculated from start of commercial production and based on dividends to shareholders
All monetary values in USD, unless otherwise stated.
Project NPV discounted at 8 %.

Figure 22.7 – Sensitivity of Project IRR (After Tax) – Scenario 2
(Production 100 % to China, 30 % Equity / 70 % Debt)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	20.1%	18.8%	17.5%	16.4%	14.6%	13.2%	12.0%
Iron Ore Prices	8.6%	11.5%	14.0%	16.4%	18.5%	20.3%	21.9%
Operating Costs	18.3%	17.7%	17.0%	16.4%	15.7%	14.9%	14.2%
Sustaining Capital Costs	16.5%	16.4%	16.4%	16.4%	16.3%	16.3%	16.2%

Figure 22.8 – Sensitivity of Project NPV (After Tax) – Scenario 2
(Production 100 % to China, 30 % Equity / 70 % Debt)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	6,322	6,104	5,838	5,519	4,887	4,249	3,612
Iron ore Prices	384	2,148	3,846	5,519	7,152	8,687	10,182
Operating Costs	6,935	6,470	5,995	5,519	5,022	4,507	3,990
Sustaining Capital Costs	5,624	5,589	5,554	5,519	5,484	5,449	5,414

22.7.3 Alternative Scenario 3

A third alternative scenario was analysed where the project is financed with 70% debt using the same assumptions listed in Alternative Scenario 2. In this case, the mine operates for 29.7 years with 80% of the production shipped to China and 20% of the production shipped to Europe.

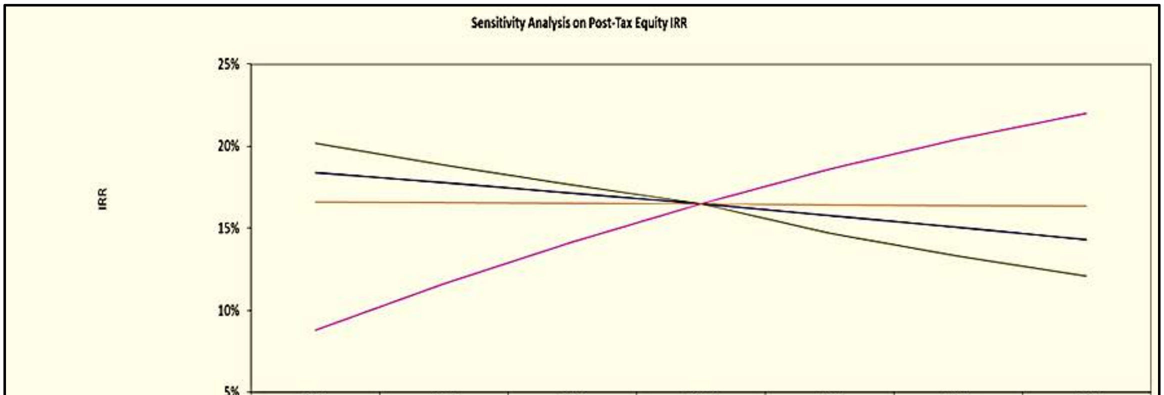
A summary of the results and the sensitivity of the after-tax equity IRR and NPV to changes in key variables are presented in Table 22.12 below and Figure 22.9 and Figure 22.10 respectively. The discount rate used for the NPV sensitivities is 8%. The after-tax Equity IRR and NPV appear to be most sensitive to changes in the estimates of the iron ore prices and least sensitive to changes in sustaining capital costs.

Table 22.12 – Summary of Financial Indicators - Scenario 3
(Production 80 % China – 20 % Europe, 30 % Equity / 70 % Debt)

	Before Taxes	After Taxes
Equity IRR*	19.8 %	16.5 %
NPV*	$ 10,393 million	$ 5,646 million
Payback Period**	7.2 yrs	7.4 yrs

