NI 43-101 Technical Report

Feasibility Study of the Southwest Pit

Chibougamau, Québec, Canada

Prepared for:

BlackRock Metals Inc.
Strategic Resources Inc.

Prepared by the following Qualified Persons:

- André Allaire, P. Eng. — BBA Inc.
- Isabelle Leblanc, P. Eng. — BBA Inc.
- Nicolas Skiadas, P. Eng. — Journeaux Associates
- Nathalie Fortin, P. Eng. — WSP
- Claude Bisaillon, P. Eng. — Independent

Effective Date: November 18, 2022
Signature Date: March 9, 2023

DATE AND SIGNATURE PAGE

This report is effective as of the 18th day of November 2022.

Signed and sealed	March 9, 2023
André Allaire, P.Eng., M.Eng., PhD. BBA Inc.	Date
Signed and sealed	March 9, 2023
Isabelle Leblanc, P.Eng., BBA Inc.	Date
Signed and sealed	March 9, 2023
Nicolas Skiadas, P.Eng., M.Eng., Journeaux Associates.	Date
Signed and sealed	March 9, 2023
Nathalie Fortin, P.Eng., M.Env. WSP	Date
Signed and sealed	March 9, 2023
Claude Bisaillon, P.Eng. Independent	Date

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CERTIFICATE OF QUALIFIED PERSON

André Allaire, Eng., M.Eng., PhD.

This certificate applies to the NI 43-101 Technical Report titled "Feasibility Study of the Southwest Pit, Chibougamau, Québec, Canada" (the "Technical Report"), prepared for BlackRock Metals Inc. and Strategic Resources Inc., dated March 9, 2023, with an effective date of November 18, 2022.

I, André Allaire, P.Eng., M.Eng., PhD., as a co-author of the Technical Report, do hereby certify that:

1. I am a currently employed as Senior Consultant, Metallurgy, Mining and Metal Processes with the consulting firm BBA Inc., located at 2020 Robert-Bourassa Blvd. Suite 300, Montréal, QC H3A 2A5.
2. I am a graduate from McGill University of Montreal with a B.Eng. in Metallurgy in 1982, an M. Eng. In 1986 and a Ph.D. in 1991.
3. I am a member in good standing of the Order of Engineers of Québec (# 38480) and a member of the Canadian Institute of Mining Metallurgy and Petroleum.
4. My relevant experience includes:
   - (1982-1984); Process Metallurgist, Horne division, Noranda Inc.
   - (1984-1988); Graduate Studies, Metallurgical Department, McGill University
   - (1988-2000); Process Metallurgist and Study Manager, Hatch & Associés Inc.
   - (2000-2004); Manager Process and Metallurgy, Met-Chem Canada Inc.
   - (2004-2011); Director, Mining and Metals, BBA Inc.
   - (2011-2013); VP Market, Mining and Metals, BBA Inc.
   - (2013-2021); President, BBA Inc.
   - (2021-to Present); Subject Matter Expert, Process
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am independent of the vendor of this Technical Report, BlackRock Metals Inc.
8. I am independent of the issuer of this Technical Report, Strategic Resources Inc.
9. I am author and responsible for the preparation of Chapters 2, 3, 13, 17, 18 (except 18.12), 19, 21, 22 and 24, as well as the relevant sections of Chapters 1, 25, 26 and 27 of this Technical Report.
10. I have visited the BlackRock site that is the subject of the Technical Report, on September 29, 2010, as part of this current mandate.
11. I have had prior involvement with the properties that are the subject of the Technical Report in earlier studies for BlackRock.

12 I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.

13 As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 9th day of March 2023.

Original signed and sealed on file

André Allaire, P.Eng., M.Eng., PhD.
Senior Consultant, Metallurgy
Mining and Metal Processes

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SEDAR CONSENT OF QUALIFIED PERSON

TO: BlackRock Metals Inc.
Strategic Resources Inc.

AND TO: *Autorité des marchés financiers* (as principal regulator)
Nova Scotia Securities Commission
Office of the Superintendent of Securities, Prince Edward Island
Financial and Consumer Services Commission, New Brunswick
Office of the Superintendent of Securities, Service Newfoundland and Labrador
Ontario Securities Commission
Manitoba Securities Commission
Financial and Consumer Affairs Authority of Saskatchewan
Alberta Securities Commission
British Columbia Securities Commission
TSX Venture Exchange

I, André Allaire, P.Eng., M.Eng., PhD., employed with BBA Inc., do hereby consent to the public filing of the NI 43-101 Technical Report prepared for BlackRock Metals Inc. and Strategic Resources Inc., entitled "Feasibility Study of the Southwest Pit, Chibougamau, Québec, Canada" (the "Technical Report"), dated March 9, 2023, and effective as of November 18, 2022, with the securities regulatory authorities referred to above.

I also consent to the use of extracts from, or a summary of, the Technical Report contained in the joint News Release of BlackRock Metals Inc. and Strategic Resources dated on December 13, 2022.

I confirm that I have read the written disclosure in the News Release and that it fairly and accurately represents the information contained in the sections of the Technical Report for which I am responsible.

Signed this 9th day of March 2023.

Original signed on file

André Allaire, P.Eng., M.Eng., PhD.,
Senior Consultant, Metallurgy
Mining and Metal Processes

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CERTIFICATE OF QUALIFIED PERSON

Isabelle Leblanc, P.Eng.

This certificate applies to the NI 43-101 Technical Report titled "Feasibility Study of the Southwest Pit, Chibougamau, Québec, Canada" (the "Technical Report"), prepared for BlackRock Metals Inc. and Strategic Resources Inc., dated March 9, 2023, with an effective date of November 18, 2022.

I, Isabelle Leblanc, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a currently employed as Vice-President, Mining and metals markets with the consulting firm BBA Inc., located at 2020 Robert-Bourassa Blvd. Suite 300, Montréal, QC H3A 2A5.
2. I am a graduate from the mining engineering program of École Polytechnique de Montreal in 2007. I have practiced my profession continuously since my graduation.
3. I am a member in good standing of the Order of Engineers of Québec (# 144395) and Canadian Institute of Mining, Metallurgy and Petroleum.
4. My relevant experience includes; expertise in a wide range of activities including pit optimization and design, long range mine planning, 'trade off' studies, equipment selection, Opex and Capex cost estimation, financial analysis and the preparation of NI 43-101-compliant reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am independent of the vendor of this Technical Report, BlackRock Metals Inc.
8. I am independent of the issuer of this Technical Report, Strategic Resources Inc.
9. I am author and responsible for the preparation of Chapters 15 and 16 as well as the relevant sections of Chapters 1, 25, 26 and 27 of this Technical Report.
10. I have not visited the BlackRock the property that is the subject to the Technical Report.
11. I have had prior involvement with the properties that are the subject of the Technical Report in earlier studies for BlackRock.

12 I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.

13 As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 9th day of March 2023.

Original signed and sealed on file

Isabelle Leblanc, P.Eng.,
Vice-President, Mining and metals markets

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SEDAR CONSENT OF QUALIFIED PERSON

TO: BlackRock Metals Inc.
Strategic Resources Inc.

AND TO: *Autorité des marchés financiers* (as principal regulator)
Nova Scotia Securities Commission
Office of the Superintendent of Securities, Prince Edward Island
Financial and Consumer Services Commission, New Brunswick
Office of the Superintendent of Securities, Service Newfoundland and Labrador
Ontario Securities Commission
Manitoba Securities Commission
Financial and Consumer Affairs Authority of Saskatchewan
Alberta Securities Commission
British Columbia Securities Commission
TSX Venture Exchange

I, Isabelle Leblanc, P.Eng., employed with BBA Inc., do hereby consent to the public filing of the NI 43-101 Technical Report prepared for BlackRock Metals Inc. and Strategic Resources Inc., entitled "Feasibility Study of the Southwest Pit, Chibougamau, Québec, Canada" (the "Technical Report"), dated March 9, 2023, and effective as of November 18, 2022, with the securities regulatory authorities referred to above.

I also consent to the use of extracts from, or a summary of, the Technical Report contained in the in the joint News Release of BlackRock Metals Inc. and Strategic Resources dated on December 13, 2022.

I confirm that I have read the written disclosure in the News Release and that it fairly and accurately represents the information contained in the sections of the Technical Report for which I am responsible.

Signed this 9th day of March 2023.

Original signed on file

Isabelle Leblanc, P.Eng., Project Engineer,
Vice-President, Mining and metals markets

JOURNEAUX ASSOC
Division LAB JOURNEAUX INC.

CERTIFICATE OF QUALIFIED PERSON

Nicolas Skiadas, P.Eng., M.Eng.

This certificate applies to the NI 43-101 Technical Report titled "Feasibility Study of the Southwest Pit, Chibougamau, Québec, Canada" (the "Technical Report"), prepared for BlackRock Metals Inc. and Strategic Resources Inc., dated March 9, 2023, with an effective date of November 18, 2022.

I, Nicolas Skiadas, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a currently employed as Project Manager with the consulting firm Journeaux Associates, Division of Lab Journeaux Inc., located at 801 Bancroft, Pointe-Claire, Quebec, Canada, H9R 4L6.
2. I am a graduate from the mining engineering program of McGill University with Bachelor's degree in Engineering in 1977 and Master's degree in Engineering in 1982 and have worked as a Civil Engineer continuously since my graduation.
3. I am a member in good standing of the Order of Engineers of Québec (# 117881).
4. My relevant experience includes:
    - Design and site supervision of construction and project management for earthwork construction, roads, dams. Design of earthworks, slope stability, hydrological studies (culverts, channels), design and construction of mining dams for tailings impoundment.
    - Participated in numerous prefeasibility and feasibility studies for the tailings disposal facilities (wet and dewatered tailings) for various mines including water balance for tailings impoundments.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am independent of the vendor of this Technical Report, BlackRock Metals Inc.
8. I am independent of the issuer of this Technical Report, Strategic Resources Inc.
9. I am author and responsible for the preparation of Chapter 18.1.13, 18.1.14.2, 20.1.7.2, 20.1.7.6, 20.1.8.1.2, 20.1.9 as well as the relevant sections of Chapters 1, 25, 26 and 27 of this Technical Report.
10. I have visited the BlackRock Property that is the subject to the Technical Report on May 19-20, 2011.
11. I have had no prior involvement with the properties that are the subject of the Technical Report.
12. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.

13 As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 9th day of March 2023.

Original signed and sealed on file

Nicolas Skiadas, P.Eng., M.Eng.,
Project Manager

JOURNEAUX ASSOC
Division LAB JOURNEAUX INC.

SEDAR CONSENT OF QUALIFIED PERSON

TO: BlackRock Metals Inc.
Strategic Resources Inc.

AND TO: *Autorité des marchés financiers* (as principal regulator)
Nova Scotia Securities Commission
Office of the Superintendent of Securities, Prince Edward Island
Financial and Consumer Services Commission, New Brunswick
Office of the Superintendent of Securities, Service Newfoundland and Labrador
Ontario Securities Commission
Manitoba Securities Commission
Financial and Consumer Affairs Authority of Saskatchewan
Alberta Securities Commission
British Columbia Securities Commission
TSX Venture Exchange

I, Nicolas Skiadas, P.Eng., M.Eng., employed with Journeaux Associates, Division of Lab Journeaux Inc., do hereby consent to the public filing of the NI 43-101 Technical Report prepared for BlackRock Metals Inc. and Strategic Resources Inc., entitled "Feasibility Study of the Southwest Pit, Chibougamau, Québec, Canada" (the "Technical Report"), dated March 9, 2023, and effective as of November 18, 2022, with the securities regulatory authorities referred to above.

I also consent to the use of extracts from, or a summary of, the Technical Report contained in the joint News Release of BlackRock Metals Inc. and Strategic Resources dated on December 13, 2022.

I confirm that I have read the written disclosure in the News Release and that it fairly and accurately represents the information contained in the sections of the Technical Report for which I am responsible.

Signed this 9th day of March 2023.

Original signed on file

Nicolas Skiadas, P.Eng., M.Eng.,
Project Manager

CERTIFICATE OF QUALIFIED PERSON

Nathalie Fortin, P.Eng., M.Env.

This certificate applies to the NI 43-101 Technical Report titled "Feasibility Study of the Southwest Pit, Chibougamau, Québec, Canada" (the "Technical Report"), prepared for BlackRock Metals Inc and Strategic Resources Inc., dated March 9, 2023, with an effective date of November 18, 2022.

I, Nathalie Fortin, P.Eng., M.Env., as a co-author of the Technical Report, do hereby certify that:

1. I am a Business Unit Director - Environmental Management /Earth & Environment with the consulting firm WSP, located at 1135, boul. Lebourgneuf, Québec (Québec), Canada, G2K 0M5.
2. I am a graduate engineer.
3. I am a member in good standing of Ordre des ingénieurs du Québec (OIQ no. 112062)
4. My relevant experience includes environmental assessment and management.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am independent of the vendor of this Technical Report, BlackRock Metals Inc.
8. I am independent of the issuer of this Technical Report, Strategic Resources Inc.
9. I am author and responsible for the preparation of Chapter 20.
10. I have visited the BlackRock metallurgical plant property that is the subject of the Technical Report on July 16, 2019.
11. I have had no prior involvement with the property that is the subject of the Technical Report.
12. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
13. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 9th day of March 2023.

Original signed and sealed on file

Nathalie Fortin, P.Eng., M.Env.
Business Unit Director - Environmental Management
/ Earth & Environment Sciences

SEDAR CONSENT OF QUALIFIED PERSON

TO: BlackRock Metals Inc.
Strategic Resources Inc.

AND TO: *Autorité des marchés financiers* (as principal regulator)
Nova Scotia Securities Commission
Office of the Superintendent of Securities, Prince Edward Island
Financial and Consumer Services Commission, New Brunswick
Office of the Superintendent of Securities, Service Newfoundland and Labrador
Ontario Securities Commission
Manitoba Securities Commission
Financial and Consumer Affairs Authority of Saskatchewan
Alberta Securities Commission
British Columbia Securities Commission
TSX Venture Exchange

I, Nathalie Fortin, P.Eng., M.Env., employed with WSP, do hereby consent to the public filing of the NI 43-101 Technical Report prepared for BlackRock Metals Inc. and Strategic Resources Inc., entitled "Feasibility Study of the Southwest Pit, Chibougamau, Quebec, Canada" (the "Technical Report"), dated March 9, 2023, and effective as of November 18, 2022, with the securities regulatory authorities referred to above.

I also consent to the use of extracts from, or a summary of, the Technical Report contained in the joint News Release of BlackRock Metals Inc. and Strategic Resources dated on December 13, 2022.

I confirm that I have read the written disclosure in the News Release and that it fairly and accurately represents the information contained in the sections of the Technical Report for which I am responsible.

Signed this 9th day of March 2023.

Original signed on file

Nathalie Fortin, P.Eng., M.Env.
Business Unit Director - Environmental Management
/ Earth & Environment Sciences

CERTIFICATE OF QUALIFIED PERSON

Claude Bisaillon, P.Eng

This certificate applies to the NI 43-101 Technical Report titled "Feasibility Study of the Southwest Pit, Chibougamau, Quebec, Canada" (the "Technical Report"), prepared for BlackRock Metals Inc. and Strategic Resources Inc., dated March 9, 2023, with an effective date of November 18, 2022.

I, Claude Bisaillon, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I was previously employed as a Project Engineer with SGS Canada (previously Geostat Systems International Inc.) at the time of the site visits and the writing of the portions of this report; a firm of consulting geologists and engineers: #203-10 Blvd de la Seigneurie Est, Blainville, Quebec J7C 3V5 Canada.
2. I am a currently employed as a Senior Geotechnical Engineer with DRA Americas (previously Met-Chem); a firm of consulting geologists and engineers: 6th floor – 555 Blvd René-Lévesque, Montréal, Quebec H2Z 1B1 Canada. My current employer is not involved with any portions of this report.
3. I am a graduate from Concordia University with a B.Sc. in Geology (1991) and from Université Laval with a B.Eng. in Geological Engineering (1996) and I have practiced my profession continuously since my graduation.
4. I am a member in good standing of the Order of Engineers of Québec (# 116407).
5. My relevant experience includes;
    a) Over 25 years of consulting in the field of Mineral Resource estimation, orebody modeling, mineral resource auditing and geotechnical engineering in Canada, USA, Asia, Africa, and South America.
    b) Participation and author of several NI43-101 Technical Reports.
    c) QP Review, audits, due diligence, interpretation of geoscientific data for several projects.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
7. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
8. I am independent of the vendor of this Technical Report, BlackRock Metals Inc.
9. I am independent of the issuer of this Technical Report, Strategic Resources Inc.
10. I am author and responsible for the preparation of Chapters 4, 5, 6, 7, 8, 9, 10, 11, 12, 14 and 23 as well as the relevant sections of Chapters 1, 25, 26 and 27 of this Technical Report. The effective date of the Mineral Resource Statement prepared for the BlackRock Project is August 26, 2022.
11. I have visited the BlackRock Property that is the subject to the Technical Report on February 23-25, 2011.
12. I have had prior involvement with the properties that are the subject of the Technical Report in earlier studies for BlackRock.

13. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
14. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 9th day of March 2023.

Original signed and sealed on file

Claude Bisaillon, P.Eng.
Project Engineer

SEDAR CONSENT OF QUALIFIED PERSON

TO: BlackRock Metals Inc.
Strategic Resources Inc.

AND TO: *Autorité des marchés financiers* (as principal regulator)
Nova Scotia Securities Commission
Office of the Superintendent of Securities, Prince Edward Island
Financial and Consumer Services Commission, New Brunswick
Office of the Superintendent of Securities, Service Newfoundland and Labrador
Ontario Securities Commission
Manitoba Securities Commission
Financial and Consumer Affairs Authority of Saskatchewan
Alberta Securities Commission
British Columbia Securities Commission
TSX Venture Exchange

I, Claude Bisaillon, P.Eng., previously employed as a Project Engineer with SGS Canada (previously Geostat Systems International Inc.) at the time of the site visits and the writing of the portions of this report; a firm of consulting geologists and engineers: #203-10 Blvd. de la Seigneurie Est, Blainville, Québec J7C 3V5 Canada; and

Currently employed as a Senior Geotechnical Engineer with DRA Americas (previously Met-Chem); a firm of consulting geologists and engineers: 6th floor – 555 Blvd René-Lévesque, Montréal, Quebec H2Z 1B1 Canada. My current employer is not involved with any portions of this report.

I do hereby consent to the public filing of the NI 43-101 Technical Report prepared for BlackRock Metals Inc. and Strategic Resources Inc., entitled "Feasibility Study of the Southwest Pit, Chibougamau, Québec, Canada" (the "Technical Report"), dated March 9, 2023, and effective as of November 18, 2022, with the securities regulatory authorities referred to above.

I also consent to the use of extracts from, or a summary of, the Technical Report contained in the joint News Release of BlackRock Metals Inc. and Strategic Resources dated on December 13, 2022.

I confirm that I have read the written disclosure in the News Release and that it fairly and accurately represents the information contained in the sections of the Technical Report for which I am responsible.

Signed this 9th day of March 2023.