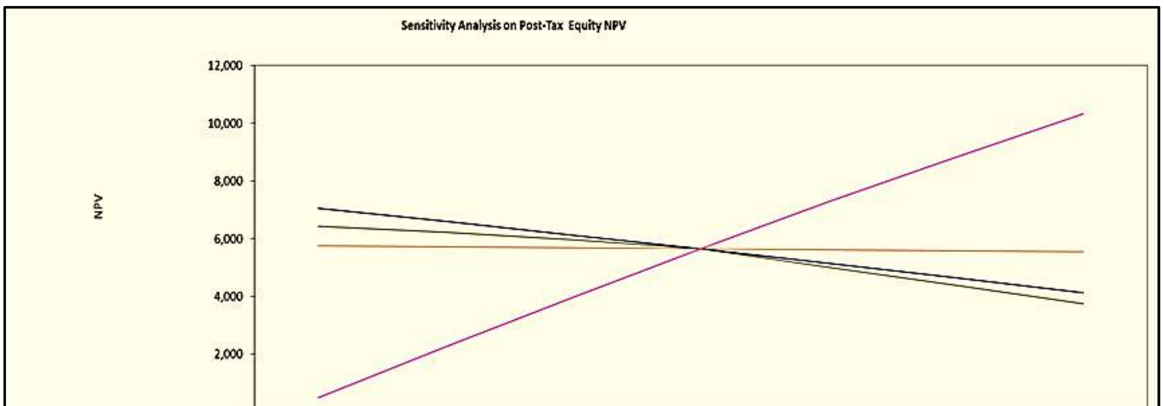
* Based on dividends to shareholders
** Calculated from start of commercial production and based on dividends to shareholders
All monetary values in USD, unless otherwise stated.
Project NPV discounted at 8 %.

Figure 22.9 – Sensitivity of Project IRR (After Tax) – Scenario 3 (Production 80 % to China – 20 % to Europe, 30 % Equity / 70 % Debt)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	20.2%	18.9%	17.6%	16.5%	14.7%	13.3%	12.1%
Iron Ore Prices	8.8%	11.6%	14.2%	16.5%	18.6%	20.4%	22.0%
Operating Costs	18.4%	17.8%	17.1%	16.5%	15.8%	15.1%	14.3%
Sustaining Capital Costs	16.6%	16.6%	16.5%	16.5%	16.4%	16.4%	16.4%

Figure 22.10 – Sensitivity of Project NPV (After Tax) – Scenario 3 (Production 80 % China – 20 % Europe, 30 % Equity / 70 % Debt)



	70%	80%	90%	100%	110%	120%	130%
Project Capital cost including contingency	6,431	6,216	5,959	5,646	5,014	4,376	3,740
Iron ore Prices	486	2,252	3,961	5,646	7,276	8,824	10,330
Operating Costs	7,056	6,593	6,118	5,646	5,141	4,636	4,120
Sustaining Capital Costs	5,750	5,715	5,681	5,646	5,611	5,575	5,540

23.0 ADJACENT PROPERTIES

A block of claims held by Kieran Prasad, Pickering, Ontario, occurs along the southwest limit, and in the central portion of the Property. One (1) claim owned by Bertrand Brassard, Lac-Kénogami, Quebec, is wedged between the Prasad claims and the claims of the Property. Contiguous along the southeastern portion of the Property, and extending southeastward, is a group of claims registered under New Millennium Iron Corp., Westmount, Quebec.

One (1) contiguous and one (1) non-contiguous block of claims owned by Northern Shield, Ottawa, Ontario, extend southeastward from the southeast extremity of the Property, and east of it.

A block of claims contiguous with the central portion of the Property on the northeast side is held by Kieran Prasad, as well as another contiguous block and a non-contiguous block on the east side of the northwestern extremity of the Property.

Focus Graphite Inc., Ottawa, Ontario, holds a few claims some distance to the north of the Property.