Original signed on file

Claude Bisaillon, P.Eng.
Project Engineer

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TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
$/t conc.	dollar per tonne
% (w/w)	mass fraction
% (v/v)	volume concentration
%SAT	percent Satmagan (% magnetite)
°C	degrees centigrade
3D	three dimensional
AACE	Association for the Advancement of Cost Engineering
ACRS	Aluminum Conductors Steel Reinforced
AE	Armitage
Ag	silver
AGB	Abitibi Greenstone Belt
Ai	Abrasion Index
Al_2O_3	aluminium oxide, alumina
AMD	acid mine drainage
AMV	Ammonium metavanadate
An	Anorthite
AQI	air quality index
ARD	acid rock drainage
As_2O_3	Arsenic
Au	gold
BaO	barium oxide
BAPE	Bureau d'audiences publiques sur l'environnement
BBA	Breton, Banville and Associates
BCS	Basal Chrome Series
BFA	Bench Face Angle
BLS	US Bureau of Labor Statistics
BRMI (or BRM)	BlackRock Metals Inc.
BW_i	Ball Mill Work Index
C	Carbon
°C	Celcius
CAD	Canadian Dollar
CaO	calcium oxide
CAPEX	Capital Expenditure

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
CCTV	closed-circuit television
CCS	Cold Compressive Strength
CH_4	Methane
CIM	Canadian Institute of Mining
CFILNQ	Canadian National Railway
CFM	Cubic Feet per Minute
CND	Contaminated neutral drainage
cm^2	Cubic centimeter
cm^2/gm	Square centimetre per gram
CM	Construction management
CMAX	Regional Economic Development Group
CN	Canadian National railway
CNSC	Canadian Nuclear Safety Commission
CO	Carbon monoxide
CoA	Certificate of Analysis
CO_2	Carbon dioxide
COG	Cut off grade
conc.	Concentrate
Cr	Chromium
CRM	County Regional Municipality
Cr_2O_3	Chromium oxide
CSA	Canadian Standard Association
CSV	Comma Separated Value
CTEU-9	Centre technologique des eaux usées – méthode 9
CTQ	Commission des transports du Québec
Cu	Copper
Cu-Au	Copper-Gold
CWi	Crusher Work Index
DC	Direct Current
DFO	Fisheries and Oceans Canada
DH	Drillhole
DMDS	Dimethyl Disulfide
DPS	Development Port Saguenay

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
DR	Direct Reduction
DRI	Direct Reduced Iron
DRP	Direct reduction plant
DTA	Davis Tube analysis
DTH	Down-the-hole
DWT	Drop Weight Test
EAF	Electric Arc Furnace
ECW	Equipment cooling water
EDO	Effluent discharge objectives
EI	Environmental impact
EIA	Environmental Impact Assessment
EIJB	Eeyou Ishtchee James Bay Regional Government
EPCM	Engineering, Procurement and Construction Management
EPMA	Electron probe micro-analyzer
EPOG	Entente de Principe d'Ordre Général
EQA	Environment Quality Act
ESPs	Electrostatic precipitators
Fe	Iron
FEE	Inlet flow of cooling gas to the reactor
Fe_3C	Iron Carbide
Fe_2O_3	Iron oxide (ferric oxide)
Fe_2TiO_4	Ulvospinelle
Fe_3O_4	Magnetite
Fe2+ OR Fe3+	Ferric oxides
FeO	Iron oxide (ferrous oxide)
FeSi	Ferrosilicon
Fe_T	Total iron
FeV	Ferrovanadium
FeV_{80}	ferro-vanadium
FeM	Metallic iron
FER	Flow of reducing gas
FNTP	Full Notice to Proceed
FOB	Free on Board

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
FS	Feasibility Study
ft	Foot
G	Gauss
g	Gram
Gcal/t	Gigacalories per ton
g/t	Gram per tonne
G&A	General and Administrative
g/cm^3	gram per cubic centimetre
Ga	billion years (giga-annum)
GESTIM	Le Système de Gestion Des Titres Miniers
GHG	Greenhouse gas
GIS	Gas Insulated Switchgear
GPS	Global Positioning System
h	hour
H_2O	Water
H2	Hydrogen
ha	hectare (10 000 m^2)
HBI	Hot Briquetted Iron
H&S	Health and Safety
HMI	Human-Machine Interface
HPPI	High Purity Pig Iron
HPPI/NPI	Nodular pig iron
HQ	Hydro-Québec
HVAC	Heating, Ventilation and Air Conditioning
HYL	Hylsa Technology Division
h/y	Hours per year
IBA	Impact and Benefits Agreement
I/O	Input/Output
ID	Identification
IDn	Inverse Distance to the power n.
ID^1	Inverse distance
ID^2	Inverse distance squared
ID^3	Inverse distance cube

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
IIMA	International Iron Metallics Association
IOS	Service Géoscientifique inc. (IOS)
INGE	Inlet of Natural Gas at the Cooling Circuit
INGR	Inlet of Natural Gas to the Reduction Circuit
IRA	Inter Ramp Angle
IRR	Internal Rate of Return
ISAQ	Archaeological Sites of Québec
ISP	Internet Service Provider
ITSP	Internet Telephone Service Provider
JBNQA	James Bay Northern Québec Agreement
K_2O	Potassium oxide
kg	kilogram
kg/cm^2	kilogram per square centimetre
kg/m^3	kilogram per cubic metre
kg/t	kilogram per tonne
km	kilometre
kt	kilotonne
ktpa	Kilo-Tonnes Per Annum
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hours per tonne
L	litre
LDC	Lac Doré Complex
LIMS	Low-Intensity Magnetic Separator
LMI	Layered Mafic Intrusions
LOI	Loss on Ignition
LOM	Life of Mine
LTD	LTD testing protocol
LVM	LVM Inc.
M	Million or mega
m	metre
m^2	square metre

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
m^3	cubic metre
m^3/t	cubic metre per ton
Ma	million years
MABA	Modified Acid Base Accounting
MCS (or MC)	Middle Cumulate Series or Middle Cumulate Horizon
MCW	Machine cooling water
M+I	mineral inventory
M&I	Measured and Inferred
MDDEFP	Ministère du Développement durable, de l'Environnement, de la Faune et des Parcs du Québec
MDDELCC (MELCC)	Ministère de l'Environnement et de la Lutte contre les changements climatiques
MDEA	Methyl diethanolamine
MDMER	Metal and Diamond Mining Effluent Regulations
MELCCFP	Ministère de l'Environnement et de la Lutte contre les changements climatiques, de la Faune et des Parcs
MERN	Ministère de l'Énergie et des Ressources naturelles
MET	Metallization
MFFP	Ministère des Forêts, de la Faune et des Parcs du Québec
MgO	Magnesium oxide
MH	Man-hours
MJB	Municipality of James Bay (James Bay Municipality)
ML	Metal Leaching
mm	millimetre
mmbtu	One Million British Thermal Units
Mm^3	Million cubic metres
M/y	Million per year
Mn	Manganese
MnO	Manganese oxide
MPI	Merchant Pig Iron
MPSO	Mine Plan Schedule Optimizer
MRNQ (MRN)	Ministère des Ressources Naturelles du Québec
Mt	Megatonne
Mtpy	Million tonnes per year

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
MTQ	Ministère des Transports du Québec.
MW	Megawatt
Na_2O	sodium oxide
Na2CO3	sodium carbonate
Na_2SO_4	Sodium Sulfate
NaCl	Sodium Chloride
$NaVO_3$	water-soluble sodium metavanadate
Nb_2O_3	Niobium
NE	Northeast
NFPA	National Fire Protection Association
Ni	Nitrogen
nm	nanometers
Nm^3	Normal cubic meter
NNE	North- Northeast
NN	Nearest-neighbor
NO^2	Nitrogen dioxide
NPV	Net Present Value
NQ	Drill core size (4.8 cm diameter)
NRC	National Research Council Canada
NTS	National Topography System
O3	Ozone
OBM	Ore Based Metallics
OHL	Overhead Line
OIQ	Ordre des Ingénieurs du Québec
OK	ordinary kriging
OPEX (OpEx)	Operating Expenditure
OSA	Overall slope angles
OSBF	Open Slag Bath Furnace
Pb	lead
P_2	phosphor
P_2O_5	phosphorus pentoxide
P_{80}	Size at which 80% of product passes
PAEI	Industrial development plan

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
PAG	Potentially Acid Generating
PAN	Panzhihua
PCN	process control network
PCS	Processor
PCW	Process cooling water
PFS	Prefeasibility Study
PGMs	Platinum Group Metals
PK	Photonic Knowledge
PLC	programmable logic controller
PMS	Primary Magnetic Separation
$PM_{2.5}$	Fine particulate matter
PM_{10}	Diameter of 10 microns or less
PP1	first pilot plant
PP2	second pilot plant
PP	Pre-Production
PPI	Producer Price Indices
PPM	parts per million
PSCS	Process & Safety Control System
PSI	Pounds per Square Inch
PST	Total suspended particles
PYR	PyroGenesis Canada Inc.
QA/QC	Quality Assurance/Quality Control
QCW	Quenching cooling water
QIT	Quebec Iron and Titanium
QP	Qualified Person (as per NI 43-101 standards)
R^2	Coefficient of Determination
ROM	Run of Mine
RQD	Rock Quality Designation
RSR	Roberval Saguenay Railway
RW_I	Rod Mill Work Index
S	Sulphur
Satmagan	SATuration MAGnetic ANalysis
SAG	Semi-Autogenous Grinding

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
SAT	Satmagan
SCADA	Supervisory Control and Data Acquisition
SCC	Standards Council of Canada
SCFM	Standard Cubic Feet Per Minute
SCIM	Induction Motors
SEM	Scanning Electron Microscope
SG	Specific Gravity
SGA	Sales and General Administration cost
SiO_2	Silica
SMC	SAG Mill Comminution Test
SMS	Secondary Magnetic Separation
SO_2	Sulphur dioxide
SrO	Strontium oxide
SPLP	Synthetic Precipitation Leaching Procedure
SQ	Sûreté du Québec
SW	Southwest
SWMM	Storm Water Management Model
t	tonnes
TCLP	Toxicity Characteristic Leaching Procedure
TER	Temperature at the inlet of the reactor
t/h	Ton per hour
Ti	Titanium
$TiFeO_3$	ilmenite
TiO_2	Titanium oxide, titania
Titan	Titaniferous Capping Series
TJCM	Table Jamésienne de Concertation Minière
TMF	Tailings Management Facilities
TMP	Total Particulate Matter
t/m^3	Tonne per cube metre
tpd	tonnes per day
tph	tonnes per hour
tpy	tonnes per year
t/y	tons/year

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
U/F	Underflow
ULS	Upper Layered Series
URSTM	Unité de recherche et de service en technologie minérale
USD	United States Dollar
UTM	Universal Transverse Mercator
UV	Ultraviolet
V	Volt
V	Vanadium
V_2O_5	Vanadium pentoxide
VEC	Valued Ecosystem Components
VFD	Variable Frequency Drive
VoIP	Voice over Internet Protocol
V-slag	Vanadium slag
VRFB	Vanadium redox flow batteries
VTM	Vanadium Titanium Magnetite
WI_B	Bond Ball Mill Work Index
WRA	Whole Rock Analysis
wt%	without steam percentage
w/w	weight for weight
XRF	X-Ray Fluorescence
y	year
ZR	zero-reformer
Zn	zinc
ZnO	zinc oxide
ZrO_2	zirconium dioxide

1. Executive Summary

1.1 Introduction

On December 13, 2022, BlackRock Metals Inc. (herein referred to as "BlackRock" or "BRM") entered into an arm's length share exchange agreement with Strategic Resources Inc. ("Strategic"), pursuant to which Strategic will acquire all of the outstanding shares in BlackRock for shares of Strategic (the "Transaction"). Upon completion of the Transaction, BlackRock will become a wholly-owned subsidiary of Strategic.

The BlackRock vanadium-titanium-magnetite (VTM) deposit is located approximately 700 km north of Montréal, 20 km southeast (60 km by road) of Chibougamau, Québec, Canada. Two deposits, Southwest and Armitage, have been defined for development as "open pits", and BlackRock owns 100% of these deposits. Previous Feasibility Studies have investigated the development, mining and processing of both pits. This Feasibility Study will investigate in detail the development of the Southwest Pit only, with indication of Resources of the Armitage pit. Bench and pilot-scale programs have successfully produced a 62% Fe magnetite concentrate from the Southwest and Armitage pits alike.

BlackRock mandated BBA to perform a Feasibility Study based on contributions from a number of independent consulting firms. The goal of the current study is to produce an Updated Feasibility Study (- 10 to + 15%) for the production of 856,000 tpy of magnetite concentrate. Although the costing and engineering do not include the ilmenite process, the layouts and logistics take into account that an ilmenite plant may potentially be built next to the magnetite plant in a future project phase.

A Feasibility Study was completed in September 2013 (Sedar link) evaluating the production of magnetite and ilmenite concentrate from both the Southwest and Armitage pits. The production goals were to produce 3.0 Mtpy of magnetite concentrate and a resulting 0.76 Mtpy of ilmenite concentrate.

In August 2015, BlackRock mandated BBA to revise the aforementioned study to lower the production to roughly 0.9 Mtpy of magnetite and 0.2 Mtpy of ilmenite concentrates and evaluate the effect on the mining strategy, processing, layouts and project financials at a pre-feasibility level (± 25%). BlackRock subsequently mandated BBA in October 2016 to revise the mine plan, mining capital costs and project operating costs to reflect a production of roughly 0.83 Mtpy of magnetite and 0.18 Mtpy of ilmenite concentrate. These reports were internal to BRM and were not published.

Between 2018 and 2020, BBA was mandated to develop detailed engineering on both Beneficiation and Metallurgical Plant, based on studies produced for the Metallurgical Plant in 2018 by SNC supported by TENOVA technology supplier. CAPEX and OPEX were then reviewed at the end of 2020, to reflect the details developed up to this date with the engineering work completed.

The BlackRock Metallurgical Plant development is located 400 km from the vanadium-titanium-magnetite mine deposit and 2.5 km from the Port of Saguenay, Québec, Canada. In 2022, BlackRock mandated BBA to produce a complete NI43-101 technical report for both the Metallurgical and Beneficiation Plants based on contributions from various independent consulting firms. In addition, BBA was mandated to update the mine plan, mine operating costs, and Beneficiation Plant operating costs to reflect a production of 856,000 tpy of magnetite concentrate.

1.2 Contributors

This Technical Report has been prepared for BlackRock based on work conducted by a number of independent consultants. A summary of the Study contributors for the Beneficiation Plant is listed in Table 1-1.

Table 1-1: Beneficiation Plant Feasibility Study contributors

Consulting Firm or Entity	Area of Responsibility	Responsible Parties
BBA Inc.	Metallurgical Testwork and Processing Reserves and Mining Methods Capital and operating cost models Site infrastructure	André Allaire Isabelle Leblanc Nathalie Blackburn Claude Catudal
/ Independent	Geology, Geological modelling and resource definition	Claude Bisaillon
Journeaux Assoc.	Tailings and water management Mine Closure logistics and estimates	Nicolas Skiadas
WSP	Waste rock and tailings geochemistry Permitting and environmental impacts	Nathalie Fortin

For the Metallurgical Plant, This Technical Report has been prepared for BlackRock based on work conducted by a number of independent consultants. A summary of the Study contributors is listed in Table 1-2

Table 1-2: Metallurgical Plant Technical Study contributors

Consulting Firm or Entity	Area of Responsibility
BBA Inc.	Metallurgical Testwork and Processing Capital and operating cost models Site infrastructure
Tenova HYL	Direct reduction testwork
Tenova Pyromet	Smelting/Pyrometallurgical testwork
Corem	Test VTM concentrate suitability for pelletizing
Mintek	Pyro metallurgical and vanadium slag processing testwork, Bench scale demonstration of the production of FeV

1.3 Key Outcomes

Key outcomes from this Feasibility Study are summarized in Table 1-3 and Table 1-4, and include some of the following information:

- Proven and Probable Mineral Reserves totalling 127.8 Mt from the Southwest Pit;
- Average Satmagan (SAT), V_2O_5 and TiO_2 grades are 18.8%, 0.46% and 7.8% respectfully for the Southwest Pit;
- The estimated total recoverable magnetite concentrate from the Southwest Pit is 33.2 Mt;
- The concentrator will process an average of 3.3 Mtpy of run of mine ore with the maximum year treating 3.5 Mtpy and can handle up to roughly 3.8 Mtpy depending on feed grade;
- The Southwest open pit stripping ratio (LOM) is 2.2 (waste and overburden to ore ratio).

Table 1-3: Key outcomes – Concentrate production - Beneficiation Plant

Parameters	Unit	Outcome
Proven and Probable Mineral Reserves (Southwest)	Mt	127.8
Southwest Average Satmagan	% SAT*	18.8
Total Magnetite Production (Southwest)	Mt	33.2
Southwest Average TiO_2 Grade	% TiO_2	7.8
Southwest Average V_2O_5 Head Grade	%V_2O_5	0.46
Southwest Average V_2O_5 Concentrate Grade	%V_2O_5	1.33
Average Magnetite Concentrate Weight Recovery (Southwest)	% weight	26.0

Table 1-4: Beneficiation Plant key outcomes – Capital expenses and financial results

Parameter	Unit	Outcome
Initial Plant and Site Infrastructure Capital Costs	$M	287.72
Other Capital Costs during Pre-Production (Leasing, Mine Equipment, Mining Operating Costs in PP, etc.)	$M	13.2
Sustaining Capital Costs	$M	160.2

* Note that Satmagan (%) is a measured value that is relative to the level of magnetite (Fe_3O_4) present in the sample. For all intents and purposes, the Satmagan level can be assumed to be equal to the Fe_3O_4 content. Stoichiometrically, 1% of Fe_3O_4 (or Satmagan) refers to 0.7236% magnetic Fe (not to be confused with total Fe that also considers other iron-bearing species).

Key outcomes for the Metallurgical Plant from this Feasibility Study are summarized in Table 1-5 and Table 1-6, and include some of the following information:

- Adequate strength of green balls (unfired pellets) is achieved with 1% bentonite binder, 55% of pellets below 45 µm, and moisture content of about 9%;
- Concentrate regrinding did not significantly improve the quality of fired pellets;
- A firing temperature of 1250 °C proved appropriate to attain adequate pellet strength;
- Fired pellets were determined to have proper physical characteristics and strength necessary for reduction under commercial condition;
- Preferred basicity ratio is 0.4 (CaO/SiO_2);
- Smelting furnace temperature of 1650 °C produced good separation between the metal and the slag;
- Average smelting recovery for vanadium to slag and metallic iron are both above 90%.

Table 1-5: Key outcomes: DR Quality pellet production

Element	Unit	Outcome
Average Metallization	%	90
Average Carbon Content	%	3.6
Average Silicon Content	%	0.007
Tumble ISO	-	92.7
Cold Compressive Strength (CCS)	Kg/pellet	266
Basicity Ratio	CaO/SiO_2	0.4

Table 1-6: Key outcomes – High-quality pig iron production

Parameter	Unit	Outcome
Iron Grade	%	93.8
Vanadium Grade	%	1.1
Carbon Grade	%	3.4
Chromium Grade	%	0.3
Titanium Grade	%	0.3
Silicon Grade	%	0.4

1.4 Property Description

1.4.1 Beneficiation Plant

By road, the BlackRock Property is located 60 kilometres from the city of Chibougamau, Québec, in the Lemoine, Rinfret, Dollier and Queylus townships. BlackRock's property covers part of the Chibougamau Municipality, the James Bay Municipality (JBM) and the Domaine-du-Roy Regional Municipality. The infrastructures are located 100% on the traditional territory of the Cree Nation, covered by the James Bay agreement. The infrastructures are also all in the limits of the Chibougamau municipality.

As of November 2015, the Property consists of 230 claims covering an area of 8,687 hectares. The extent of the Property covers 22 km along a northeast-southwest axis and is registered under the name of BlackRock Metals Inc. The site claims have been updated as of November 9, 2022, by Daniel Dutton of BRM.

1.4.2 Metallurgical Plant

BRM has chosen to construct its metallurgical production facility on an already qualified heavy industrial land belonging to the Saguenay Port Authority near the maritime terminal of Grande-Anse in the City of Saguenay, La Baie district. The land will remain the property of the Saguenay Port Authority and will be leased to BlackRock Metals via a long-term lease. The federal industrial-port zone offers several advantages to the construction and operation of the metallurgical production facility:

- Deep-water port accessible year-round allowing shipped transport to deliver and export materials and products at low cost. At the same time rail and road links offer alternative links to markets;
- Qualified labour in the steel industry sector;

- Site provided by the Port Authority of Saguenay already cleared and levelled;
- Situated in a designated industrial use zone;
- Located away from populated areas in a non-sensitive site chosen to mitigate environmental disturbance.

1.5 Adjacent Properties

BlackRock's Southwest and Armitage deposits are surrounded by multiple exploration projects at various stages of exploration and for various commodities. Figure 1-1 shows the relationship between BlackRock's Project (in dark red) and the surrounding explorers. Information is gathered from the GESTIM website and is up to date as of January 22, 2023. More details on the Adjacent Properties can be found in Section 23 of the Report.

1.5.1 Beneficiation Plant

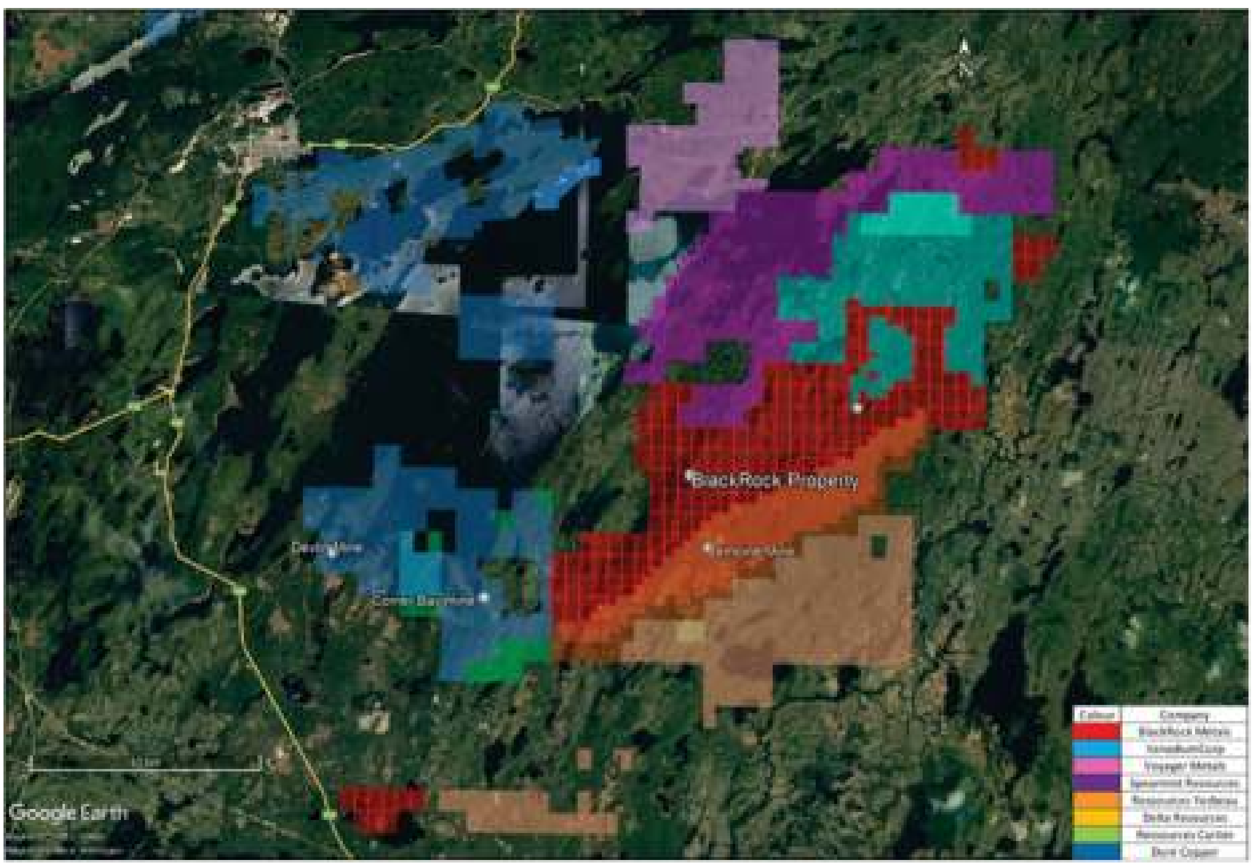

Figure 1-1: Adjacent properties

1.6 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.6.1 Beneficiation Plant

The project area is accessible from Highway 167 and by the forestry road No. 210. A network of secondary roads accesses the deposit, stemming from road 210. These roads will require maintenance for regular travel.

A deep-water port is available in Grande-Anse, in the Saguenay area, for the loading of Cape Size Vessels.

BlackRock Metals Inc. will transport its concentrate by truck to a train loadout located approximately 25 km from the concentrator. From there, the magnetite concentrate will be sent to the Metallurgical plant for further treatment.

A commercial airport is located approximately 60 km from the Property.

A high-voltage (161 kV) power line located close to the CN railroad is accessible. Water is available locally. Other regional infrastructures are available in the nearby town of Chibougamau, 161 kV and 735 kV power stations, a hospital, an airport, various shops and road construction services, etc.

The climate is typically cold and continental, with tepid summers (15°C July average) and cold winters (-20°C January average). Average annual precipitation is approximately 919 mm. Snow is usually present from late October to early May.

1.6.2 Metallurgical Plant

The Metallurgical Plant is accessible by road and sea. From Québec City, the beginning of the primary access route to the region is by paved Highway 175 North, running east of Lac Kénogami. Highway 175 North is intersected by Highway 170 East that leads to Chemin de la Grande-Anse. Chemin du Quai-Marcel-Dionne cuts on Chemin de la Grande-Anse and leads to the project site. The well-maintained road to the dock of Port of Grande-Anse will grant passage for trucks to haul bulk materials back and forth from site.

The VTM concentrate will be transported from the mine to the Metallurgical Plant by rail.

The climate is typically cold during winters (-10°C January average high) and continental during summers (25°C July average high). Average annual precipitation is approximately 1200 mm. Snow is usually present from late October to early May.

The Metallurgical Plant site is characterized by a non-homogenous soil with bedrock at different elevations below the surface.

1.7 Project History

Before BlackRock acquired the Property, two phases of work had been completed specifically on the BlackRock claims. In the 1970s, the first phase consisted of two holes drilled by the Ministry of Natural Resources of Québec on the Southwest Zone. The second phase took place from 1997 to 2001, under the management of McKenzie Bay. Eleven holes were drilled, and a dozen trenches were dug.

In 2006, government geologists undertook a geological compilation at a scale of 1:50,000 over the whole region located south of Lac Chibougamau, including most of the BlackRock Property. The Property has been inactive until the involvement of BlackRock in November 2008 when BlackRock carried out an airborne magnetometer survey covering the entire length of the Lac Doré Complex (Novatem, 2008). An interpretation report was produced in March 2009 by Gérard Lambert Geosciences Ltée (Lambert, 2009). Later that year, the company consolidated all of the historical work completed on the project.

The BRM Metallurgical Plant project received its environment approval in 2019. Assuming a full notice to proceed is issued in May 2023 and start of the construction is in the same year, the construction is expected to be completed in May 2026.

1.8 Geological Setting and Mineralization

The BlackRock Project covers part of the Lac Doré Complex (LDC) within the Abitibi Greenstone Belt at the eastern edge of the Archean Superior Province within the Canadian Shield. The LDC was emplaced as a sill-like mafic layered intrusion formed during the arc-magmatic and collisional stages of orogenesis at 2.727 Ga. It was warped into a broad anticline during continued compressive accretion of the Abitibi-Wawa Terrane between 2.698-2.690 Ga. It has been subsequently affected by low-grade thermal metamorphism related to the accretion contact with the Proterozoic Grenville Province (1.1 Ga).

The Lac Doré Complex is composed of thick sequences of anorthosite that encase the historical copper-gold deposits. The anorthosite is stratigraphically overlain by a layered sequence that contains vanadiferous magnetites interlayered with ferrogabbros and ferropyroxenites. The layered sequence is finally surmounted by a late-stage granophyre and the contact zone with the felsic volcanic rocks of the Waconichi Formation.

BlackRock used geology, geophysics and geochemistry to define the attitude and shape of the orebody and to infer the location and orientation of all significant structural features that affect the mineralized stratigraphy. Within the Blackrock project, all the units form a sequence striking from N30° to N50° and dipping between 45° to 80° to the southeast. Stratigraphic tops face to the southeast. Faults and shear zones trend predominantly in three preferred orientations: E-W and NE to NNE. There are also reports of NW faults. The proximity of the Grenville Front produced late brittle faulted blocks that are well outlined on the BlackRock magnetic survey. These faults generally run parallel to the Grenville Front following the NNE direction.

Fe-Ti-V mineralization occurs at the top of the Anorthosite Zone and the base of the Layered Zone. As expressed by the ground and aeromagnetic surveys, these mineralized horizons extend almost continuously over the entire 17 km length of the Blackrock project. True thickness of the mineralization in the Armitage and Southwest deposits averages 110 m with a minimum of 57 m and a maximum at 195 m. These variations in thickness may be related to primary depositional mechanisms and/or due to pinch and swell (boudinage) style deformation.

The deposit is notably fresh and there is little surface weathering. Ore phase mineralogy consists of titano-magnetite and ilmenite. The main gangue constituents are chlorite, actinolite/grunerite and saussuritized anorthite. There are also remnants of the original mineralogy that consisted of pyroxene, anorthite plagioclase and minor amphibole.

The entire layered sequence displays various scales of rhythmic layering where the dark magnetite/ilmenite layers alternate with less mineralized or barren pale anorthositic or gabbroic compositions. The single most important tool for defining stratigraphy on the BlackRock project is the presence of a well-developed cryptic layering characterized by the variation in the chemical composition. This has formed the basis for BlackRock's stratigraphic interpretation, which is adapted from the stratigraphic model developed by Allard and Girard (1998).

This stratigraphy coupled with the detailed geophysical delineation of the large-scale magnetic anomaly evidences a strong geologic continuity along strike and down dip, giving confidence to the interpretation of the deposit. The geochemical fingerprinting that defines the stratigraphy has also enabled the identification of non-related intrusions, such as later dykes and sills.

1.9 Deposit Types

The geological setting and mineralization encountered in the LDC on the BlackRock Property indicate some similarities with other world-class magmatic Fe-Ti-V oxide deposits associated with layered mafic intrusive rocks.

This family of deposits includes the world class Bushveld complex in South Africa (producing district), the Panzhihua layered intrusion in China (producing district), the Windimurra complex in Australia (producing district), the Maracas Deposit in Brazil (producing), the Skaergaard complex in Greenland (advanced exploration) and the Bell River Archean complex in the Matagami region of Québec (prospect).

1.10 Exploration

Since 2008, BlackRock has carried out extensive exploration work over the Southwest and Armitage Deposits summarized in the following and in Table 1-7:

- Airborne magnetometer survey and digital topography of the entire magnetite bearing envelope over the claims area;
- Compilation of historical work;
- Trenching and channel sampling in the Southwest deposit on three new trenches done by BlackRock and seven old trenches from McKenzie Bay;
- From 2010 to 2012, BlackRock completed three drilling programs over the Southwest Deposit and two on the Armitage Deposit;
- Mineralogical and metallurgical testing at Corem and SGS including Satmagan (measures magnetite content), thin and polished sections, Davis Tube Analysis, XRF, pycnometer and specific gravity;
- Hyperspectral scanning of all 2011/12 drill core;
- Magnetite and V_2O_5 resource estimates for the Southwest (2010) and Armitage (2011) deposits;
- Updated magnetite and TiO_2 resource estimates for the Southwest deposit (2012);
- Magnetite, TiO_2 and V_2O_5 resource estimates for the Southwest and Armitage deposits in the current study.

1.11 Drilling

From 2010 to 2012, BlackRock completed three drilling campaigns over the Southwest Deposit, and two on the Armitage Deposit. Globally, 219 diamond drill holes, totalling 49,680 metres, were drilled on the BlackRock Project. An additional six holes were drilled on the neighbouring Cogitore Resources Inc. Property as part of a condemnation program for waste facilities. All drilling was carried out with NQ size core, except metallurgical (bulk sample) drill holes, drilled with HQ core. A list and detailed description of the drillholes can be found in Table 1-7.

Table 1-7: Summary of BlackRock drilling on the BlackRock Property

Southwest Deposit	# of Holes	Metres
Resource Drilling	83	20,534
Metallurgical (Bulk sample, HQ size)	20	2,985
Geotechnical	8	1,833
Condemnation	6	1,740
Total Southwest	117	27,092
Armitage Deposit	**# of Holes**	**Metres**
Resource Drilling	81	19,573
Metallurgical (Bulk sample, HQ size)	21	3,015
Total Armitage	102	22,588
Total Resource Drilling	164	40,107
Total Metallurgical	41	6,000
Grand Total	219	49,680
Cogitore Resources Inc. Property (extra drilling)	6	1,644

1.12 Sampling Method, Approach and Analyses

BlackRock's 2010 and 2011-2012 drilling programs included an infield QA/QC during sampling that included insertion of blanks and duplicates into the sample stream in addition to the in-lab standards and duplicates that were run. The QA/QC program succeeded in validating the laboratory results for all providers examined. QA/QC data from all drilling programs from 2010 to 2012 indicates that WRA and Satmagan assays at ALS CHEMEX, Corem Inc. and SGS Canada were consistent and reliable. QA/QC data indicate excellent correlation for WRA assays between ALS Chemex and COREM and ALS Chemex and SGS.

Correlation coefficients between original Satmagan and WRA assays and their duplicates range from 0.94349 to 0.99888. Correlation coefficients are above 0.99 for all major elements. All assay data are considered reliable for use in resource estimates and feasibility-level work.

1.13 Data Verification

SGS Geostat visited the Chibougamau core storage location on three occasions from 2010 to 2013 to collect independent samples. During the site visits, control samples of the core were taken for validation testing at SGS Lakefield.

No material issues were found in the database during the validation process conducted by SGS. In terms of QA/QC, SGS Geostat was satisfied with the in-house QA/QC program set up and used by BlackRock Metals from the inception of the development work in 2009. Correlation coefficients of the BlackRock data versus the SGS validation data are all over 0.96.

As a result of its data validation efforts, SGS believes that the drillhole data representing the mineralization intersected by drilling at both the Southwest and Armitage Deposits are appropriate for use in the preparation of the Mineral Resource estimates.

1.14 Metallurgical Testwork – Beneficiation Plant

Metallurgical testwork for the concentrator was carried out at both the COREM laboratory in Québec City and SGS in Lakefield, Ontario, to establish design criteria and to size equipment. The testwork included magnetic separation tests by Davis Tube, Drop-Weight tests, SMC tests and grindability tests. Furthermore, benchscale tests were performed on Southwest and Armitage material in order to determine metallurgical recovery flowsheets. Finally, the proposed flowsheets were confirmed via pilot plant testwork that tested composite samples from each pit (separately). These samples were selected in order to represent the entire orebody. Further testing was performed to validate the flowsheet at MetChib laboratories in Chibougamau, Québec. Roughly 65 tonnes of concentrate was produced during a piloting campaign on the four lithologies present.

1.14.1 Satmagan Correlation Equation

The results of several Davis Tube tests, from each lithology present, were used in conjunction with Satmagan measurements and chemical analysis to develop a correlation equation. Using these results, the total weight recovery of magnetite concentrate can be determined by knowing the Satmagan, as seen in the equation below.

$$BCS\ Zone:\ Wt\ Rec\ (\%)_{Magnetite} = 1.3353 * Satmagan$$

$$MCS\ Zone:\ Wt\ Rec\ (\%)_{Magnetite} = 1.4102 * Satmagan$$

$$TITAN\ Zone:\ Wt\ Rec\ (\%)_{Magnetite} = 1.4692 * Satmagan$$

$$ULS\ Zone:\ Wt\ Rec\ (\%)_{Magnetite} = 1.3527 * Satmagan$$

The aforementioned recovery equations, derived from Davis Tube Analysis (DTA), fits well with the data obtained from the benchscale and pilot plant data from both the Southwest and Armitage test campaigns. The correlations between Satmagan and weight recovery for the Armitage and Southwest Pits are similar and their behaviour has been determined to be the same (thus only one correlation equation has been taken to cover both pits).

The key grindability and hardness information is presented in Table 1-8.

Table 1-8: Summarized grindability results for the Southwest and Armitage Pits

Criteria	Southwest	Armitage
Ball Mill Work Index (WI_B) Design: 80th Percentile	13.6 kWh/t	12.82 kWh/t
Crusher Work Index Design: 80th Percentile	11.4 kWh/t	11.86 kWh/t
Axb Factor Design: 75th Percentile	31.81	30.0

Competency results obtained from the Drop-Weight (DWT) and SAG Mill Comminution (SMC) tests on the material from Armitage and Southwest suggest that the ore from Armitage is more competent than that of Southwest. SAG mill sizing was calculated using empirical power model equations for the Southwest Pit with safety factors to account for potential instabilities in feed grade and hardness.

1.14.2 Vanadium Correlation Equation

The behaviour of vanadium (V_2O_5) in the magnetite recovery process was investigated in order to predict the vanadium grade in the final magnetite concentrate. As vanadium is associated with iron species, the investigation focused on determining the upgrading factor of vanadium in relation to iron in the final concentrate. From these studies, a global concentrate vanadium grade equation was determined as follows:

$$BCS\ Zone:\ \text{Concentrate}\%V_2O_5 = \left(1.3765\left(\frac{62\%Fe}{Head\%Fe}\right) - 0.3149\right) \times \text{Head}\%V_2O_5$$

$$MCS\ Zone:\ \text{Concentrate}\%V_2O_5 = \left(1.7863\left(\frac{62\%Fe}{Head\%Fe}\right) - 0.9135\right) \times \text{Head}\%V_2O_5$$

$$ULS\ Zone:\ \text{Concentrate}\%V_2O_5 = \left(1.8004\left(\frac{62\%Fe}{Head\%Fe}\right) - 0.874\right) \times \text{Head}\%V_2O_5$$

$$TITAN\ Zone:\ \text{Concentrate}\%V_2O_5 = \left(2.3311\left(\frac{62\%Fe}{Head\%Fe}\right) - 1.7015\right) \times \text{Head}\%V_2O_5$$

1.14.3 Magnetite Beneficiation Results

The magnetite beneficiation flowsheet was obtained through benchscale testwork and substantiated via pilot plant trials. A block diagram of the basic stages of the magnetite beneficiation process can be found in Figure 1-2. The pilot plant and benchscale tests behaved as predicted by the Satmagan vs. weight recovery correlation equations and responded very well to conventional magnetic separation techniques.

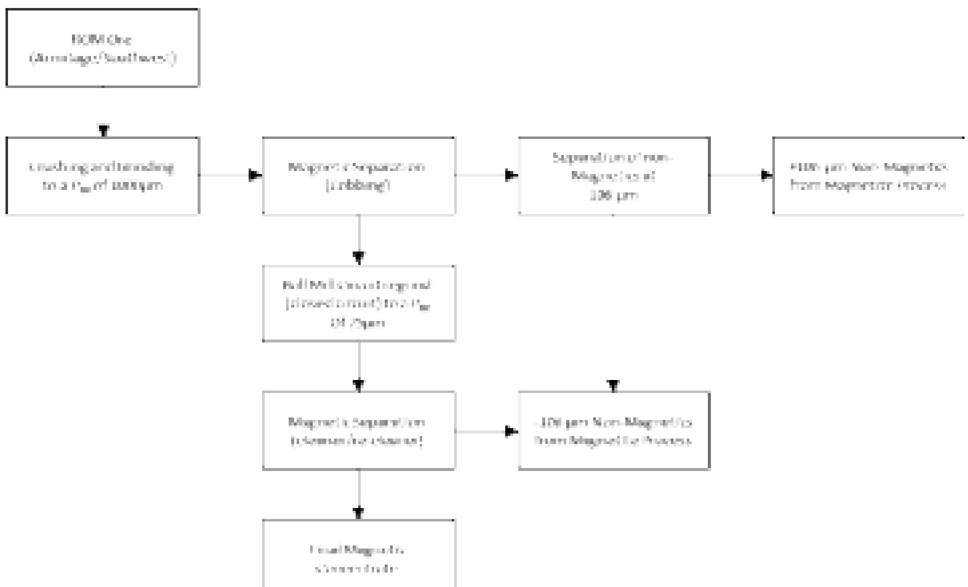

Figure 1-2: Magnetite flowsheet block diagram

Benchscale and pilot plant magnetic separation tests led to a conclusive flowsheet including cobbing (primary magnetic separation), regrinding to a liberation size of 75 µm, cleaning and recleaning stages (secondary magnetic separation), as seen in the figure above. The product specifications and the average testwork results achieved are presented in Table 1-9.

Table 1-9: Magnetic concentrate specification targets and actual pilot plant results

Criteria	Target	Result Southwest	Result Armitage
Fe grade	62 - 65%	61.4% (avg)	62.2%
Magnetic Fe Recovery	n/a	95.9 - 96.9%	95.2%
$SiO_2 + Al_2O_3$	< 3%	2.25% (avg)	2.13%
$(CaO + MgO)/(SiO_2 + Al_2O_3)$	< 0.3	0.21% (avg)	0.22% (avg)
P grade	< 0.06%	0.01% (avg)	0.01%

During benchscale and pilot testwork the proposed flowsheet provided a recovery of between 94 to 97% magnetic Fe. The final grade obtained was around 62% Fe. The Fe grade of the magnetite concentrate is sensitive to the amount of TiO_2 and V_2O_5 found in solid solution within the crystal lattice of the magnetite.

1.15 Mineral Resources

The Southwest and Armitage Deposits are 2.5 km and 3.3 km segments (respectively) of the layered ferrogabbros with vanadiferous magnetite and ilmenite. These deposits are part of the Lac Doré anorthosite complex located about 20 km south of Chibougamau, Québec. The layered sequence is exposed along a NE-SW strike length of about 24 km, a continuous 17 km outcrop of which is on BlackRock Metal's claims. The initial evaluation of the Mineral Resources was supervised by Mr. Michel Dagbert, a QP and employee of SGS at the time. Mr. Dagbert has since retired; but was involved in the recent update of the BRM Deposits (Table 14-22 of Chapter 14) under the supervision of Claude Bisaillon of SGS. Mr. Bisaillon has taken responsibility for Chapter 4 to 12, 14 and 23 of the Report.

1.15.1 Southwest Zone

In the Southwest Zone, the magnetite mineralization is concentrated in up to four magnetite-ilmenite-vanadium rich chlorite altered ferrogabbro cumulate layered units of variable thickness, separated by barren anorthositic layers. The layered intrusion dips on average 75° (from 60° to 90°) to the N130. Its thickness varies from about 100 m to about 300 m perpendicular to strike.

The Mineral Resources of the Southwest Zone are mostly documented by 83 core holes (NQ diameter) totalling 20,534 m and drilled by BlackRock along the 2.7 km strike length of the Southwest Zone. Holes were drilled on 26 N130 sections 100 m apart. On the sections, holes are generally dipping 45° to N310 with three holes per section. The distance between the holes along the dip of mineralized structure is generally between 50 m and 100 m. Data from historical trenches is no longer used in the resource estimation. The cut-off date for data used in the resource model is February 24, 2014.

Along the drill holes, sampling was done according to a composite interval of 3 m. Altogether there are data for 4,344 sample intervals. In the majority of the samples (94%), the percentile magnetite content has been measured by the Satmagan method (%SAT). In a majority of them too (85%), there is a complete XRF determination of major and minor elements, including vanadium. Satmagan measurements were done at the COREM laboratory in Québec City and the SGS laboratory in Lakefield, while the XRF analyses were performed at ALS Chemex in Vancouver. All of these laboratories used 100 g splits of 200 mesh pulp prepared from half cores at the *Table Jamésienne de Concertation Minière* (TJCM) preparation laboratory facility in Chibougamau.

BlackRock has implemented a QA/QC program with duplicate and blank samples. As part of this and earlier study, SGS Canada Inc. collected 69 independent check core samples. Based on those results, SGS is of the opinion that the 4092 Satmagan values reflect adequately the magnetite content of the corresponding composite 3 m hole interval.

As the proposed resource block model is based on estimates of the %SAT in the blocks, in as much as possible, this %SAT should be available in all the 3,006 three -meter DH intervals in the mineralized units. In the 298 intervals where it is missing, one can take advantage of the useable correlation (R^2 from 0.47 to 0.95 depending of mineralized layer) of %SAT and %Fe_2O_3 in the samples that have both %SAT and %Fe_2O_3 analyses in order to calculate the missing %SAT. The regression lines derived from the correlation plots indicate very little magnetite for samples with less than 12%Fe_2O_3 and a predicted %SAT above 20% for samples with more than 45%Fe_2O_3.

The resource estimate of the Southwest Zone is based on the interpolation of the %SAT, %TiO_2, %V_2O_5 and %Fe_2O_3 of small blocks 10 m x 5 m x 7 m, filling the 3D solids of interpreted mineralized units from the same values for drillhole samples within the same solids. The block interpolation is conducted using a standard inverse squared distance method with search ellipsoids parallel to the local orientation of the mineralized layers on sections.

In addition to its %SAT, %TiO_2, %V_2O_5 and %Fe_2O_3 content, each block of the resource model is given an estimated concentration of the following secondary elements: %Al_2O_3, %S and %P_2O_5. Those estimates are also derived by inverse squared distance interpolation from available values in samples. In the case of alumina with a significant negative correlation with magnetite, the block estimate is also dependent of the magnetite grade estimate of the block through the interpolation of regression residuals.

The resource blocks are categorized into the Measured, Indicated and Inferred resources. The solid of the Measured resources is drawn 50 m below the base of the drillholes on each section while that of Indicated resources does not extend more than 100 m from that base of the drill holes. The proposed categorization of the resources is supported by a comparison of current estimates and the previous ones in 2010-11: with an almost doubling of the DH information, differences of tonnage or grade above cut-offs do not exceed 10%.

Estimated resources are given at any given %SAT cut-off applied to block estimates. Conversion of volume into tonnage uses density derived from the %SAT, %TiO_2 and %Fe_2O_3 of each block. That relationship is established from a set of pycnometer density data for 526 core samples. The average density of the resource blocks is 3.5 t/m^3.

1.15.2 Armitage Zone

As in the SW Zone, the magnetite mineralization of the Armitage Zone is concentrated in up to four magnetite-ilmenite-vanadium rich chlorite altered ferrogabbro cumulate layered units of variable thickness separated by barren anorthositic layers. The layered intrusion dips on average 67.5° (from 60° to 75°) bearing N-160 degrees. Its thickness varies from 100 m to 300 m perpendicular to strike.

The Mineral Resources of the Armitage Zone are documented by 81 core holes (NQ diameter) totalling 19,573 m and drilled by BlackRock along a 2.5 km strike length of the Armitage Zone. Holes were drilled on 27 N160 sections 100 m apart. On the sections, holes are generally dipping 45° to N340 with generally three holes per section. The distance between the holes along the dip of mineralized structure is generally between 50 m and 100 m. The cut-off date for data used in the resource model is February 11, 2013.