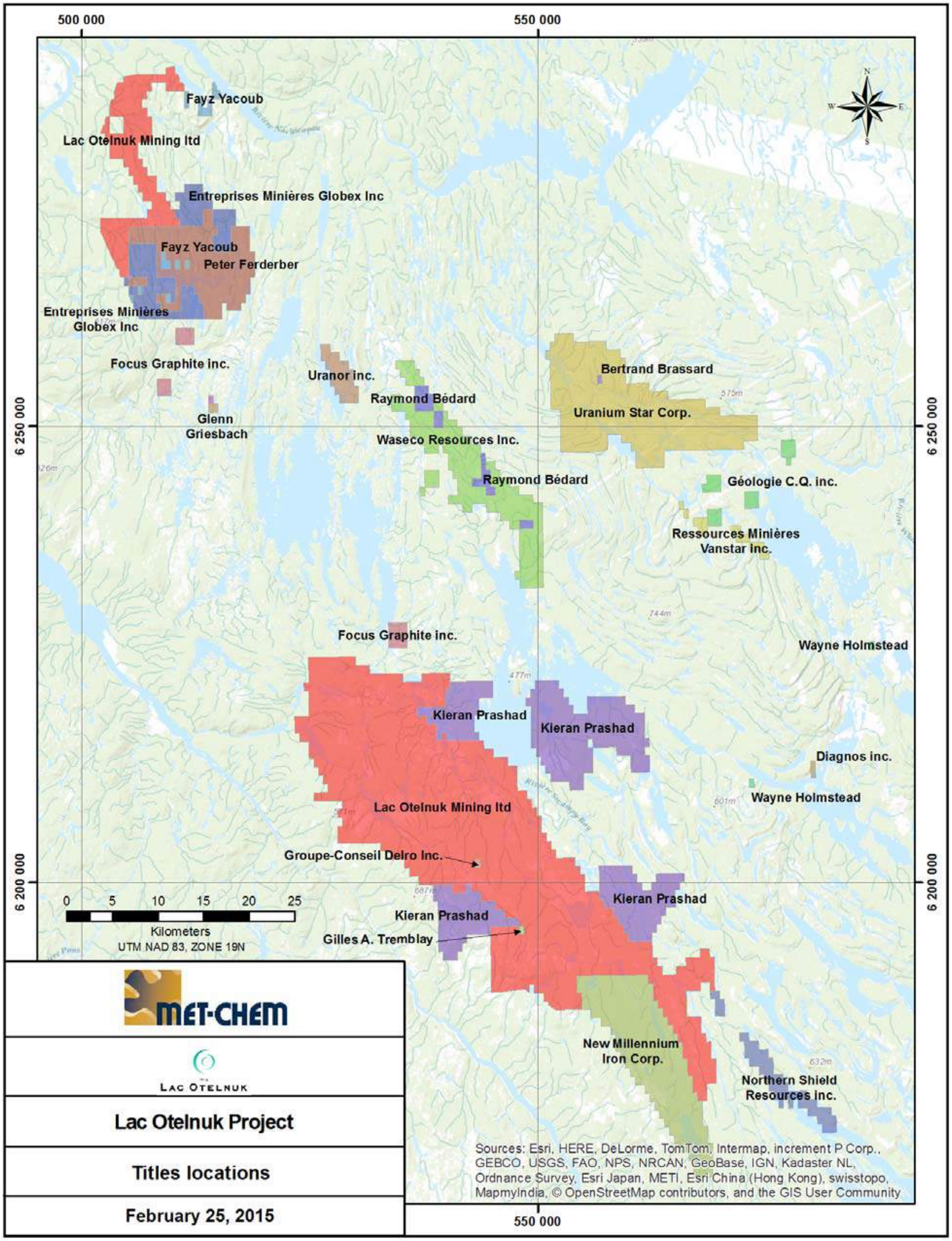
The south boundary of LOM's December Lake property lies some 47 km to the northwest of the northern extremity of the Otelnuk Property.

Please refer to Figure 23.1 for a map showing the adjacent properties.

The information in this Section was extracted from the public registry of the Quebec Ministry of Mines (GESTIM) where filed work and status for these claims can be checked.

The validity of the adjacent properties has not been verified and data on these adjacent properties is not necessarily indicative of mineralization on the Otelnuk Property. Met-Chem is not aware of the exploration activities on the adjacent properties and no production from them seems to have been reported.

Figure 23.1 – Titles Location Map



24.0 OTHER RELEVANT DATA AND INFORMATION

None.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resources

In 2014, Met-Chem completed a review of existing data supplied by WGM and information summarized in WGM's NI 43-101 compliant technical report, dated October 31st, 2013, on the updated Mineral Resource estimate. The data transmitted to Met-Chem by WGM are related to the drillhole database, the block model and a spreadsheet with the bulk density and pycnometer measurement results.

Based on the review of the available information for the Lac Otelnuk Iron Property, we offer the following conclusions:

- The Lac Otelnuk deposits are composed of iron formations of the Lake Superior-type which consists of banded sedimentary rocks composed principally of bands of magnetite and hematite within quartz (chert)-rich rock, with variable amounts of silicate, carbonate and sulphide lithofacies. Lithofacies that are not highly metamorphosed or altered by weathering are referred to as taconite and the Lac Otelnuk deposits are examples of taconite-type iron formation;
- Mineralization in the Lac Otelnuk iron formation consists mainly of magnetite (Fe_3O_4) and hematite (Fe_2O_3); some iron also occurs in silicates, siderite and ferro-ankerite but is economically insignificant. Iron oxide bands containing concentrations of magnetite and/or hematite alternate with grey chert of jasper and are the economically interesting parts of the iron formation that is a gently east dipping interbanded sequence of rocks;
- WGM is satisfied that sampling and assaying for Adriana's programs since 2007 have been performed well and have been effective leading to the generation of a data set sufficient in quality to support the Mineral Resource estimate;
- Specific gravities for the 2013 Mineral Resource estimation of tonnage were completed using a variable density model based on the relationship generated by WGM between % TFe and measured densities, as WGM determined that a variable density model would more accurately define the local variations based on grade rather than using an average density on a per sub-unit basis;
- As with the previous Mineral Resource estimate, WGM built a relationship between the magnetic Fe determined by Satmagan and that determined by DT where both techniques were used to account for the changeover to Satmagan measurements to replace Davis Tube results during the most recent assaying programs. For consistency with previous Mineral Resource estimates, a % DTWR cut-off was retained based on this relationship. A % Magnetic Fe value was determined for each block and this is reported in the current Mineral Resource estimate along with the % DTWR;
- The 2013 Mineral Resource estimate included the new drilling results from the 2012 exploration program and uses of total of 370 drillholes. WGM re-modeled the upper geological sub-units of the Lac Otelnuk iron formation that were previously defined (2a, 2b, 2c, 3a and 3b) and retaining the transitional 2b-c sub-unit identified in the 2012 estimate. A new internal shale waste unit was also defined in the northern part of the Property. Internally, the continuity of the sub-units was excellent, so WGM had no

issues with extending the interpretation beyond 600 m distance. This extension was taken into consideration when classifying the Mineral Resources and these areas were given a lower confidence category. A summary of the NI 43-101 compliant Mineral Resources is provided in Table 25.1 below;

Table 25.1 – 2013 Categorized Mineral Resource Estimate for Lac Otelnuk Iron Project (Cut-Off of 18 % DTWR)

Resource Classification	Tonnes (in billions)	TFe Head (%)	DTWR (%)	Magnetic Fe (%)
Measured	16.21	29.3	25.8	17.8
Indicated	4.43	31.5	24.1	16.7
Total M&I	**20.64**	**29.8**	**25.4**	**17.6**
Inferred	6.84	29.8	26.3	17.8

- The drilling programs have illustrated that the iron formation units have excellent continuity of geology/geometry and TFe grades, with the magnetic Fe grades being more variable due to changes in the magnetite/hematite ratio within the sub-units. The average thickness of the units does not significantly change in the main part of the deposit, but are more variable to the north and south. There appears to be some structural complexity to the northeast of the deposit where possible thrusting has occurred but this was not further explored during the 2013 drilling program as it was not the focus of the campaign;
- The metallurgical testing to date demonstrates that the Lac Otelnuk mineralization can be recovered by fine grinding and magnetic concentration to saleable concentrates with low silica and high iron grades. Test work performed on the 30 Y composite samples has demonstrated concentrate grades with <4 % SiO_2 and >68.5 % iron with a 27.6 % weight recovery;
- The open pit designed for the Lac Otelnuk Project will provide for a 30-year mine life. The Proven and Probable Mineral Reserves within this open pit include 4,993 Mt of ore at an average Davis Tube Weight Recovery of 26.5 %. The open pit is 11.6 km long and 2.8 km wide, reaches a maximum depth of 130 m and has waste to ore stripping ratio of 0.28 to 1;
- The 30-year mine plan that was developed follows the phased approach of the Feasibility Study, producing 30 Mt/y of concentrate in Phase 1 and 50 Mt/y in Phase 2. During peak production, the total number of 363-tonne haul trucks is expected to reach 50, along with ten (10) cable shovels, two (2) hydraulic shovels, four (4) front end wheel loaders, 16 production drills and a large fleet of support and service equipment;
- The processing plant will use proven and reliable equipment to produce a 68.5 % Fe concentrate that will be transported via two (2) pipelines (755 km) to the Pointe-Noire area in Sept-Îles, QC, where is will be dewatered, store and then ship to its final destination;
- It can be stated that the level of confidence in the test work results is high, since (a) a reputable laboratory was used, (b) significant effort was placed on sample

representativeness, and (c) the results were largely coherent and repeatable and were given explanations for instances where these were not true;