Along the drill holes, sampling was done according to a composite interval of 3 m. Altogether; there are data for 3,980 sample intervals. In the majority of the samples (99%), the percentile magnetite content has been measured by the Satmagan method (SAT). In a majority of them too (99%), there is a complete XRF determination of major and minor elements, including vanadium. Satmagan measurements were done at the COREM laboratory in Québec City and the SGS laboratory in Lakefield, while the XRF analyses were performed at ALS Chemex in Vancouver. All those laboratories used 100 g splits of 200 mesh pulp prepared from half cores at the *Table Jamésienne de Concertation Minière* (TJCM) preparation laboratory facility in Chibougamau.

BlackRock has implemented a QA/QC program with duplicate and blank samples. As part of this study, SGS Canada Inc. collected 50 independent check core samples. Based on those results, SGS is of the opinion that the 3,932 Satmagan values reflect adequately the magnetite content of the corresponding composite 3 m hole interval.

Although no longer necessary since a %Satmagan measurement was taken for all the Whole Rock Analysis (WRA) samples, the correlation of %SAT and %Fe_2O_3 was done, and the result is similar to that obtained in the SW Zone. Similarly, density and vanadium continue to be strongly correlated to the %SAT. Correlations between %SAT and other elements with an estimate required for the blocks of the resource model were carried out.

As in the SW Zone, the magnetite resource estimate of the Armitage Zone is based on the interpolation of the %SAT, %TiO_2, %V_2O_5 and %Fe_2O_3 of small blocks, 10 m x 5 m x 7 m, filling the 3D solids of interpreted mineralized units from the same values for drillhole samples within the same envelope. The block interpolation is conducted using a standard inverse squared distance method with search ellipsoids parallel to the local orientation of the mineralized layers on sections.

As in the SW Zone, each block of the resource model is given an estimated concentration of the following secondary elements: %Al_2O_3, %S and %P_2O_5. Those estimates are also derived by inverse squared distance interpolation from available values in samples. In the case of alumina with a significant negative correlation with magnetite, the block estimate is also dependent of the magnetite grade estimate of the block through the interpolation of regression residuals.

As in the SW Zone, the solid of the Measured resources is drawn 50 m below the base of the drillholes on each section while that of Indicated resources does not extend more than 100 m from that base of the drill holes. The proposed categorization of the resources is supported by a comparison of current estimates and the previous ones in 2010-11. With an almost tripling of the DH information, differences of tonnage or grade above cut-offs do not exceed 10%.

Estimated resources are given at any given %SAT cut-off applied to block estimates. Conversion of volume into tonnage uses density derived from the %SAT, % TiO_2 and %Fe_2O_3 of each block. That relationship is the same as the one derived in the SW Zone and it is supported by a set of pycnometer density data for 544 core samples. The average density of the resource blocks is slightly less than 3.5 t/m^3.

1.15.3 Resource Estimates

Following CIM guidelines, resources are made part of the mineral inventory, with "a reasonable prospect of economic extraction". In practice, a final pit shell is optimized with reasonable technical and economic conditions, and resources are made of all blocks within that shell and above an economic cut-off corresponding to these conditions.

Optimized pit shells have been produced by BBA using the same technical and economic conditions as those used to define the reserves (See Section 15). The only distinction is that the shells for resources use Inferred Resources, whereas those for reserves do not. The cut-off used in the resource statement is 10% Satmagan, i.e., the same as the reserve cut-off. The statement of resources defined in those conditions is found in Table 1-10. The effective date of this Mineral Resource Statement prepared for the BlackRock Project is August 26, 2022.

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. Mineral resource estimates may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, and other relevant issues.

Table 1-10: Estimated Resources of the BRM Deposits

Dep.	Cat.	Vol. (Mm^3)	Ton. (Mt)	Dens. (t/m^3)	%SAT	$\%TiO_2$	$\%V_2O_5$	$\%Fe_2O_3$	$\%Al_2O_3$	$\%P_2O_5$	%S
SW	Meas	40.0	145.0	3.63	17.9	7.9	0.48	41.2	13.0	0.033	0.17
SW	Ind	10.3	37.4	3.63	17.9	7.9	0.47	41.1	12.9	0.032	0.18
SW	M+I	50.2	182.4	3.63	17.9	7.9	0.48	41.1	13.0	0.032	0.17
SW	Inf	10.3	45.2	4.39	18.2	8.1	0.48	41.5	12.8	0.032	0.17
AE	Meas	40.1	142.2	3.55	15.4	7.1	0.41	36.9	14.3	0.028	0.28
AE	Ind	8.7	30.8	3.54	14.8	7.2	0.40	36.4	14.2	0.026	0.32
AE	M+I	48.8	173.1	3.55	15.3	7.1	0.41	36.8	14.3	0.028	0.29
AE	Inf	7.9	28.1	3.55	14.7	7.5	0.39	36.9	14.0	0.026	0.37
All	Meas	80.0	287.2	3.59	16.7	7.5	0.45	39.0	13.7	0.030	0.22
All	Ind	19.0	68.3	3.59	16.5	7.6	0.44	39.0	13.5	0.029	0.24
All	M+I	99.0	355.5	3.59	16.7	7.5	0.44	39.0	13.6	0.030	0.23
All	Inf	18.2	73.3	4.02	16.8	7.9	0.44	39.7	13.3	0.029	0.25

Resources are defined at a minimum cut-off of 10% Satmagan. Due to the necessary rounding of estimates, the rounded totals may slightly differ from the sum of rounded individual estimates.

The Mineral Resource estimate was completed by Claude Bisaillon, P.Eng. (OIQ #116407) formerly from SGS Geostat at the time of writing the Report, an independent Qualified Person as defined in the Canadian National Instrument 43-101.

1.15.4 Upside Potential of the Property

In addition to the SW and Armitage Zones, potential for additional magnetite resources on the BlackRock Property is promising since it covers about 17 km of the 24 km strike length of the Lac Doré complex. This geological favorability is supported by firstly, excellent logistics and local infrastructures (town, power access), secondly data availability including metallurgical tests.

1.16 Mineral Reserve Estimate

The evaluation of the open pit reserves was supervised by Isabelle Leblanc (BBA Inc.). The open pit reserves were generated from the block model provided by SGS Geostat for the Southwest Zone. The block model consists of blocks measuring x=10 m by y=5 m by z=7 m.

The pit optimization and the detailed engineered pit design were carried out to convert Mineral Resources into Mineral Reserves for the Southwest Pit only. For the Armitage Deposit, the level of detail of the mine plan and quantity of metallurgical testworks and geotechnical analysis are not sufficient to convert the In-pit Resources into Mineral Reserves.

In accordance with the NI 43-101 standards of mineral classification, the measured and indicated resources inside the final pit limits for Southwest have been transferred into Proven and Probable reserves after the application of the modifying factors. The open pit Mineral Reserves for the Southwest Pit are shown in Table 1-11. The total Mineral Reserves for the Southwest Pit amounts to 127.8 Mt proven and probable at a grade of 18.8% SAT, 0.46% V_2O_5 and 7.8% TiO_2. The Southwest Pit reserves are sufficient for a 39-year mine life at an average milling feed rate of 3.3 Mt.

Table 1-11: Southwest Pit Mineral Reserves

Mineral Category	Tonne (kt)	SAT (%)	Fe_2O_3 (%)	TiO_2 (%)	V_2O_5 (%)	Expected V_2O_5 in concentrate (%)	Expected metallurgical weight recovery (%)
Proven	123, 900	18.9	40.2	7.7	0.46	1.34	26.0
Probable	3, 900	17.9	40.3	8.1	0.42	1.24	25.0
Total Reserve	127, 800	18.8	40.2	7.8	0.46	1.33	26.0

Notes:
1. The effective date of the Mineral Reserve estimate is October 30, 2022.
2. The Mineral Reserves 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 in May 2014.
3. Qualified Person: The Mineral Reserve statement was prepared by Isabelle Leblanc (OIQ #144395) of BBA, an “independent qualified person”, as that term is defined by National Instrument 43-101.
4. Open pit Mineral Reserves have been estimated using a 0.29 net revenue factor apply on High Purity Pig Iron (HPPI) price of 670 CAD/t of product, a Ferrovanadium (FeV) price of 54,341CAD/t of product, a foreign exchange rate of CAD1.33 to USD1.00.
5. Open pit reserves have been estimated using a cut-off grade of 10% Satmagan.
6. The LOM strip ratio is 2.2.
7. Reserves are derived from the Satmagan Resources Statement (182.4Mt of resources in the Measured and Indicated categories at a cut-off grade of 10%) prepared by Claude Bisaillon (OIQ #116407) from DRA Americas formerly from SGS Geostat.
8. The reference point for the Mineral Reserves is the crusher feed.
9. Expected % V_2O_5 in concentrate and % metallurgical weight recovery are based on Davis Tube Analysis (DTA) metallurgical testwork. The formulas by mineralized units, are presented in Chapter 13.1.3.
10. BBA is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Reserves estimate, except for those already discussed in this Report.

1.17 Mining Methods

1.17.1 Open Pit Mine Design

The Southwest Pit will be mined using a conventional open pit drill and blast, load and haul mining method using a drill, truck and shovel and loader mining fleet. To reduce power plant load requirements and also for ease of movement around the pit, a fleet of diesel-powered equipment has been selected. The current Feasibility Study is based on an "Owner-Operator" approach except for the pre-production period which will use a contractor for the drilling and extraction of the rock as well as construction of the infrastructure.

1.17.2 Mining Operations

The production schedule was developed based on a mill throughput of approximately 3.3 Mtpy for the Southwest Pit. The target concentrate production was approximately 856 ktpy of 18% to 20% SAT. The mine will operate 365 days per year, less 5 days accounted for bad weather, 7 days a week and 2 x 12-hour shifts per day.

The mining production schedule is based on a pre-stripping period approximately 24 months in length and a mine life of 39 years for Southwest Pit.

The primary production fleet consists of the following type of equipment or their equivalent to be selected during the detailed engineering stage:

- Blast Hole Drills: 215.9 mm (8 ½ in) diesel powered DTH blast hole drills;
- Loading Equipment: 9 m^3 capacity hydraulic shovels (diesel) and 10.7 m^3 capacity wheel loaders;
- Haul Trucks: 90 t capacity.

The Auxiliary Mining Fleet consists of the following:

- Track dozers: 475 HP class machine;
- Motor Graders: 16 ft class machine;
- Wheel Dozer: 570 HP class machine;
- Water Truck 45 kL.

The selection and sizing of the production fleet is based on cycle time estimations, equipment mechanical availability and utilization factors, as well as average yearly haulage profiles.

1.17.3 Mine Manpower Requirements

The manpower requirements for the mine are divided into three categories: the in-pit mine operations staff, the mine maintenance staff, and the mine technical services staff. The number of personnel reaches a peak of 210 during Year 16 of operations.

The number of operators required for the major mining equipment (haul trucks, shovels, and drills) was determined according to the number of operating units and number of rotations during which time the equipment is in operation. Most of the operators for the major mine equipment are based on a four-crew rotation. Hourly maintenance employee requirements were determined based on the number of machines to be maintained.

1.18 Mine and Beneficiation Plant

1.18.1 Recovery Methods

Run of mine ore will be screened to remove fines prior to being crushed by a jaw crusher. The two streams (fines and crushed ore) will be recombined and then stored in a covered stockpile. The crushed ore from the stockpile will be withdrawn from beneath with belt feeders and ground in a semi-autogenous (SAG) mill in a closed circuit with a scalping screen and a horizontal classification screen. The magnetic iron minerals will be recovered by a primary stage of low-intensity magnetic separation (cobbing). The cobber concentrate will undergo further size reduction in a ball mill to achieve the determined liberation size (P80 75 µm) to separate the vanadium titanium magnetite (VTM) from the gangue material. A second magnetic separation step (cleaning and re-cleaning) will then achieve the final grade of the concentrate.

The final VTM concentrate product will be dewatered using disc filters and the dewatered magnetite product will be loaded into trucks, where it will be transported to a train loadout and later to the Metallurgical Plant for transformation into high purity nodular pig iron, vanadium rich slag and titanium slag.

The tailings from the magnetic circuits will be pumped to the tailings management facility to be decanted (tailings pond followed by a polishing pond). The water reclaimed from the polishing pond and the tailings thickener overflows will be recycled to the process water tank.

1.18.2 Infrastructure

1.18.3 Buildings and Infrastructure

Several roads will be built, and existing roads will be upgraded to allow for site traffic, maintenance and material transport.

The pad around the jaw crusher will be located approximately 300 m from the Southwest Pit edge. The crusher feeds a covered stockpile which acts as a buffer between the crusher and the concentrator.

The concentrator will include the process area, in addition to other service facilities such as the electrical rooms, HVAC, compressor room, and boiler rooms, service building, and the concentrate load-out system.

Other types of facilities on site will include the medical station, metallurgical laboratory, offices, warehouse, truck wash area, mine garage and concentrate truck load out.

1.18.4 Site Utilities

Electricity will be supplied by a new 22 km, 161 kV power line. This line will connect to the existing 161 kV Hydro-Québec line #1627 (Obalski/Otabogamau), servicing Chibougamau.

The main substation will be located near the concentrator building, where the large electric loads, fed by the main substation, are installed. Two main 161-25 kV, 24/32 MVA transformers will be used for a combined firm power of 24 MVA. The electrical distribution to the site infrastructure will consist of a dedicated 25 kV OHL distribution network.

The open pit mine will not require electrical distribution, however, the explosives storage centre will require an electrical connection via the overhead line. The major mining equipment, vehicles and pumps will be operated using diesel fuel.

The site will also have skid mounted potable water treatment which consists of filtration, chlorination, and UV sterilization units to produce potable quality water.

Light fuel and propane will be delivered to site by road tankers and stored in several tanks. The mine equipment fuel will be stored in a tank farm consisting of five tanks at the mine garage, where the first of two double-walled Gasboys will also be located. The second will be placed at the concentrator.

1.18.5 Site Water Management

A global water balance was calculated using the concentrator mass balance. A site-wide balance using environmental/topographic data for the surrounding area will need to be evaluated in a water management site development plan.

The BlackRock water balance has been designed to maximize the recycling of water within the process, with any shortfalls being made-up by water collected from precipitation and run-off in the surrounding watershed and from water reclaimed from the open pit mine.

1.18.6 Tailings Management Facility

The tailings pond will be located west of the Southwest Pit. Dams will be built around most of the perimeter. During the life of the Southwest Pit, 55.1 Mm^3 of tailings will be pumped to the tailings pond. The tailings dam construction uses the downstream method.

At the tailings pond, the water in the pond will consist of a mixture of process water, mine water and rainwater falling on the surface of the pond. This pond also serves to collect the general run-off of the entire BlackRock site, which will be essential to serve as reclaim water to the process. This water will be transferred by pumping into the polishing pond to allow for further settling of particles. Water will be pumped from the polishing pond to the concentrator, to be reused. Any excess water not required for process make-up will be released into the environment after passing through the treatment and monitoring pond. The polishing pond overflow will be chemically treated prior to being placed in the treatment pond. The treatment pond will serve to monitor the effluent to ensure the standards for mining effluents are met.

1.18.7 Market and Contracts

1.18.7.1 Market

BlackRock will be transporting all the VTM concentrate produced to its Metallurgical plant. The concentrate will be transformed into high purity pig iron, Vanadium and titanium slag.

1.18.7.2 Contracts

The CN Railway has a Common Carrier Obligation and the Tariffs are ruled by Transport Canada. Negotiations are underway with CN. A Letter of Intent was signed with Roberval Saguenay Railway (RSR) in July 2013. Negotiations for the Tariffs are ongoing between BlackRock and RSR. The railway section with the Port Authority was included in the LOI and signed with the Port Authority.

BlackRock signed an Impact and Benefits Agreement (IBA), called the BallyHusky agreement on June 20, 2013, with the Oujé-Bougoumou Cree Nation, the Grand Council of the Crees (Eeyou Istchee) and the Cree Regional Authority, for the future development of the BlackRock project in the Eeyou Istchee territory. This agreement was updated in March 2015 to include the Second Transformation Plant.

BRM also signed two Partnership and Development Agreements with the Innu First Nations of Pekuakamiulnuatsh (community of Mashteuiatsh) and Pessamit and Essipit in August 2020.

A declaration of partnership cooperation agreement was signed with the municipalities of Chapais and Chibougamau in May 2013. This partnership agreement was established to promote and develop a sustainable project, taking into account the social, economic and environmental aspects.

1.18.7.3 Environmental and Permitting

In 2011, an environmental impact (EI) statement was submitted for the exploitation of the Southwest Pit. BRM obtained its Certificate of Authorization (CoA) from the provincial government in 2013 (EIA approval). In October 2014, the project was amended to change the wording and to extend the delay of certain conditions and a modified CoA was issued in 2015. Following the decision to build a secondary process plant for the transformation of magnetite, vanadium and titanium concentrate into high purity pig iron and ferrovanadium in Saguenay, another amendment of the CoA was requested in December 2017 to adjust the project to the production rate that could be achieved by the secondary process plant, for the new tailings management strategy, to include means of transporting the concentrate (road and railway) and for other minor modifications. An Environmental Authorization (EIA) Decree was issued in 2019 for the Metallurgical Plant and is still valid.

The EIA approval contains 40 conditions. One is no longer applicable (worker camp), nine are on mitigation measures or monitoring and 30 conditions require that additional information be submitted to the *Ministère de l'Environnement et de la Lutte contre les changements climatiques* (MELCC) for information, or approval, within 1 or 2 years of the issuance of the last amended CoA and/or prior to the construction phase. Of those 30 conditions, 11 conditions are completed and/or approved by the MELCC.

BRM has submitted its wetlands compensation plans in May 2020 for approval to the MELCC. Comments were received in January 2021, mostly concerning mining titles (*Ministère de l'Énergie et des Ressources naturelles* - "MERN") near or at the proposed locations and commitments. BRM is presently waiting for responses from the MERN as to the acceptability of the projects located near the sites under mining titles (borrow pits and old mine site).

Under the *Mining Act*, a person who performs prescribed exploration or mining work must submit a closure plan for the land affected by their operations, subject to approval by the MERN and is conditional upon receipt of a favourable decision from the MELCCFP. This approval is required for the release of the mining lease and the mining operations to begin (including the construction phase).

1.18.8 Capital Costs

Capital costs were calculated for the mining equipment, processing equipment and all required infrastructure, based on a basic engineering, and material take-offs, from bids received by vendors and contractors, and data from historical projects.

The initial capital cost estimate does not include taxes, replacement capital or additional capital requirements after commissioning and start-up. The reference date for the capital cost estimate updates is October 21, 2022. The 2022 refresh has not revisited or modified the estimate as it pertains to scope, quantities, unit construction hours, construction work week, or productivity factors.

The expected accuracy range for the present capital cost estimate is -10%/+15% and meets the AACE Class 2 requirements and is established on the technological complexity of the project, appropriate reference information, and the inclusion of an appropriate contingency (P50 using a Monte Carlo simulation).

For the work to be accomplished at the beneficiation plant, estimated labour costs are calculated on an assumption of an average of 50 hours per week. For the process plant area 6000, the rates are calculated as per Québec Construction Collective Agreement.

Indirect costs include, but are not limited to, EPCM, the construction of the temporary construction complex, equipment transportation freight and spares parts, and contingency.

The purchase of the initial fleet of mining equipment required in Year 0 and Year 1 are considered as the initial mining cost. The rail loadout facilities have also been accounted for in this manner and as such are not included in the capital cost estimate.

A breakdown of direct and indirect costs for the Beneficiation Plant is shown in Table 1-12.

Table 1-12: Beneficiation Plant capital costs summary by area

Area	Area Description	Total (CAD $M)
0000	Off-site – excluding rail preparation	4.489
0000	Off-site – rail preparation	3.124
1000	Infrastructures	10.413
2000	Administration and Services	12.082
3000	Mine (excludes all mine equipment)	15.272
4000	Crushing	23.301
5000	Stockpiling and Conveying	22.356
6000	Processing Plant and Load-out System	139.578
7000	Tailings and Water Management	28.832
Subtotal Direct Costs		**259.446**
8000	Owner's Costs	42.766
9100	EPCM Services	13.657
9200	Construction Indirects	8.553
9500	Commissioning	3.058
9900	Common Distributables (Freight, spares)	10.371
Subtotal Indirect Costs		**78.406**
9800	Contingencies	35.304
Grand Total		**373.156**

1.18.9 Operating Costs

The operating expenditures (OPEX) for the Beneficiation Plant encompass the open pit mine, process plant and general and administrative. Transport and handling, rail and port facilities operating costs and leasing (except for mining equipment) have been provided by BRM. Please note that the figures in the following tables may not add up due to rounding.

The Beneficiation Plant operating costs estimate base currency is Canadian dollars.

Table 1-13 and Table 1-14 summarize the annual operating costs for the first ten years, as well as for the Life of Mine (LOM).

Table 1-13: Operating costs summary (Life of Mine)[3]

Cost Area	Average Operating Costs (CAD)		
	$M/y	$/t milled	$/t Fe Conc.
Mining	55.99	14.83	57.09
Process	27.63	8.47	32.47
General and Administration	8.59	2.62	10.10
Leasing [2]	1.41	0.53	2.02
Transport Logistics[1]	25.22	7.70	29.64
Iron Concentrate Rail Car Maintenance	0.16	0.05	0.20
Other (coarse tailings, environmental)	0.56	0.17	0.65
Total	**119.6**	**34.4**	**132.2**

Notes:
1. Costs calculated/obtained by BlackRock Metals
2. Leasing includes costs for the train loadout and rolling stock over the life of mine (LOM).
3. Average costs include expenditures in the pre-production period.

Table 1-14: Operating Costs Summary (First ten years)[3]

Cost Area	Average Operating Costs for Y1-Y10 (CAD)		
	CAD $M/y	$/t milled	$/t Fe Conc.
Mining	47.70	13.33	50.30
Process	26.98	8.56	32.31
General and Administration	8.59	2.62	10.10
Leasing [2]	3.00	0.23	0.90
Transport Logistics[1]	25.22	7.70	29.64
Iron Concentrate Rail Car Maintenance	0.14	0.05	0.17
Other (coarse tailings, environmental)	0.56	0.17	0.65
Total	**112.2**	**32.7**	**124.1**

Notes:
1. Costs calculated/obtained by BlackRock Metals
2. Leasing includes costs for the train loadout and rolling stock for the first ten years.
3. Operating costs for the first ten years do not consider any expenditures in the pre-production period.

1.18.9.1 Mining

Mining costs include the operating and maintenance costs of the equipment, as well as costs of blasting, personnel, and other costs such as allowances for dewatering, clearing and grubbing costs. Some key supply prices used in the mining operating cost estimates are as follows:

- Diesel Fuel: $1.5/litre;
- Emulsion: $121/100 kg (based on quote);
- Equipment unit operating and maintenance and blasting costs were developed from quotations received from suppliers, BBA's internal database of similar projects and experience. The mining operating costs over the first ten years (the PP costs are excluded) are estimated to be $13.28/t milled or $50.16/t concentrate (the PP costs are excluded). The average LOM mining operating cost is $14.57 /t milled or $46.11 /t concentrate. A breakdown of the average mining operating costs per tonne can be found in Table 1-15.

Table 1-15: Annual cost breakdown for mine operating costs

Cost Area	Mining Operating Costs (CAD)			
	$M/y	$/t mined	$/t milled	$/t concentrate
Labour	17.50	1.66	5.33	20.53
Loading	4.24	0.40	1.30	4.99
Hauling	11.91	1.13	3.64	14.00
Drilling	2.97	0.28	0.91	3.49
Support and Service Equipment	2.93	0.28	0.89	3.44
Blasting	6.22	0.59	1.90	7.31
Dewatering	0.61	0.06	0.18	0.71
Grade Control	0.62	0.06	0.19	0.73
Miscellaneous	0.73	0.07	0.22	0.85
Total	**47.70**	**4.52**	**14.56**	**56.06**

Notes:
1. The cost for the contractor is only for the Pre-Production period.

1.18.9.2 Process Operating Costs

The process operating costs are based on metallurgical testwork, the mine plan, a recent salary survey, literature and recent supplier quotations. The basis of estimate also includes quoted prices for reagents and consumables, as well as industrial and internal references for consumption rates. The life of mine operating cost breakdown is shown in Table 1-16.

Table 1-16: Process operating costs summary (Life of Mine)

Cost Area	Process Operating Costs (CAD)	
	Overall CAD $M/y	Magnetite $/t Fe Con.
Consumables	**8.4**	**9.8**
Spares	3.0	3.5
Maintenance and Parts	2.5	3.0
Reagents	2.8	3.3
Grinding Media	**6.1**	**7.2**
Personnel	**4.5**	**5.2**
Utilities	**7.4**	**8.6**
Power	6.4	7.5
Fuel	1.0	1.1
Sampling	**0.6**	**0.7**
Contracts	**0.7**	**0.9**
Total	**27.6**	**32.5**

1.18.9.3 General and Administrative

The general and administrative (G&A) costs include site and infrastructure maintenance, site electrical power, labour, insurance and legal expenses, health and safety expenses, laboratory and environmental costs, as well as miscellaneous expenses. As in Table 1-17, the G&A costs are broken down as follows:

- Infrastructure and maintenance;
- Fuel;
- Power;
- Manpower;
- Insurance and legal;
- Miscellaneous supplies;
- Laboratory;
- Environment.

Table 1-17: G&A costs summary (Life of Mine)

Cost Area	G&A Costs (CAD)	
	Overall CAD $M/y	Magnetite $/t Fe Con.
Infrastructure and Maintenance	1.5	1.8
Fuel	0.8	0.9
Power	0.5	0.6
Personnel	2.1	2.4
Contracts	1.2	1.4
Insurance and Legal	0.8	0.9
Miscellaneous Supplies	1.3	1.5
Laboratory	0.2	0.2
Environment	0.1	0.2
Total	**8.6**	**10.1**

1.18.9.4 Coarse Tailings Manipulation

Coarse tailings will be used to aid in the construction of the tailings dykes over the life of the mine. Coarse tailings handling operating costs were determined in collaboration with BlackRock and were estimated to be $0.50/t of tailings. The use of a hydraulic excavator, a dozer and a foreman at 50% availability was taken into account in the estimation.

1.19 Metallurgical Plant

1.19.1 Metallurgical Testwork

Responsibility for testwork performed as a part of the Metallurgical Plant Feasibility Study was given to Tenova Core. Again, COREM laboratory in Québec City was selected in addition to Tenova HYL in Monterrey, Mexico and Tenova Pyromet in Midrand, South Africa. Testwork performed included pelletizing tests (COREM), direct reduction bag tests (Tenova HYL), smelting and refining tests (Tenova Pyromet) and titanium slag tests (Symphony Trade via Tenova Pyromet). Through benchscale tests, pilot plant testwork and simulation the metallurgical recovery flowsheets were designed and confirmed. Tenova was given 5 t of vanadium titanium magnetite (VTM) concentrate with which to perform the appropriate tests.

1.19.1.1 Pelletizing Test Results

COREM was charged with characterizing the concentrate and performing the necessary pelletizing testwork and their findings were positive. Their balling and firing standards have been in place since the 1960’s and their results indicated very good green ball quality with reproducibility at commercial scale. The average drop number was found to be 15, demonstrating a very durable green ball formed from the as-received concentrate.

Regrinding tests were performed on the concentrate due to the coarse nature of the concentrate and comparison tests between as-received and re-ground concentrate demonstrated little risk associated with no re-grinding circuit. The pellets also indicated low sticking tendency and a very low swelling tendency, which are both favourable for HYL direct reduction. The pellets had lower than anticipated reducibility and that requires a higher operating temperature and/or higher gas recycle rate.

1.19.1.2 Direct Reduction Test Results

The bags test results and production campaign demonstrated that the BRM pellets produced from VTM concentrate reach an average metallization of 90.0% when processed under the normal plant operating conditions of the Ternium DR Reactor. The average DRI quality from the bag tests results can be seen in Table 1-18.

Table 1-18: Average quality of the reduced pellets

Pellet	% Met	%C	%S
BRM	90	3.58	0.0069

The friability of the pellets was quite low. The carbon content of 3.58% was expected for the process. The operating conditions selected for the BRM reduction plant will result in even higher levels of carbon in the DRI, which is a highly desirable and important factor for later steps in the iron-making process.

1.19.1.3 Smelting Test Results

Test work successfully confirmed that the BlackRock ore concentrate could be used to manufacture commercially acceptable products, as defined by standard industrial practices in the global steel industry and Ferro-alloy industries. In addition to producing high purity pig iron, another objective was the production of ferrovanadium and the testwork procedure applied reflects this. However, the vanadium slag will no longer be further refined onsite to produced FeV, it will be transformed at a third-party vanadium conversion plant to FeV and as a result the related testwork will not be discussed.

Smelting was conducted at laboratory scale and in the DC arc furnace. The smelting campaign was conducted in a 100 kVA DC furnace operating at 30 kW over a period of two days. The pig iron showed an average assay of 93.78% Fe and the slag showed an average TiO_2 content of 39.0%. Primetals performed simulations to determine whether a ladle furnace is required to achieve the desired production product grades. The simulation results demonstrated that the ladle furnace is not required, and a converter would suffice to produce the desired products. Testwork conducted by Mintek clearly showed that vanadium could be recovered from the pig iron to obtain the desired vanadium grade in high-purity pig iron concentrate.

1.19.2 Recovery Methods

VTM concentrate from the BRM Beneficiation Plant, in Chibougamau, will be transported to the metallurgical plant, in Saguenay, for a secondary transformation into high purity nodular pig iron, vanadium rich slag and titanium slag.

VTM concentrate arrives at the Metallurgical Plant by rail and will be conveyed to a concentrate storage area. Concentrate will be reclaimed and conveyed to the pelletizing plant, where mixing with additives, balling, indurating and hearth layer separation result in the formation of green pellets. Pellets will be screened, coated in cement and fed to the Energiron Direct Reduction module.

The DRI product will be fed directly into the AC electric furnace, which will be operated as an open slag bath furnace (OSBF). Titanium slag will be tapped from the OSBF, treated with FeSi producing higher purity titanium slag and a metal alloy strip.

Pig iron will be tapped from the furnace and transferred to the converter. Vanadium rich slag will be tapped from the converter, cooled and will thereafter be fed to a crushing, grinding and metal recovery circuit. The vanadium slag will be screened, bagged and stored prior to shipping the vanadium to a third-party vanadium transformation plant to produce FeV. The final product of high purity pig iron (target Fe 95.74%) will be tapped, cooled and fed to the pig caster in order to produce ingots, which are stored prior to sale.

1.19.3 Infrastructure

1.19.3.1 Buildings and Infrastructure

Access roads to the port, railway, and potable water are currently available. The main road will connect the site to the dock of Port of Grande-Anse. On-site roads will be built to allow for site traffic, maintenance and material transport. The roads on-site will respect the international road technical specifications. On-site roads drainage will be possible due to a ditch drainage network.

The Metallurgical Plant will include a VTM concentrate receiving and storage area, a reclaiming and mechanical handling system, a pelletizing plant, direct reduction plant, and a smelting and refining plant. Other types of facilities on site will include administration buildings, a warehouse, and a laboratory.

The city of Saguenay is to build a pipeline to transport potable water to the metallurgical plant. The potable water line will also be used as a backup for the process water line. Hydro-Québec is to build a new electrical power transmission line and Énergir a natural gas pipeline.

1.19.3.2 Site Utilities

Electricity will be supplied by a new 9 km, 161 kV power line. This line will connect to the existing 161 kV Hydro-Québec line #1640-1641, servicing the Saguenay region.

The main substation shall be designed to allow future expansion and meet Hydro-Québec requirements stated in under "Technical Requirements for Customer Facilities Connected to the Hydro-Québec Transmission System". The electrical distribution network dedicated to the site infrastructure will consist of multiple substation stations feeding power to all three areas of the Metallurgical Plant at different voltage intensities. Three main 161-34.5 kV, 75 MVA furnace transformers will be used.

The major mobile equipment such as vehicles will be operated using diesel fuel.

The site will incorporate a water treatment and cooling plant which consists of demineralization, filtration, scrubbing, cooling towers and clarifier units to produce potable quality water.

Natural gas will be used as a reductant for the direct reduction process and a fuel for the process gas heater, pelletizing plant, and heat production in the buildings during the winter. Natural gas will be delivered via a pipeline to the Metallurgical Plant site.

1.19.3.3 Site Water Management

A global water balance was calculated using the Metallurgical Plant mass balance. A site-wide balance using environmental/topographic data for the surrounding area will need to be evaluated in a later Feasibility Study.

The water treatment plant will be designed to be a zero-discharge operation. To minimize rejects to the environment, the BlackRock water balance has been designed to recycle quenching cooling water, hydrocyclone reject water, and identify an excess of purge water which can be used as make-up water.

1.19.3.4 Solid Waste and Off-gases Management

All air and solid emissions from the Metallurgical Plant will be accounted for. Waste slag and dust from the dust collectors will be stored in an engineering area. Access for monitoring and inspection will be provided in the waste disposal engineering areas. Off-gases will report to a gas cleaning circuit.

1.19.4 Market Studies and Contracts

Market studies from Woodmac and Project Blue were used to analyze the market with reference to the Steel, Pig Iron and Vanadium market. The impact of the Ukrainian and Russian war was also part of the market analysis to determine the potential future impact on the Iron and Steel sectors.

1.19.5 Environmental and Permitting

The project falls under the southern Québec regime. The purpose of the Environmental Assessment is to allow the relevant regulators to properly assess the impact of the project and to seek input from local stakeholders on the proposed development.

In this context, BlackRock submitted a Project Notice and received the Guideline issued by the BAPE via the MDDELCC.

An EIA (Environmental Impact Assessment) was filed with the MELCC under the southern regime, since the selected site is in Saguenay. Public Hearings were held in 2018 and a Decree (Certificate of Authorization) was received in April 2019, and is still valid, for the construction of the metallurgical facility. Once the provincial administrators have issued authorizations for project development, final permits will be sought from the MELCC, the MERN, and all relevant municipal authorities.

During the period spanning from 2016 to 2018, several field inventories, environmental studies, analyses, and reports have been completed to support the EIA statement. Additional studies have been carried out between 2018 and 2021 to support the authorization request for the beginning of the site preparation and construction activities. The following sub-sections summarize the Metallurgical Plant current biophysical environmental conditions.

At the end of the plant's useful life, a closure plan will be developed to minimize the impacts of the closure and maximize the success of the site rehabilitation. The lease with the Port Authority includes provisions for the Port Authority to acquire buildings and other infrastructure for future use.

The closure plan will be developed in compliance with MELCC legal requirements relating to the protection and rehabilitation of land. This will require measures concerning the environmental characterization of the site and, where applicable, the rehabilitation plan, the appropriate registration of notices in the Land Register and notices to authorities or neighbours.

1.19.6 Capital Costs

Capital costs were calculated for the metallurgical processing equipment and all required infrastructure, based on a basic engineering, and material take-offs, from bids received by vendors and contractors, and data from historical projects. The summary of capital costs for the Metallurgical Plant are shown in Table 1-19.

The initial capital cost estimate does not include taxes, replacement capital or additional capital requirements after commissioning and start-up. The reference date of for the capital cost estimate updates is October 21, 2022. The 2022 refresh has not revisited or modified the estimate as it pertains to scope, quantities, unit construction hours, construction work week, or productivity factors.

The expected accuracy range for the present capital cost estimate is -10%/+15% and meets the AACE Class 2 requirements and is established on the technological complexity of the project, appropriate reference information, and the inclusion of an appropriate contingency (P50).

For the work to be accomplished at the metallurgical plant, estimated labour costs are calculated on an assumption of an average of 40 hours per week. The first 40 hours are figured at a single rate and an allowance has been included for casual overtime equivalent to four hours per week, paid at double time across all activities.

Table 1-19: Metallurgical Plant Capital Costs Summary by Area

Area Description	Total (CAD $M)
00000 - General	29.6
01000 - VTM Concentrate and Pelletizing Plant	176.2
02000 - Direct Reduction Plant	346.0
03000 - OSBF Electrical Furnace	78.7
04000 - OSBF Furnace Off-gas Treatment	12.6
05000 - Auxiliary Plants	54.4
06000 - Hot Metal and Slag Handling	129.5
07000 - Electrical General Systems, Automation and Controls	39.5
08000 - Administration and Ancillary Facilities	8.4
Subtotal Direct Costs	**875.0**

Area Description	Total (CAD $M)
Owner's Costs	57.9
EPCM Services	29.7
Construction Tempo Facilities and Site Maintenance	19.3
Professional Services - Third Party	2.9
Commissioning Services	6.7
Common Distributables (freight, spares, initial fill, tech assistance)	14.7
Subtotal Indirect Costs	**131.4**
Contingency	91.5
Total Costs	**1 097.9**

1.19.7 Operating Costs

Operating costs for the Metallurgical Plant were calculated by area for each product produced. Processing operating costs were calculated based on Metallurgical Plant processing labour, consumables, maintenance, and electricity. General and administrative costs cover all costs not included in the process operating costs. The operating costs are summarized in Table 1-20, Table 1-21, and Table 1-22.

Table 1-20: Pig iron summary cost by process area

Summary per Process Area	%	Cost (USD/year)	Cost (USD/t)
Raw Material Receiving Area	52%	88,181,066	164.37
Iron Ore Pelletizing Area	6%	9,433,634	17.58
Energiron® DR Area	19%	32,139,466	59.91
OSBF Area and Refractories	15%	24,498,372	45.66
Secondary Metallurgy (Converter)	8%	12,909,882	24.06
Pig Casting	1%	2,357,970	4.40
Iron Plant Auxiliaries	5%	8,070,562	14.76
Ti - Slag Credits	-1%	-1,888,992	-3.52
V - Slag Credits	-8%	-12,909,882	-24.06
Other	4%	6,081,685	11.34
Total		**168,873,762**	**314.49**

Table 1-21: Operating Cost for the Titanium Slag Treatment Area

Summary Titanium Plant	Cost (USD/year)	USD/t of TiO_2 slag
Titania Slag from Iron Plant	1,888,992	15.65
Mobile Equipment	2,051,582	17.00
Crushing, milling and magsep	920,225	7.47
Total Cash TiO_2 slag Production Cost	**4,860,800**	**40.12**

Table 1-22: Operating Cost for the Vanadium Slag Treatment Area

Summary Vanadium Plant	Cost (USD/Year)	USD/t of V slag
Labour - V-slag - Crushing, Milling and bagging	1,002 491	25.15
V-slag from Iron Plant	12,909 882	323.86
Crushing, Milling, Screening & bagging	1,380 337	34.63
Total Cash V slag Production Cost	**15,292,710**	**383.63**

1.20 Financial Analysis

The economic evaluation of the BlackRock Project was performed by BRM using a discounted cash flow model on a pre-tax and post-tax basis. The capital and operating cost estimates, as presented in Chapter 21 were used as input into the model. The various inputs were produced by BBA (Mining Cash Flow Model, Mine OPEX and Mine and Met-Plant CAPEX) and BRM (OPEX model developed by BRM but audited and reviewed by BBA). The Internal Rate of Return (IRR) on total investment was calculated based on 100% equity financing. The Net Present Value (NPV) resulting from the net cash flow generated by the Project was calculated based on a discount rate varying between 0% and 10%. The Project Base Case NPV was calculated with a discount rate of 8%. The payback period is also indicated as a financial measure. Furthermore, a sensitivity analysis was also performed to assess the impact of a ±25% variation in the initial capital cost, annual operating costs and final products selling price of the Project. On the revenue side, a decrease in the relative strength of the Canadian dollar with respect to the American dollar will be beneficial to the project financials.

The Financial Analysis was performed with the following assumptions and basis:

- All prices and costs are listed in Canadian dollars unless specified otherwise;
- High Purity Pig Iron selling price was derived from the forecast provided by Wood Mackenzie. An average price of USD786/tonne was used;
- Ferrovanadium average selling price of USD38.17/kg was used;

- Titanium Slag average selling price of USD300/t was used;
- The construction period is approximately 3.5 years and the production life is 39 years;
- The project financial analysis is carried out using a constant money basis;
- Where applicable, the exchange rate assumed is $1.00 CAD = $0.76 USD;
- The mine reclamation costs have been disbursed over three years as follows:
- Year -1: 50%
- Year 0: 25%
- Year 1: 25%

The Project IRR, cumulative undiscounted cash flow, NPV at an 8% discount rate, as well as the simple payback period are presented in Table 1-23. Additional NPV values at discount rates of 0, 5, 8 and 10% are shown in Table 1-24.

Table 1-23: IRR and cumulative cash flow values (CAD$ M)

Pre-tax IRR	Pre-tax Cashflow	Pre-tax NPV @	Payback Period (Years)
21.5%	$16,369	$2,854	4.7
Post-tax IRR	**Post-tax Cashflow**	**Post-tax NPV @**	**Payback Period (Years)**
18.2%	$12,055	$1,932	5.4

Table 1-24: NPV Values at various discount rates (CAD$ M)

Discount Rate	Pre-Tax NPV	Post-Tax NPV
10%	$1,933	$1,241
8%	$2,854	$1,932
5%	$5,222	$3,708
0%	$16,369	$12,055

1.21 Other Relevant Data and Information (Schedule)

1.21.1 Beneficiation Plant

The design/construction period, as per the proposed schedule, is 32 months from the start of the detailed engineering. The schedule presented in this study takes into account the following milestones:

- Environmental Assessment and Operating Permits;
- Logging-Deforestation Permits;
- Power Permits, Construction and Activation;
- Project Detailed Engineering;
- Initial Pre-Stripping and Dyke Preparation;
- Mill Construction and Commissioning.

1.21.2 Metallurgical Plant

The estimated duration of the design/construction period is 42 months from the restart of the detailed engineering stage to the commissioning completion of the metallurgical plant. The schedule presented in this study is driven by engineering and is on a fast-track mode. It is assumed that market conditions will be favourable, and that industrial construction labour will be available and qualified. The following key project tasks and milestones were taken into account:

- Testwork and Engineering Studies (Completed);
- Environmental Assessment and Operating Permits;
- Logging-Deforestation Permits;
- Power Permits, Construction and Activation;
- Project Detailed Engineering;
- Initial Pre-Stripping and Dyke Preparation;
- Mill Construction and Commissioning;
- Metallurgical Plant Construction and Commissioning.

1.22 Risk Analysis

The risk management process identifies and assesses potential threats to all aspects of the Project including engineering, economics, environmental, permitting and more. The risks are analyzed and are subjected to a mitigation plan or are flagged to be watched as the Project advances. Each risk and resulting consequence were identified and ranked based upon risk consequence and probability levels.

Based on the guidelines set forth for evaluating risk probability and consequences, 81 potential risk elements were identified. Of these elements, 53 were determined to have a certain level of risk. For the remaining 28 elements, no risks were identified.

The top 5 risks are the following:

- Increased capital cost linked to the high inflation following the market disturbances caused by the Covid-19 pandemic and global recent events;
- Schedule delays caused by long lead equipment deliveries and shipping being disturbed by the current market and supply chain disturbance;
- Ramp-up slower than planned due to inexperienced operators and incompleteness of operational readiness plan;
- Shortage of qualified construction management resources in the industry, increased CAPEX risk resulting from the selected execution strategy which is based on client project management team assembled from contractual hires;
- Ratio of RWi/BWi is in lower Grey Zone for creation of Pebbles in SAG Mill.

1.23 Conclusions and Discussion

The results of the current Feasibility Study indicate that it is technically feasible to produce a magnetite concentrate from the Southwest Pit. Additionally, for the Metallurgical Plant, results indicate that it is also technically feasible to use VTM concentrate from the BRM deposit in Chibougamau to produce high purity pig iron, as well as titanium and vanadium products.

It is BBA's opinion that the BlackRock Project is technically feasible and could proceed to the next stage of project development, consisting of the remaining of the EPCM started in 2019, if the global financial analysis performed by BlackRock Metals concludes to viable economical results.