- Test work conducted on representative samples of tailings and waste rock have concluded that they should be considered to be non-acid generating with low metal leaching potential and therefore meet the "low risk" classification of the Quebec Directive 019 regulations;
- The system for the tailings and water management has been developed taking into considerations of local topographical, geotechnical, geomorphological and climatic conditions, stringent engineering design and reliable construction concepts, environmental management and closure requirements. The configuration of the TMF provides a tailings storage volume of about 2,350 Mm^3;
- The product delivery system is a concentrate slurry transport system, an economical and reliable means of transporting iron ore concentrate to the project port of export. Further optimization, particularly with respect to the route alignment, will be assessed early in the basic engineering phase of the Project, along with further field investigations;
- A product dewatering-storage-reclaiming facility is designed located near the Port of Sept-Îles. The final product which contains less than 8 % of moisture content will be loaded on the large Cape Size to Chinamax Size bulk carriers through a conveyor/ship loading system at the deep water wharf jointly build by LOM and the Sept-Îles Port Authorities;
- The power for the Project will be supplied by a 735 kV power transmission line connected to the existing Hydro-Québec 735/315 kV substation at Tilly. The 735 kV overhead transmission line is approximately 466 km long and includes one (1) single circuit, one (1) overhead shield wire, and one (1) optical ground wire;
- The Project market study by the SNL Metals & Mining indicates future demand for pellet feed. Both trends and forecasts indicate a rebalancing of the pellet feed premium by the time LOM enters into production;
- The economics of the project are based on the concentrate being shipped primarily to China;
- The Project is technically feasible: the ore can be mined, treated, and delivered to the Port of Sept-Îles for export by employing proven processes and technologies;
- Based on a 30-year mine life and a production of 50 Mt/y of iron ore concentrate as well as the parameters and assumptions set out in this Report, the ROI before taxes varies between 15.9% and 16.1% depending on the scenario.

25.2 Risk Evaluation

There will always have risk associated to a project of this nature and size. Its sheer size in itself is a risk as it becomes more difficult to control and more variables are at play. Being located in a remote area of the province also creates logistic problems not seen anywhere else.

The Iron Ore Market is also going through a major change and readjustment. Commodity prices have fallen recently and although the indicators say the price will go back eventually, there is no guarantees the market will behave as forecasted.

Due to the construction schedule on which the Project is based, it is also hard to predict availability, productivity and competency of construction crews. Fees are based on today's industry rates and can change rapidly due to market conditions.

26.0 RECOMMENDATIONS

Moving forward, we would recommend:

- Carry-out a geotechnical study for the stability of the waste rock piles and overburden stockpiles to confirm the design parameters used in the Feasibility Study.
- Complete geochemical testing on the shale waste unit that appears at the north end of the 30-year open pit. If the test work identifies that the shale waste unit is not a producer of acid rock drainage the pit should be redesigned to include this area in the next phase of the Project.
- That pelletizing tests be done, since the product of the Lac Otelnuk Project is a pellet feed.
- A final review of the options SAG vs. AG milling, considering:
  - Critical size build-up in SAG in relation with the ROM coarse size distribution; blasting and primary crushing optimization using simulation software in regard with potential SAG throughput limitation;
  - Grinding media consumption;
  - Pebble crushing: potential loss of magnetite via the tramp metal rejection; plant footprint and layout.
- A review, during basic engineering, to examine whether tower mills should be used on the Project:
  - Tower mills have not been considered in the plant design because of space requirements, complexity of multiple units and the lack of a competitive market in the large size range required. However, the potential savings in electricity and grinding media costs would be significant.
- Evaluation of the metallurgical performance of the proposed flow sheets:
  - A variability-testing bench-scale program should submit geographically dispersed samples from the first ten (10) years of the mining plan to a process that emulates the proposed flow sheets in order to evaluate the metallurgical performance of the proposed flow sheets in terms of assay quality and mass recovery.
- Mapping of the ore hardness in the mine plan:
  - The block model shall include the ore SPI and BWI; perform a metallurgical model in regards to Fe grade and magnetite grade and calculate concentrate quantity and quality for each block.
- Development of a metallurgical model and calculation of different mass balances for three (3) qualities of ore (rich-average-poor) in order to validate the capacity of the plant and the sizing of the equipment.
- As soon as the site preparatory works are launched and the mine access roads are opened, a bulk sample of ore, about 200 tonnes, could be extracted in order to perform a continuous demonstration test run of the process.

- Complete geotechnical investigations for the main access road, the process plant area, the TMF and Water Management structures, the alignment along the 735 kV power interconnection line and the PDS and the Dewatering-Storage-Reclaiming facilities.
- Perform topographical survey (LiDAR) for the alignment along the Power Transmission line and the PDS and for the Dewatering-Storage-Reclaiming facility area.
- Perform bathymetry survey on the water bodies along the PDS.
- Finalize the EIA reports for submission to the Quebec and Canadian governments.
- Finalize the SIA reports for submission to the Quebec and Canadian governments.
- Submit the Project to public consultation on the EIA/SIA reports, permitting, and licences to obtain local community buy-in.

Budgets have been included in the capital costs estimate to cover for the above work under the EPCM and Owner's costs categories.

27.0 REFERENCES

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