1.24 Recommendations and Future Work Program

The Feasibility Study is comprised of a comprehensive testwork program and engineering that was completed to produce this Report. In order to further reinforce the current findings, as well as address potential issues and risks, a series of recommendations could be implemented and are discussed in this Chapter. Table x presents the recommended work program. This program aims at providing information to support the detailed engineering and/or start the pre-production phase; the decision to advance to detailed engineering is not contingent on the results of the work program.

Table 1-25: Recommended Work Program

Program	Cost (CAD)
Hydrogeology (see section 26.1)	300,000$
Geotechnical for waste pile (see Section 26.1)	150,000$

This work program is included in the working capital of the project cost structures and will be executed upon notice to proceed.

1.24.1 Mining

Further to the completion of the Feasibility Study, the following activities should occur concurrently with the advancement of the Detailed Engineering phase of the Project:

- Conduct an infill exploration drilling program to tighten the spacing between drillholes where mining will occur during the first five years of operation;
- Commission a hydrogeology study to better define the pit dewatering requirements and mitigate any potential impact from ground water on pit wall stability;
- Based on the previous two points, develop a detailed short-term and medium-term mine plan to be used for early pit development and production for the first two years of operation, as well as a more detailed long-term mine plan using detailed phase designs;
- Review mining phases with the potential for shorter phases;
- Obtain firm pricing on mining equipment;
- Obtain firm pricing from mining contractors;
- Based on the results of the recent density measurement program, further assess the density of the Mineralized Material and host rock;
- Conduct geotechnical studies to confirm waste rock pile slopes and benching arrangement.

1.24.2 Metallurgical Testing and Processing (Beneficiation Plant)

Several metallurgical recommendations are proposed that could be undertaken to reduce capital and operating costs, reduce power and reagent consumption, as well as potentially improve the economics and accuracy of the project. These recommendations are summarized as follows:

- Hydraulic testing could be performed to validate the tailings pumping characteristics;
- Creation of an operational readiness plan during detailed engineering;
- Purity of pig iron for sale to be confirmed in order to recommend the appropriate process technology.

1.24.3 Environmental and Tailings

A more detailed investigation into tailings deposition and water management over the life of mine could help ensure the minimum level of available water is maintained.

1.24.4 Engineering and Infrastructure

- Update the detailed construction plan before starting construction to ensure the proper civil grades and quantities will be available;
- Complete the detailed engineering phase.

1.24.5 Construction, Execution and Capital Costs

- Prepare complete update of the control budget inclusive of updated major purchase orders and contracts pricing;
- Update procurement plan and re-award previously committed purchase orders;
- Update detailed project and construction execution plan during the next phase of the project and start negotiations with high PCM experience individuals targeted to join the Owner's project team;
- Update survey of lodging availability in the area;
- Renew necessary permits in order to restart site preparation;
- Validate the construction resource to confirm the construction execution plan.

2. Introduction

On December 13, 2022, BlackRock Metals Inc. (herein referred to as "BlackRock" or "BRM") entered into an arm's length share exchange agreement with Strategic Resources Inc. ("Strategic"), pursuant to which Strategic will acquire all of the outstanding shares in BlackRock for shares of Strategic (the "Transaction"). Upon completion of the Transaction, BlackRock will become a wholly owned subsidiary of Strategic. The BlackRock Vanadium, Titanium and Magnetite (VTM) deposit is located approximately 700 km north of Montreal and 25 km southeast (60 km by road) of Chibougamau, Québec, Canada. The Southwest (SW) deposit has been defined for development. BlackRock owns 100% of the deposit.

2.1 Scope of Study

The following Technical Report (the Report) presents the results of the Feasibility Study (-10/+15%) for the development of the Southwest deposit. BlackRock mandated BBA to perform a feasibility study based on contributions from several independent consulting firms. In 2015 a pre-feasibility study was performed to evaluate the production of roughly 950,000 tpy of a 62% Fe grade magnetite concentrate and 200,000 tpy of a 46% TiO_2 ilmenite concentrate. An updated pre-feasibility study was evaluated in 2016 to optimize the mine plan and to reduce the magnetite concentrate production to approximately 830,000 tpy. In the current feasibility study, the mine plan targets the same concentrate production with a maximum of 5% excess concentrate production being tolerated.

The present study, in 2022, evaluates the production of approximately 856,000 tpy of a magnetite concentrate; however, the production of an ilmenite concentrate is not being considered. In addition, the current study evaluates the production from the magnetite concentrate of high purity nodular pig iron, vanadium rich slag, and titanium slag in the Metallurgical (secondary transformation) Plant.

The study includes the review and validation of all pertinent existing data, including ongoing studies such as laboratory metallurgical testing, estimation of the mineral reserves, mine planning, infrastructure and service facilities, environmental impact evaluation, capital investment and operating costs. The battery limits for the scope of the project are from the mine to the Port of Saguenay where the Metallurgical Plant products are shipped. Prices for the final transfer/selling prices and rail and port operating costs provided by BlackRock Metals were used for mining optimizations; however, no financial analysis is being performed by BBA. It is understood that BlackRock will perform their own financial analysis based upon the capital and operating costs provided by BBA.

This Report was prepared at the request of Daniel Dutton, Vice-President, Technical Services & Metallurgical Products and Processes, BlackRock Metals. BlackRock's head office is situated at:

120 Adelaide Street West, Suite 2500
Toronto, Ontario, Canada
M5H 1T1
and,

Strategic Resources Inc. Canadian head office is situated at:
410 - 625 Howe Street
Vancouver, BC
V6C 2T6
Canada

2.2 Effective Dates and Declaration

This report is considered effective as of November 18, 2022. BBA's opinion contained herein is based on the information collected by BBA throughout the course of BBA's investigations, which in turn reflect various technical and economic conditions at the time of writing.

BBA is not an insider, associate or an affiliate of BlackRock and neither BBA nor any affiliate has acted as Advisor to BlackRock, its subsidiaries or its affiliates in connection with this Project. The results of the technical review by BBA are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' 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 is intended for use by BlackRock subject to the terms and conditions of its respective contracts with the Report authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this report by any third party is at the sole risk of that party.

All the Qualified Person are independent of BlackRock Metals Inc. and Strategic Resources Inc as defined in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101")

Table 2-1: Breakdown of report Chapters and their Qualified Party

Chapter	Title of Chapter	Qualified Person	Company
1	Executive Summary	All firms contributed based on their responsible chapters and scope of work.	Various
2	Introduction	André Allaire	BBA Inc.
3	Reliance on Other Experts	André Allaire	BBA Inc.
4	Property Description and Location	Claude Bisaillon	Independent
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Claude Bisaillon	Independent
6	History	Claude Bisaillon	Independent
7	Geological Setting and Mineralization	Claude Bisaillon	Independent
8	Deposit Types	Claude Bisaillon	Independent
9	Exploration	Claude Bisaillon	Independent
10	Drilling	Claude Bisaillon	Independent
11	Sample Preparation and Analyses	Claude Bisaillon	Independent
12	Data Verification	Claude Bisaillon	Independent
13	Mineral Processing and Metallurgical Testing	André Allaire	BBA Inc.
14	Mineral Resources	Claude Bisaillon	Independent
15	Mineral Reserve Estimate	Isabelle Leblanc	BBA Inc.
16	Open Pit Mining Methods	Isabelle Leblanc	BBA Inc.
17	Recovery Methods	André Allaire	BBA Inc.
18 Except 18.12	Project Infrastructure	André Allaire	BBA Inc.
18.12	Tailings Management	Nicolas Skiadas	Journeaux Associates
19	Market Studies and Contracts	André Allaire	BBA Inc.
20 Except 20.1.7.2, 20.1.7.6, 20.1.8.1.2 and 20.1.9	Environmental Studies, Permitting and Social or Community Impact	Nathalie Fortin	WSP
20.1.7.2, 20.1.7.6,	Site Water Management, Tailings Management and	Nicolas Skiadas	Journaux Associates

Chapter	Title of Chapter	Qualified Person	Company
20.1.8.1.2 and 20.1.9	Tailings Pond Construction Plan		
21	Capital and Operating Costs	André Allaire	BBA Inc.
22	Economic Analyses	André Allaire	BBA Inc.
23	Adjacent Properties	Claude Bisaillon	Independent
24	Other Relevant Data and Information	André Allaire	BBA Inc.
25	Interpretation and Conclusions	All firms contributed based on their responsible chapters and scope of work.	Various
26	Recommendations	All firms contributed based on their responsible chapters and scope of work.	Various
27	References	All firms contributed based on their responsible chapters and scope of work.	Various

2.3 Sources of Information

This Report is based on testwork results, internal company technical reports, maps, published government reports and public information, in addition to items listed in Chapter 27 "References" of this Report. This Report also includes technical information that requires calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and, consequently, introduce a margin of error. Where this occurs, BBA does not consider it to be material.

The overall Study was collated and integrated by BBA personnel. Sections from reports authored by other consultants may have been directly quoted or summarized in this Report, and are so indicated, where appropriate.

This Feasibility Study has been completed using previous Technical Reports as well as available information contained in, but not limited to, the following reports, documents and discussions:

- COREM batch and pilot plant testwork for development of a magnetic separation flowsheet and ilmenite flotation circuit;
- SGS and COREM tests involving Satmagan measurements and Davis Tube tests for development of correlation equations used to estimate sample head grades and to predict weight and elemental recoveries after magnetic separation. Validation of pilot plant testwork using a homogenized bulk sample including flotation testwork for validation of planned magnetite concentrate desulphurization;
- COREM pilot plant for the ilmenite recovery process;
- Equipment-related testwork;

- Environment testwork reports evaluating the potential environmental impact of BlackRock tailings, effluents and waste rock materials;
- Journeaux Associates conceptualization of the tailings dam construction/management scenario;
- Mine Rehabilitation conception and costs from WSP report;
- Process flowsheets of similar existing operations;
- Budgetary quotes and/or firm bids from suppliers for all major equipment;
- Block Model given by SGS Canada;
- Rail car requirements, costing and leasing parameters have been provided by BlackRock;
- Train Concentrate Loadout design construction and operation were outsourced;
- Geotechnical Pit Slopes provided by Englobe Corp;
- Previous Technical Reports done for the BlackRock Project;
- COREM concentrate characterization to determine the physical properties, chemical composition and particle size distribution of 5 t of VTM concentrate. Pelletizing testwork to determine the VTM concentrate suitability for pelletizing;
- Tenova HYL direct reduction bag tests to determine the VTM concentrate reducibility under commercial conditions;
- Tenova Pyromet smelting and refining tests to investigate the smelting behaviour, particularly when the pig iron produced from smelting is subjected to the oxidation (converting) step to recover the vanadium to a slag phase;
- Mintek via Tenova Pyromet titanium slag digestion tests to compare the titanium slag produced with existing slag on the market.

BBA believes that the basic assumptions contained in the information above are factual and accurate, and that the interpretations are reasonable. BBA has relied on this data and has no reason to believe that any material facts have been withheld. BBA also has no reason to doubt the reliability of the information used to evaluate the mineral resources presented herein.

2.4 Terms of Reference

Unless otherwise stated:

- All units of measurement in the Report are in the metric system;
- All currency amounts in this Report are stated in Canadian dollars (CA$);
- A foreign exchange rate of US$1.00 = CA$ 1.315 was used;
- A foreign exchange rate of EUR€ 1.00 = CA$ 1.43 was used;

- A foreign exchange rate of AU$1.00 = CA$ 0.97 was used;
- All cost estimates have a base date of Q4 2022.

2.5 Site Visit

2.5.1 BBA

Visits to the BlackRock site were made by André Allaire, Eng., M.Eng., PhD, and Patrice Live, P.Eng., representing BBA Inc., on June 30, 2010, and September 29, 2010, respectively. The site inspection concentrated on the following: review of potential infrastructure location, presence of water courses, road accesses, etc.

Over the last decade, BBA has been supporting the Southwest project regularly, completing mandate for the mine and beneficiation plant.

Since BBA visited the mine site, there has been no significant work undertaken on site and consequently, no material changes have occurred.

2.5.2 SGS

SGS Canada representatives visited the site to perform independent sampling for data verification; visits were made by Vincent Cloutier, between April 26 to 29, 2010. Claude Bisaillon, between February 23 and 25, 2011 and Karina Sarabia, under Mr. Bisaillon's supervision, between May 28 to 30, 2013. Since this project was done over a long period of time, numerous site visits were required. The site visits took place at various times in the exploration stages in 2010, 2011 and 2013. Site visits consisted of visiting the drill during a drilling campaign, visiting the logging and splitting facilities, the sampling and analytical facilities and meeting with various BlackRock personnel involved in the Project. The site visits also consisted of verification of the logging, core cutting, sampling procedures. Representative independent samples were also selected and sent to an independent laboratory to confirm the various grades of mineralization.

Since SGS last visited the mine site in 2013, there has been no significant work undertaken on site and consequently, no material changes have occurred.

2.5.3 Journeaux

Nikolas Skiadas (P.Eng.) from Journeaux Associates, along with a multidisciplinary team comprised of civil engineers and procurement personnel, travelled to Chibougamau and to the site on May 19 and 20, 2011, to visit local suppliers, examine terrain, etc.

2.5.4 WSP

Nathalie Fortin visited the metallurgical site location on July 16, 2019.

3. Reliance on Other Experts

BBA prepared this Feasibility Study using the reports and documents noted in Chapter 27 "References". BBA did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, but has relied upon information provided by the Client for any agreement(s) discussed in Chapter 19 and for the land tenure information provided in Chapter 4.

Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Report. BBA is not aware of any known litigation potentially affecting the BlackRock Deposit. BBA's responsibility is to assure that this Technical Report meets the stipulated guidelines and standards, given that certain sections of this Report were contributed by SGS Canada, Journeaux Associates, Lamont or other BlackRock consultants.

3.1. Study Contributors

For testwork purposes concerning the Beneficiation Plant, drill core samples for metallurgical testing were collected and prepared by BRM and submitted to SGS Mineral Services (Lakefield, Ontario, Canada) and COREM (Québec City, Québec, Canada), which are both accredited laboratories. BBA has reviewed all testwork results generated by SGS and COREM, and believes they are generally accurate. The extent of the testwork pertains to the metallurgical response of the BlackRock material from the Southwest Pit. The information was taken from references 3, 7, 8 and 10-13 in Chapter 27 and was used in the preparation of the Beneficiation Plant section of Chapter 13 of this report. BBA is relying on COREM and SGS as independent experts for all of the information contained in the above-mentioned studies.

A geotechnical study for the Southwest Pit was carried out by LVM (now Englobe Corp.). This study, entitled "Pit Slope Design Report for Southwest Pit" reported on July 22, 2013, was referenced during the preparation of Chapters 15 and 16 of this report. BBA is relying on Englobe Corp. as an independent expert for all of the information contained in the above-mentioned study.

For testwork purposes concerning the Metallurgical Plant, a 5-tonne sample of VTM concentrate for metallurgical testing was collected and prepared by BRM and submitted to Tenova HYL (Monterrey, Mexico), COREM (Québec City, Québec, Canada), and Mintek via Tenova Pyromet (South Africa), which are recognized accredited laboratories. BBA has reviewed all testwork results generated by Tenova, COREM, and Mintek, and believes they are generally accurate. The extent of the testwork pertains to the metallurgical response of the BlackRock VTM concentrate produced from the Beneficiation Plant. The information was taken from references 26-29 in

Chapter 27 and was used in the preparation of the metallurgical plant section of Chapter 13 in this report. BBA is relying on COREM, Tenova, and Mintek as independent experts for all of the information contained in the above-mentioned studies.

3.2. Other Contributors

The initial evaluation of the Mineral Resources in 2010 to 2014 was supervised by Michel Dagbert, an employee of SGS at the time and a QP. Mr. Dagbert has since retired and is not an active member of the OIQ and as such does not fulfil the requirements to be an independent qualified person for the purpose of NI 43-101. Mr. Bisaillon as co-author of the initial evaluations of the Mineral Resources has since taken responsibility for all Mineral Resources chapters of the Technical Report.

4. Property Description and Location

4.1. Location and Access

The site claims have been updated as of November 9, 2022 by Daniel Dutton of BlackRock that will be a wholly owned subsidiary of Strategic. As of this date, there are 230 BlackRock claims picked up from the GESTIM website covering an area of 8,687 hectares. These claims can be seen in Figure 4-2, Figure 4-3 and Figure 4-4 below. Detailed information on the claims is found in Table 4-1. The Property extends over 22 km along a northeast-southwest axis.

The BlackRock Property is located southeast of the Chibougamau Municipality, Québec, Canada in the unsurveyed Lemoine, Rinfret, Dollier and Queylus townships, on NTS map sheets 32G/09, 32G/16 and 32H/13 (Figure 4-1). The infrastructures are located 100% on the traditional territory of the Cree Nation, covered by the James Bay Agreement. The infrastructures are also all in the limits of the Chibougamau municipality.

From Chibougamau, the future mine site is within a distance of 26 km as the crow flies and at 60 km by road. It is accessible via the paved Highway 167, between Chibougamau and St-Félicien and then by forestry roads. A network of practicable gravel roads provides access to most of the Property and, more specifically, to both projected pits and future mining site.

A Hydro-Québec 161 kV power line and a Canadian National Railway line (CFILNQ) are located along Highway 167.

BRM has chosen to construct its metallurgical production facility on an already qualified heavy industrial land belonging to the Saguenay Port Authority near the maritime terminal of Grande-Anse in the City of Saguenay, La Baie district. The land will remain the property of the Saguenay Port Authority and will be leased to BlackRock Metals via a long-term lease. The federal industrial-port zone offers several advantages to the construction and operation of the metallurgical production facility:

- Deep-water port accessible year-round allowing shipped transport to deliver and export materials and products at low cost. At the same time rail and road links offer alternative links to markets;
- Qualified labor in the steel industry sector;
- Site provided by the Port Authority of Saguenay already cleared and levelled;
- Situated in a designated used industrial zone;
- Located away from populated areas in a non-sensitive site chosen to mitigate environmental disturbance.

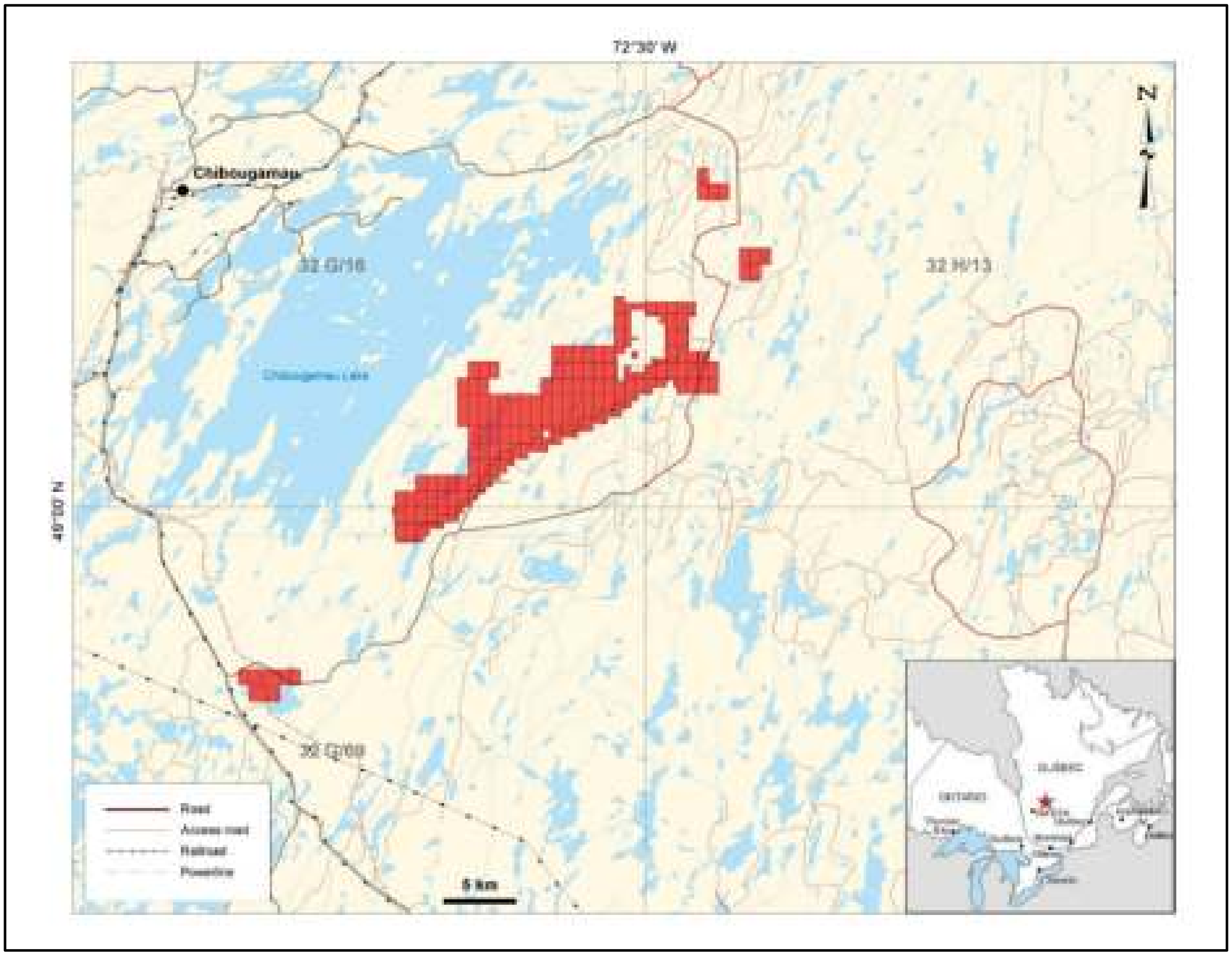

Figure 4-1: Property location as of September 30, 2021

4.2. Property Ownership, Agreement and Environmental Obligation

On December 13, 2022, BlackRock Metals Inc. (herein referred to as "BlackRock" or "BRM") entered into an arm's length share exchange agreement with Strategic Resources Inc. ("Strategic"), pursuant to which Strategic will acquire all of the outstanding shares in BlackRock for shares of Strategic (the "Transaction"). Upon completion of the Transaction, BlackRock will become a wholly-owned subsidiary of Strategic.

All claims of the BlackRock Property belong 100% to BlackRock and are current and duly registered under the name of Métaux BlackRock Inc. at "*Services des titres miniers*" of the MRNQ, and under number 88633 in the Gestim system. With the exception to Strategic which Blackrock will be a subsidiary of, there is no other company or third party that owns an interest in the project.

The Property is not subject to any royalties, back-in rights, payments, or other agreements.

This area of the project is included in the James Bay and Northern Québec Agreement ("Convention de la Baie-James et du Nord Québécois"), as well as the subsequent "Paix des Braves" treaty between the Québec Government and the Cree Nation. It is indicated as "Terres de catégorie III", and shall be therefore free of any encumbrances relating to exploration activities.

According to the "*Paix des Braves*" agreement, any intervention affecting traditional trapping, such as logging, needs approval from the trapline tallymen.

Some of the claims, outside the project area, on the southeast corner of the Property, are located in non-organized territories (*Territoires non–organisés*) managed by the Domaine-du-Roy regional municipality (*MRC Domaine-du-Roy*) in St-Félicien. This area is not included in the James Bay and Northern Québec agreements, and therefore falls under the general regulations of the "*Ministère des Ressources naturelles du Québec*" (thereafter MRNQ) for matters related to forestry and mining, and under the general regulations of the "*Ministère du Développement durable, de l'Environnement, de la Faune et des Parcs du Québec*" (thereafter MDDEFP) for environmental issues. The area lies within the Nitassinan lands, the ancestral and claimed homeland of the Innu of Mashteuiatsh, as defined in the EPOG (Entente de principe d'ordre général), and the Québec and Canadian governments.

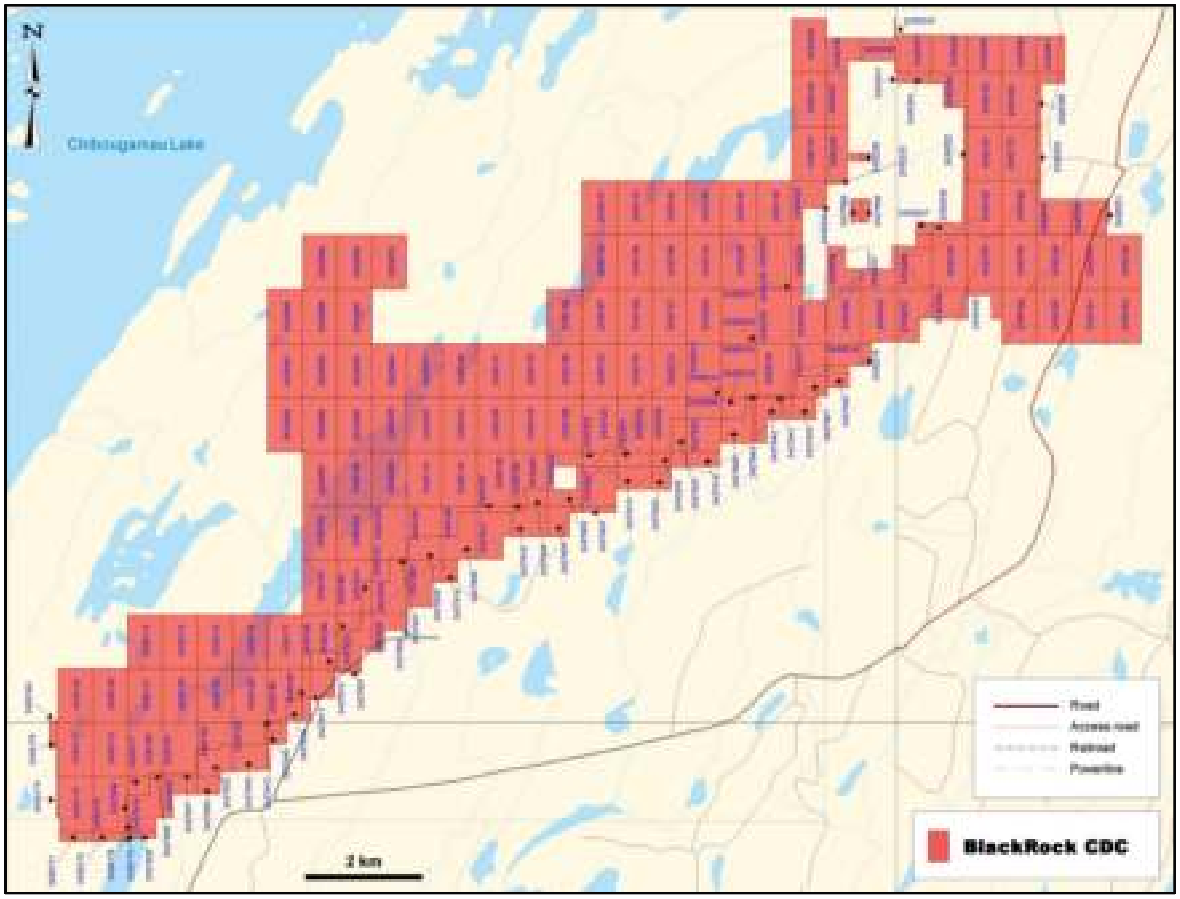

Figure 4-2: Maps of BlackRock mineral exploration and development claims
(as of September 30, 2021)

The upper map shows main claim block that contains the defined Mineral Resources that are the subject of this Feasibility Report. Note that this 2021 claim outline has a new group of 14 claims on the NW edge extending toward Chibougamau Lake that was acquired in 2018 through staking. Subsequent maps in this report use the 2017 claim outline to show relative location of the claims.

Figure 4-3: Map of BlackRock infrastructure claims
(as of September 30, 2021)

These claims represent a potential area of infrastructure development in proximity to the main rail line as well as Highway 167.

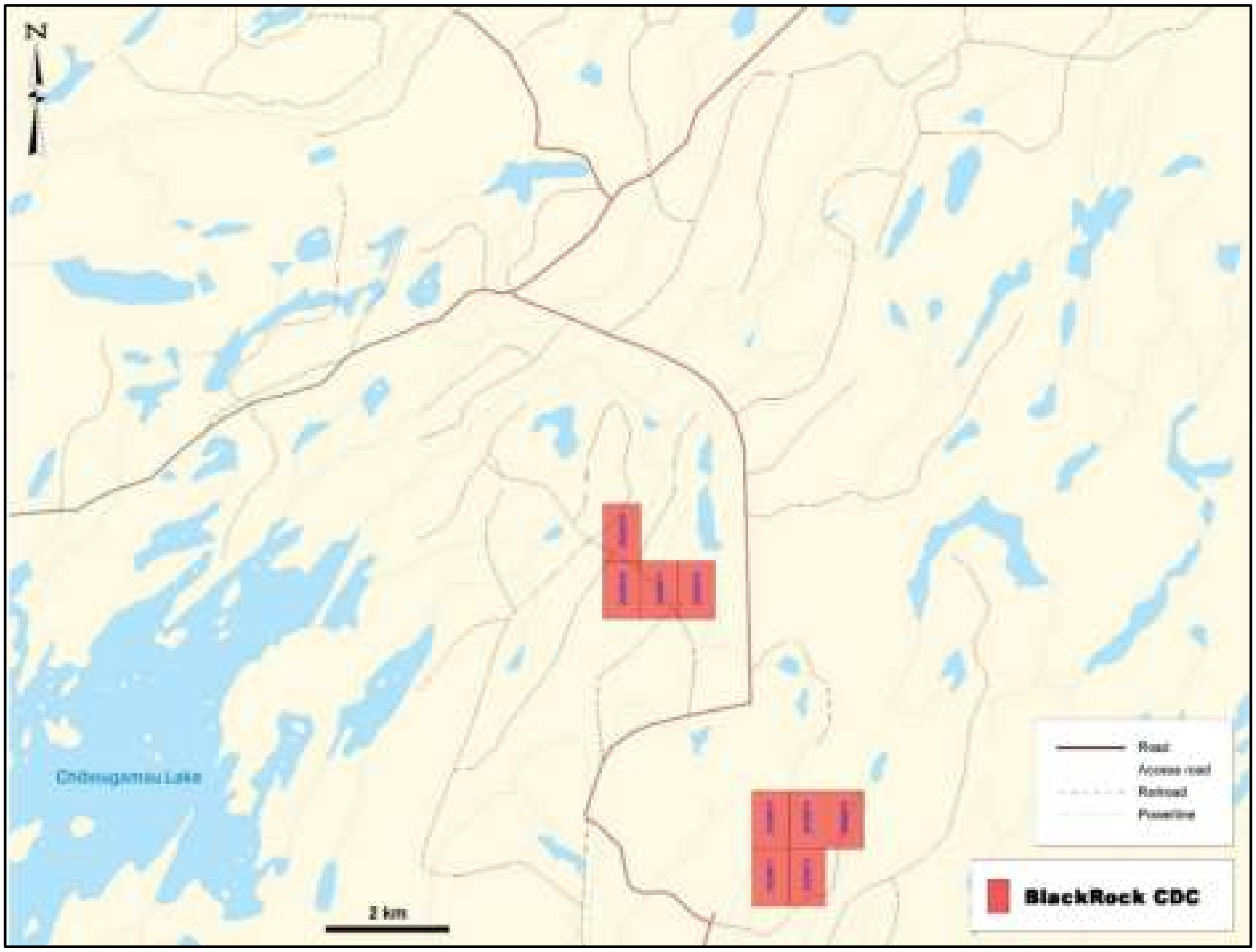

Figure 4-4: Map of two groups of outlying claims staked along trend of magnetic anomalies to the NE of the main block that hold potential for additional mineralization to be delineated

These blocks are not part of the resource modelling, and will not be discussed in any detail in the following report.

Table 4-1: List of BlackRock claims

Type	Index	Status	SNRC Leaf	Range/Block	Lot/Block	Registration Date	Expiry Date	Renewal	Area (has)	Required Fees	Work Required	Exceeding Credits	Owner	Percentage
CDC	2233502	Active	32G09	19	23	May 12, 2010	May 11, 2025	6	55.73	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2233503	Active	32G09	19	24	May 12, 2010	May 11, 2025	6	55.73	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2233504	Active	32G09	19	25	May 12, 2010	May 11, 2025	6	55.73	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2233505	Active	32G09	20	22	May 12, 2010	May 11, 2025	6	55.72	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2233506	Active	32G09	20	23	May 12, 2010	May 11, 2025	6	55.72	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2233507	Active	32G09	20	24	May 12, 2010	May 11, 2025	6	55.72	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2233508	Active	32G09	20	25	May 12, 2010	May 11, 2025	6	55.72	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2233509	Active	32G09	20	26	May 12, 2010	May 11, 2025	6	55.72	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2233510	Active	32G09	20	27	May 12, 2010	May 11, 2025	6	55.72	$68.75	$1,800.00	$0.00	Minière BlackRock	100
CDC	2427688	Active	32G16	10	59	June 30, 2015	March 30, 2026	6	7.59	$35.25	$750.00	$0.00	Minière BlackRock	100
CDC	2427689	Active	32G16	10	60	June 30, 2015	March 30, 2026	6	5.49	$35.25	$750.00	$0.00	Minière BlackRock	100
CDC	2427900	Active	32G16	2	47	July 3, 2015	April 17, 2026	16	0.46	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427901	Active	32G16	7	58	July 3, 2015	April 17, 2026	16	18.85	$35.25	$1,000.00	$3 739.97	Minière BlackRock	100
CDC	2427902	Active	32G16	5	51	July 3, 2015	April 17, 2026	16	20.76	$35.25	$1,000.00	$4 387.45	Minière BlackRock	100
CDC	2427903	Active	32G09	29	41	July 3, 2015	April 17, 2026	16	11.82	$35.25	$1,000.00	$1 356.87	Minière BlackRock	100
CDC	2427904	Active	32G16	3	47	July 3, 2015	April 17, 2026	16	41.20	$68.75	$2,500.00	$7 341.42	Minière BlackRock	100
CDC	2427905	Active	32G16	6	56	July 3, 2015	April 17, 2026	16	31.89	$68.75	$2,500.00	$4 185.41	Minière BlackRock	100
CDC	2427906	Active	32G16	4	48	July 3, 2015	April 17, 2026	16	37.49	$68.75	$2,500.00	$6 083.76	Minière BlackRock	100
CDC	2427907	Active	32G16	1	45	July 3, 2015	April 17, 2026	16	3.30	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427908	Active	32G16	5	50	July 3, 2015	April 17, 2026	16	10.85	$35.25	$1,000.00	$1 028.05	Minière BlackRock	100
CDC	2427909	Active	32G09	29	40	July 3, 2015	April 17, 2026	16	24.80	$35.25	$1,000.00	$5 756.97	Minière BlackRock	100
CDC	2427910	Active	32G16	2	44	July 3, 2015	April 17, 2026	16	13.91	$35.25	$1,000.00	$2 065.36	Minière BlackRock	100
CDC	2427911	Active	32G16	1	44	July 3, 2015	April 17, 2026	16	27.27	$68.75	$2,500.00	$2 619.28	Minière BlackRock	100
CDC	2427912	Active	32G09	30	43	July 3, 2015	April 17, 2026	16	14.13	$35.25	$1,000.00	$2 139.93	Minière BlackRock	100
CDC	2427913	Active	32G16	5	53	July 3, 2015	April 17, 2026	16	23.64	$35.25	$1,000.00	$5 363.74	Minière BlackRock	100
CDC	2427914	Active	32G16	3	45	July 3, 2015	April 17, 2026	16	9.62	$35.25	$1,000.00	$611.08	Minière BlackRock	100
CDC	2427915	Active	32G16	4	50	July 3, 2015	April 17, 2026	16	29.06	$68.75	$2,500.00	$3 226.07	Minière BlackRock	100
CDC	2427916	Active	32G09	29	39	July 3, 2015	April 17, 2026	16	40.08	$68.75	$2,500.00	$6 924.46	Minière BlackRock	100
CDC	2427917	Active	32G16	4	49	July 3, 2015	April 17, 2026	16	42.72	$68.75	$2,500.00	$7 856.68	Minière BlackRock	100
CDC	2427918	Active	32G16	3	48	July 3, 2015	April 17, 2026	16	10.05	$35.25	$1,000.00	$756.86	Minière BlackRock	100
CDC	2427919	Active	32G16	5	55	July 3, 2015	April 17, 2026	16	14.36	$35.25	$1,000.00	$2 217.90	Minière BlackRock	100
CDC	2427920	Active	32G16	5	54	July 3, 2015	April 17, 2026	16	19.80	$35.25	$1,000.00	$4 062.01	Minière BlackRock	100
CDC	2427921	Active	32G16	4	46	July 3, 2015	April 17, 2026	16	0.78	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427922	Active	32G16	6	58	July 3, 2015	April 17, 2026	16	15.68	$35.25	$1,000.00	$2 665.38	Minière BlackRock	100
CDC	2427923	Active	32G16	4	52	July 3, 2015	April 17, 2026	16	6.53	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427924	Active	32G16	3	46	July 3, 2015	April 17, 2026	16	44.24	$68.75	$2,500.00	$8 371.95	Minière BlackRock	100

Type	Index	Status	SNRC Leaf	Range/Block	Lot/Block	Registration Date	Expiry Date	Renewal	Area (has)	Required Fees	Work Required	Exceeding Credits	Owner	Percentage
CDC	2427925	Active	32G16	6	54	July 3, 2015	April 17, 2026	16	14.76	$35.25	$1,000.00	$2 353.51	Minière BlackRock	100
CDC	2427926	Active	32G16	2	46	July 3, 2015	April 17, 2026	16	23.63	$35.25	$1,000.00	$5 360.36	Minière BlackRock	100
CDC	2427927	Active	32G16	5	52	July 3, 2015	April 17, 2026	16	39.97	$68.75	$2,500.00	$6 924.46	Minière BlackRock	100
CDC	2427928	Active	32G09	28	39	July 3, 2015	April 17, 2026	16	5.14	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427929	Active	32G16	1	43	July 3, 2015	April 17, 2026	16	11.07	$35.25	$1,000.00	$1 102.63	Minière BlackRock	100
CDC	2427930	Active	32G16	7	59	July 3, 2015	April 17, 2026	16	14.75	$35.25	$1,000.00	$2 350.12	Minière BlackRock	100
CDC	2427931	Active	32G16	2	45	July 3, 2015	April 17, 2026	16	36.28	$68.75	$2,500.00	$5 673.59	Minière BlackRock	100
CDC	2427932	Active	32G09	30	41	July 3, 2015	April 17, 2026	16	28.83	$68.75	$2,500.00	$3 148.11	Minière BlackRock	100
CDC	2427933	Active	32G16	6	55	July 3, 2015	April 17, 2026	16	33.76	$68.75	$2,500.00	$4 819.33	Minière BlackRock	100
CDC	2427934	Active	32G16	4	51	July 3, 2015	April 17, 2026	16	22.80	$35.25	$1,000.00	$5 079.00	Minière BlackRock	100
CDC	2427935	Active	32G09	30	42	July 3, 2015	April 17, 2026	16	28.69	$68.75	$2,500.00	$3 100.65	Minière BlackRock	100
CDC	2427936	Active	32G09	30	39	July 3, 2015	April 17, 2026	16	1.37	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427937	Active	32G09	30	40	July 3, 2015	April 17, 2026	16	4.38	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427938	Active	32G16	4	47	July 3, 2015	April 17, 2026	16	22.63	$35.25	$1,000.00	$5 021.36	Minière BlackRock	100
CDC	2427939	Active	32G09	29	38	July 3, 2015	April 17, 2026	16	0.68	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427940	Active	32G16	1	42	July 3, 2015	April 17, 2026	16	0.12	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427941	Active	32G16	7	57	July 3, 2015	April 17, 2026	16	2.31	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2427942	Active	32G16	6	57	July 3, 2015	April 17, 2026	16	29.48	$68.75	$2,500.00	$3 368.45	Minière BlackRock	100
CDC	2427943	Active	32G16	7	56	July 3, 2015	April 17, 2026	16	0.65	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430111	Active	32G09	30	37	July 17, 2015	March 2, 2025	8	55.63	$68.75	$2,500.00	$64 070.85	Minière BlackRock	100
CDC	2430112	Active	32G09	30	38	July 17, 2015	March 2, 2025	8	55.63	$68.75	$2,500.00	$64 070.85	Minière BlackRock	100
CDC	2430113	Active	32G16	2	39	July 17, 2015	March 2, 2025	8	55.61	$68.75	$2,500.00	$64 045.60	Minière BlackRock	100
CDC	2430114	Active	32G16	2	40	July 17, 2015	March 2, 2025	8	55.61	$68.75	$2,500.00	$62 845.60	Minière BlackRock	100
CDC	2430115	Active	32G16	2	41	July 17, 2015	March 2, 2025	8	55.61	$68.75	$2,500.00	$62 845.60	Minière BlackRock	100
CDC	2430116	Active	32G16	2	42	July 17, 2015	March 2, 2025	8	55.61	$68.75	$2,500.00	$62 845.60	Minière BlackRock	100
CDC	2430117	Active	32G16	2	43	July 17, 2015	March 2, 2025	8	55.61	$68.75	$2,500.00	$62 845.60	Minière BlackRock	100
CDC	2430118	Active	32G16	5	47	July 17, 2015	March 2, 2025	8	55.58	$68.75	$2,500.00	$55 607.71	Minière BlackRock	100
CDC	2430119	Active	32G16	5	48	July 17, 2015	March 2, 2025	8	55.58	$68.75	$2,500.00	$63 007.71	Minière BlackRock	100
CDC	2430120	Active	32G16	5	49	July 17, 2015	March 2, 2025	8	55.58	$68.75	$2,500.00	$63 580.83	Minière BlackRock	100
CDC	2430121	Active	32G16	6	48	July 17, 2015	March 2, 2025	8	55.57	$68.75	$2,500.00	$60 395.08	Minière BlackRock	100
CDC	2430122	Active	32G16	6	50	July 17, 2015	March 2, 2025	8	55.57	$68.75	$2,500.00	$63 995.08	Minière BlackRock	100
CDC	2430123	Active	32G16	6	52	July 17, 2015	March 2, 2025	8	55.57	$68.75	$2,500.00	$62 995.08	Minière BlackRock	100
CDC	2430124	Active	32G16	6	53	July 17, 2015	March 2, 2025	8	55.57	$68.75	$2,500.00	$63 995.08	Minière BlackRock	100
CDC	2430125	Active	32G16	1	37	July 17, 2015	March 2, 2025	8	55.62	$68.75	$2,500.00	$64 058.21	Minière BlackRock	100
CDC	2430126	Active	32G16	1	38	July 17, 2015	March 2, 2025	8	55.62	$68.75	$2,500.00	$64 058.21	Minière BlackRock	100
CDC	2430127	Active	32G16	1	39	July 17, 2015	March 2, 2025	8	55.62	$68.75	$2,500.00	$64 058.21	Minière BlackRock	100
CDC	2430128	Active	32G16	1	40	July 17, 2015	March 2, 2025	8	55.62	$68.75	$2,500.00	$63 564.13	Minière BlackRock	100
CDC	2430129	Active	32G16	1	41	July 17, 2015	March 2, 2025	8	55.62	$68.75	$2,500.00	$63 672.63	Minière BlackRock	100

Type	Index	Status	SNRC Leaf	Range/Block	Lot/Block	Registration Date	Expiry Date	Renewal	Area (has)	Required Fees	Work Required	Exceeding Credits	Owner	Percentage
CDC	2430130	Active	32G16	7	48	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 982.45	Minière BlackRock	100
CDC	2430131	Active	32G16	7	49	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 982.45	Minière BlackRock	100
CDC	2430132	Active	32G16	7	50	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 982.45	Minière BlackRock	100
CDC	2430133	Active	32G16	7	52	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 115.52	Minière BlackRock	100
CDC	2430134	Active	32G16	7	53	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 507.58	Minière BlackRock	100
CDC	2430135	Active	32G16	8	51	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 202.03	Minière BlackRock	100
CDC	2430136	Active	32G16	8	53	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430137	Active	32G16	8	54	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430138	Active	32G16	9	52	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430139	Active	32G16	9	53	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430140	Active	32G16	9	55	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430141	Active	32G16	10	52	July 17, 2015	March 2, 2025	8	55.54	$68.75	$2,500.00	$63 957.20	Minière BlackRock	100
CDC	2430142	Active	32G16	10	53	July 17, 2015	March 2, 2025	8	55.54	$68.75	$2,500.00	$63 957.20	Minière BlackRock	100
CDC	2430143	Active	32G16	10	54	July 17, 2015	March 2, 2025	8	55.54	$68.75	$2,500.00	$63 267.01	Minière BlackRock	100
CDC	2430144	Active	32G16	10	55	July 17, 2015	March 2, 2025	8	55.54	$68.75	$2,500.00	$63 957.20	Minière BlackRock	100
CDC	2430145	Active	32G16	10	56	July 17, 2015	March 2, 2025	8	55.54	$68.75	$2,500.00	$63 957.20	Minière BlackRock	100
CDC	2430146	Active	32G16	10	57	July 17, 2015	March 2, 2025	8	55.54	$68.75	$2,500.00	$61 457.20	Minière BlackRock	100
CDC	2430147	Active	32G16	11	58	July 17, 2015	March 2, 2025	8	55.53	$68.75	$2,500.00	$63 619.57	Minière BlackRock	100
CDC	2430148	Active	32G16	12	58	July 17, 2015	March 2, 2025	8	55.52	$68.75	$2,500.00	$63 931.95	Minière BlackRock	100
CDC	2430149	Active	32H13	8	4	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 982.45	Minière BlackRock	100
CDC	2430150	Active	32H13	8	5	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 982.45	Minière BlackRock	100
CDC	2430151	Active	32H13	8	6	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 982.45	Minière BlackRock	100
CDC	2430152	Active	32H13	8	7	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 982.45	Minière BlackRock	100
CDC	2430153	Active	32H13	9	2	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430154	Active	32H13	9	3	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430155	Active	32H13	9	4	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430156	Active	32H13	9	5	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430157	Active	32H13	9	6	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430158	Active	32H13	9	7	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430159	Active	32H13	10	3	July 17, 2015	March 2, 2025	8	55.54	$68.75	$2,500.00	$63 957.20	Minière BlackRock	100
CDC	2430160	Active	32H13	10	4	July 17, 2015	March 2, 2025	8	55.54	$68.75	$2,500.00	$63 957.20	Minière BlackRock	100
CDC	2430161	Active	32H13	11	3	July 17, 2015	March 2, 2025	8	55.53	$68.75	$2,500.00	$63 944.57	Minière BlackRock	100
CDC	2430162	Active	32H13	12	3	July 17, 2015	March 2, 2025	8	55.52	$68.75	$2,500.00	$63 931.95	Minière BlackRock	100
CDC	2430163	Active	32H13	12	4	July 17, 2015	March 2, 2025	8	55.52	$68.75	$2,500.00	$63 931.95	Minière BlackRock	100
CDC	2430164	Active	32G16	3	44	July 17, 2015	March 2, 2025	8	55.60	$68.75	$2,500.00	$64 032.95	Minière BlackRock	100
CDC	2430165	Active	32G16	6	49	July 17, 2015	March 2, 2025	8	55.57	$68.75	$2,500.00	$62 995.08	Minière BlackRock	100
CDC	2430166	Active	32G16	7	51	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$63 982.45	Minière BlackRock	100
CDC	2430167	Active	32G16	8	52	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100

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CDC	2430168	Active	32G16	9	54	July 17, 2015	March 2, 2025	8	55.55	$68.75	$2,500.00	$63 969.82	Minière BlackRock	100
CDC	2430169	Active	32G16	6	51	July 17, 2015	March 2, 2025	8	55.57	$68.75	$2,500.00	$63 995.08	Minière BlackRock	100
CDC	2430170	Active	32H13	11	4	July 17, 2015	March 2, 2025	8	55.53	$68.75	$2,500.00	$158 513.04	Minière BlackRock	100
CDC	2430171	Active	32G09	28	37	July 17, 2015	March 2, 2025	8	9.57	$35.25	$1,000.00	$9 595.99	Minière BlackRock	100
CDC	2430172	Active	32G09	28	38	July 17, 2015	March 2, 2025	8	10.01	$35.25	$1,000.00	$10 151.55	Minière BlackRock	100
CDC	2430173	Active	32G09	28	39	July 17, 2015	March 2, 2025	8	1.10	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430174	Active	32G09	29	36	July 17, 2015	March 2, 2025	8	0.95	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430175	Active	32G09	29	37	July 17, 2015	March 2, 2025	8	54.82	$68.75	$2,500.00	$63 048.09	Minière BlackRock	100
CDC	2430176	Active	32G09	29	38	July 17, 2015	March 2, 2025	8	54.96	$68.75	$2,500.00	$63 224.86	Minière BlackRock	100
CDC	2430177	Active	32G09	29	39	July 17, 2015	March 2, 2025	8	11.59	$35.25	$1,000.00	$12 146.53	Minière BlackRock	100
CDC	2430178	Active	32G09	29	39	July 17, 2015	March 2, 2025	8	1.58	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430179	Active	32G09	30	36	July 17, 2015	March 2, 2025	8	6.27	$35.25	$1,000.00	$5 429.26	Minière BlackRock	100
CDC	2430180	Active	32G09	30	39	July 17, 2015	March 2, 2025	8	54.26	$68.75	$2,500.00	$62 341.01	Minière BlackRock	100
CDC	2430181	Active	32G09	30	40	July 17, 2015	March 2, 2025	8	51.25	$68.75	$2,500.00	$165 715.84	Minière BlackRock	100
CDC	2430182	Active	32G09	30	41	July 17, 2015	March 2, 2025	8	25.09	$68.75	$2,500.00	$25 509.71	Minière BlackRock	100
CDC	2430183	Active	32G09	30	42	July 17, 2015	March 2, 2025	8	20.66	$35.25	$1,000.00	$23 598.70	Minière BlackRock	100
CDC	2430184	Active	32G16	1	36	July 17, 2015	March 2, 2025	8	0.43	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430185	Active	32G16	1	42	July 17, 2015	March 2, 2025	7	55.50	$68.75	$2,500.00	$63 375.36	Minière BlackRock	100
CDC	2430186	Active	32G16	1	43	July 17, 2015	March 2, 2025	7	43.62	$68.75	$2,500.00	$48 906.50	Minière BlackRock	100
CDC	2430187	Active	32G16	1	44	July 17, 2015	March 2, 2025	8	7.32	$35.25	$1,000.00	$6 755.05	Minière BlackRock	100
CDC	2430188	Active	32G16	2	44	July 17, 2015	March 2, 2025	8	41.70	$68.75	$2,500.00	$46 482.23	Minière BlackRock	100
CDC	2430189	Active	32G16	2	45	July 17, 2015	March 2, 2025	8	13.64	$35.25	$1,000.00	$14 734.96	Minière BlackRock	100
CDC	2430190	Active	32G16	3	45	July 17, 2015	March 2, 2025	8	45.98	$68.75	$2,500.00	$51 886.33	Minière BlackRock	100
CDC	2430191	Active	32G16	3	46	July 17, 2015	March 2, 2025	8	11.36	$35.25	$1,000.00	$11 856.13	Minière BlackRock	100
CDC	2430192	Active	32G16	4	44	July 17, 2015	March 2, 2025	8	55.59	$68.75	$2,500.00	$50 573.19	Minière BlackRock	100
CDC	2430193	Active	32G16	4	45	July 17, 2015	March 2, 2025	8	55.59	$68.75	$2,500.00	$63 389.02	Minière BlackRock	100
CDC	2430194	Active	32G16	4	46	July 17, 2015	March 2, 2025	8	54.81	$68.75	$2,500.00	$63 035.48	Minière BlackRock	100
CDC	2430195	Active	32G16	4	47	July 17, 2015	March 2, 2025	8	32.96	$68.75	$2,500.00	$35 446.72	Minière BlackRock	100
CDC	2430196	Active	32G16	4	48	July 17, 2015	March 2, 2025	8	17.54	$35.25	$1,000.00	$19 659.27	Minière BlackRock	100
CDC	2430197	Active	32G16	4	49	July 17, 2015	March 2, 2025	8	6.12	$35.25	$1,000.00	$5 239.88	Minière BlackRock	100
CDC	2430198	Active	32G16	4	50	July 17, 2015	March 2, 2025	8	2.32	$35.25	$1,000.00	$441.84	Minière BlackRock	100
CDC	2430199	Active	32G16	5	45	July 17, 2015	March 2, 2025	8	55.58	$68.75	$2,500.00	$14 133.30	Minière BlackRock	100
CDC	2430200	Active	32G16	5	46	July 17, 2015	March 2, 2025	8	55.58	$68.75	$2,500.00	$42 568.03	Minière BlackRock	100
CDC	2430201	Active	32G16	5	50	July 17, 2015	March 2, 2025	8	44.73	$68.75	$2,500.00	$50 308.03	Minière BlackRock	100
CDC	2430202	Active	32G16	5	51	July 17, 2015	March 2, 2025	8	19.09	$35.25	$1,000.00	$21 616.36	Minière BlackRock	100
CDC	2430203	Active	32G16	5	52	July 17, 2015	March 2, 2025	8	15.30	$35.25	$1,000.00	$16 830.95	Minière BlackRock	100
CDC	2430204	Active	32G16	5	53	July 17, 2015	March 2, 2025	8	15.15	$35.25	$1,000.00	$16 641.55	Minière BlackRock	100
CDC	2430205	Active	32G16	5	54	July 17, 2015	March 2, 2025	8	8.18	$35.25	$1,000.00	$7 840.93	Minière BlackRock	100

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CDC	2430206	Active	32G16	6	46	July 17, 2015	March 2, 2025	8	55.57	$68.75	$2,500.00	$17 365.68	Minière BlackRock	100
CDC	2430207	Active	32G16	6	47	July 17, 2015	March 2, 2025	8	55.57	$68.75	$2,500.00	$42 770.05	Minière BlackRock	100
CDC	2430208	Active	32G16	6	54	July 17, 2015	March 2, 2026	9	40.81	$68.75	$2,500.00	$43 839.39	Minière BlackRock	100
CDC	2430209	Active	32G16	6	55	July 17, 2015	March 2, 2026	9	21.41	$35.25	$1,000.00	$21 104.86	Minière BlackRock	100
CDC	2430210	Active	32G16	6	56	July 17, 2015	March 2, 2025	8	14.49	$35.25	$1,000.00	$15 808.20	Minière BlackRock	100
CDC	2430211	Active	32G16	7	47	July 17, 2015	March 2, 2025	8	55.56	$68.75	$2,500.00	$749.29	Minière BlackRock	100
CDC	2430212	Active	32G16	7	54	July 17, 2015	March 2, 2026	9	55.56	$68.75	$2,500.00	$56 200.67	Minière BlackRock	100
CDC	2430213	Active	32G16	7	55	July 17, 2015	March 2, 2026	9	54.73	$68.75	$2,500.00	$8 220.23	Minière BlackRock	100
CDC	2430214	Active	32G16	7	55	July 17, 2015	March 2, 2025	8	0.84	$35.25	$1,000.00	$8 238.61	Minière BlackRock	100
CDC	2430215	Active	32G16	7	56	July 17, 2015	March 2, 2026	9	54.91	$68.75	$2,500.00	$39 584.29	Minière BlackRock	100
CDC	2430216	Active	32G16	7	57	July 17, 2015	March 2, 2025	8	53.26	$68.75	$2,500.00	$61 078.38	Minière BlackRock	100
CDC	2430217	Active	32G16	7	58	July 17, 2015	March 2, 2025	8	36.16	$68.75	$2,500.00	$39 487.18	Minière BlackRock	100
CDC	2430218	Active	32G16	7	59	July 17, 2015	March 2, 2025	8	22.49	$35.25	$1,000.00	$25 909.35	Minière BlackRock	100
CDC	2430219	Active	32G16	7	60	July 17, 2015	March 2, 2025	8	6.39	$35.25	$1,000.00	$5 580.79	Minière BlackRock	100
CDC	2430220	Active	32G16	8	55	July 17, 2015	March 2, 2026	9	55.55	$68.75	$2,500.00	$54 660.25	Minière BlackRock	100
CDC	2430221	Active	32G16	8	56	July 17, 2015	March 2, 2026	9	55.47	$68.75	$2,500.00	$6 338.90	Minière BlackRock	100
CDC	2430222	Active	32G16	8	56	July 17, 2015	March 2, 2025	8	0.08	$35.25	$1,000.00	$8 238.61	Minière BlackRock	100
CDC	2430223	Active	32G16	8	57	July 17, 2015	March 2, 2026	9	55.55	$68.75	$2,500.00	$41 718.16	Minière BlackRock	100
CDC	2430224	Active	32G16	8	58	July 17, 2015	March 2, 2025	8	47.92	$68.75	$2,500.00	$54 335.86	Minière BlackRock	100
CDC	2430225	Active	32G16	8	60	July 17, 2015	March 2, 2025	8	54.33	$68.75	$2,500.00	$62 429.41	Minière BlackRock	100
CDC	2430226	Active	32G16	9	56	July 17, 2015	March 2, 2026	9	55.55	$68.75	$2,500.00	$68 688.22	Minière BlackRock	100
CDC	2430227	Active	32G16	9	57	July 17, 2015	March 2, 2026	9	54.97	$68.75	$2,500.00	$44 041.42	Minière BlackRock	100
CDC	2430228	Active	32G16	9	57	July 17, 2015	March 2, 2025	8	0.57	$35.25	$1,000.00	$8 238.61	Minière BlackRock	100
CDC	2430229	Active	32G16	9	58	July 17, 2015	March 2, 2026	9	18.85	$35.25	$1,000.00	$28 390.56	Minière BlackRock	100
CDC	2430230	Active	32G16	9	59	July 17, 2015	March 2, 2025	8	33.30	$68.75	$2,500.00	$35 876.03	Minière BlackRock	100
CDC	2430231	Active	32G16	9	60	July 17, 2015	March 2, 2025	8	24.13	$35.25	$1,000.00	$27 980.08	Minière BlackRock	100
CDC	2430232	Active	32G16	10	58	July 17, 2015	March 2, 2025	8	36.45	$68.75	$2,500.00	$39 853.35	Minière BlackRock	100
CDC	2430233	Active	32G16	10	59	July 17, 2015	March 2, 2025	8	2.10	$35.25	$1,000.00	$164.05	Minière BlackRock	100
CDC	2430234	Active	32G16	10	59	July 17, 2015	March 2, 2025	8	0.01	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430235	Active	32G16	11	59	July 17, 2015	March 2, 2025	8	36.93	$68.75	$2,500.00	$40 459.42	Minière BlackRock	100
CDC	2430236	Active	32G16	11	60	July 17, 2015	March 2, 2025	8	1.96	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430237	Active	32G16	12	59	July 17, 2015	March 2, 2025	8	34.10	$68.75	$2,500.00	$36 886.14	Minière BlackRock	100
CDC	2430238	Active	32G16	13	58	July 17, 2015	March 2, 2025	8	55.26	$68.75	$2,500.00	$0.00	Minière BlackRock	100
CDC	2430239	Active	32G16	13	59	July 17, 2015	March 2, 2025	8	30.97	$68.75	$2,500.00	$32 934.06	Minière BlackRock	100
CDC	2430240	Active	32G16	13	60	July 17, 2015	March 2, 2025	8	25.66	$68.75	$2,500.00	$22 252.10	Minière BlackRock	100
CDC	2430241	Active	32G16	12	60	July 17, 2015	March 2, 2025	8	0.03	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430242	Active	32G16	13	60	July 17, 2015	March 2, 2025	8	0.15	$35.25	$1,000.00	$0.00	Minière BlackRock	100
CDC	2430243	Active	32H13	8	1	July 17, 2015	March 2, 2025	8	46.19	$68.75	$2,500.00	$52 151.49	Minière BlackRock	100

Type	Index	Status	SNRC Leaf	Range/Block	Lot/Block	Registration Date	Expiry Date	Renewal	Area (has)	Required Fees	Work Required	Exceeding Credits	Owner	Percentage
CDC	2430244	Active	32H13	8	2	July 17, 2015	March 2, 2025	8	30.47	$68.75	$2,500.00	$32 302.74	Minière BlackRock	100
CDC	2430245	Active	32H13	8	3	July 17, 2015	March 2, 2025	8	14.00	$35.25	$1,000.00	$15 189.50	Minière BlackRock	100
CDC	2430246	Active	32H13	9	1	July 17, 2015	March 2, 2025	8	49.44	$68.75	$2,500.00	$56 255.08	Minière BlackRock	100
CDC	2430247	Active	32H13	10	1	July 17, 2015	March 2, 2025	8	5.77	$35.25	$1,000.00	$4 797.96	Minière BlackRock	100
CDC	2430248	Active	32H13	10	2	July 17, 2015	March 2, 2025	8	17.64	$35.25	$1,000.00	$19 785.53	Minière BlackRock	100
CDC	2430249	Active	32H13	10	5	July 17, 2015	March 2, 2025	8	38.62	$68.75	$2,500.00	$42 593.28	Minière BlackRock	100
CDC	2430250	Active	32H13	10	6	July 17, 2015	March 2, 2025	8	36.41	$68.75	$2,500.00	$39 802.85	Minière BlackRock	100
CDC	2430251	Active	32H13	10	7	July 17, 2015	March 2, 2025	8	2.44	$35.25	$1,000.00	$593.36	Minière BlackRock	100
CDC	2430252	Active	32H13	11	2	July 17, 2015	March 2, 2025	8	5.03	$35.25	$1,000.00	$3 863.60	Minière BlackRock	100
CDC	2430253	Active	32H13	11	5	July 17, 2015	March 2, 2025	8	7.15	$35.25	$1,000.00	$6 540.40	Minière BlackRock	100
CDC	2430254	Active	32H13	12	1	July 17, 2015	March 2, 2025	8	10.75	$35.25	$1,000.00	$11 085.92	Minière BlackRock	100
CDC	2430255	Active	32H13	12	2	July 17, 2015	March 2, 2025	8	28.82	$68.75	$2,500.00	$30 219.38	Minière BlackRock	100
CDC	2430256	Active	32H13	12	5	July 17, 2015	March 2, 2025	8	15.40	$35.25	$1,000.00	$16 957.21	Minière BlackRock	100
CDC	2430257	Active	32H13	13	1	July 17, 2015	March 2, 2025	8	38.99	$68.75	$2,500.00	$43 060.46	Minière BlackRock	100
CDC	2430258	Active	32H13	13	2	July 17, 2015	March 2, 2025	8	38.69	$68.75	$2,500.00	$42 681.67	Minière BlackRock	100
CDC	2430259	Active	32H13	13	3	July 17, 2015	March 2, 2025	8	38.34	$68.75	$2,500.00	$42 239.74	Minière BlackRock	100
CDC	2430260	Active	32H13	13	4	July 17, 2015	March 2, 2025	8	38.10	$68.75	$2,500.00	$41 936.71	Minière BlackRock	100
CDC	2430261	Active	32H13	13	5	July 17, 2015	March 2, 2025	8	30.85	$68.75	$2,500.00	$31 582.55	Minière BlackRock	100
CDC	2430262	Active	32G16	8	59	July 17, 2015	March 2, 2025	8	55.49	$68.75	$2,500.00	$62 077.94	Minière BlackRock	100
CDC	2525657	Active	32G16	5	44	November 2, 2018	November 1, 2025	2	55.58	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525658	Active	32G16	6	43	November 2, 2018	November 1, 2025	2	55.57	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525659	Active	32G16	6	44	November 2, 2018	November 1, 2025	2	55.57	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525660	Active	32G16	6	45	November 2, 2018	November 1, 2025	2	55.57	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525661	Active	32G16	7	43	November 2, 2018	November 1, 2025	2	55.56	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525662	Active	32G16	7	44	November 2, 2018	November 1, 2025	2	55.56	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525663	Active	32G16	7	45	November 2, 2018	November 1, 2025	2	55.56	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525664	Active	32G16	7	46	November 2, 2018	November 1, 2025	2	55.56	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525665	Active	32G16	8	43	November 2, 2018	November 1, 2025	2	55.55	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525666	Active	32G16	8	44	November 2, 2018	November 1, 2025	2	55.55	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525667	Active	32G16	8	45	November 2, 2018	November 1, 2025	2	55.55	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525668	Active	32G16	9	44	November 2, 2018	November 1, 2025	2	55.54	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525669	Active	32G16	9	45	November 2, 2018	November 1, 2025	2	55.54	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525670	Active	32G16	9	46	November 2, 2018	November 1, 2025	2	55.54	$68.75	$1,200.00	$0.00	Minière BlackRock	100
CDC	2525671	Active	32H13	15	10	November 2, 2018	November 1, 2025	2	55.49	$68.75	$1,200.00	$1 537.22	Minière BlackRock	100
CDC	2525672	Active	32H13	15	11	November 2, 2018	November 1, 2025	2	55.49	$68.75	$1,200.00	$337.22	Minière BlackRock	100
CDC	2525673	Active	32H13	16	10	November 2, 2018	November 1, 2025	2	55.48	$68.75	$1,200.00	$337.22	Minière BlackRock	100
CDC	2525674	Active	32H13	16	11	November 2, 2018	November 1, 2025	2	55.48	$68.75	$1,200.00	$337.22	Minière BlackRock	100
CDC	2525675	Active	32H13	16	12	November 2, 2018	November 1, 2025	2	55.48	$68.75	$1,200.00	$337.22	Minière BlackRock	100

Type	Index	Status	SNRC Leaf	Range/Block	Lot/Block	Registration Date	Expiry Date	Renewal	Area (has)	Required Fees	Work Required	Exceeding Credits	Owner	Percentage
CDC	2525676	Active	32H13	20	6	November 2, 2018	November 1, 2025	2	55.44	$68.75	$1,200.00	$337.22	Minière BlackRock	100
CDC	2525677	Active	32H13	20	7	November 2, 2018	November 1, 2025	2	55.44	$68.75	$1,200.00	$337.22	Minière BlackRock	100
CDC	2525678	Active	32H13	20	8	November 2, 2018	November 1, 2025	2	55.44	$68.75	$1,200.00	$337.22	Minière BlackRock	100
CDC	2525679	Active	32H13	21	6	November 2, 2018	November 1, 2025	2	55.43	$68.75	$1,200.00	$337.24	Minière BlackRock	100
	230								**8,677**	**$ 13,367**	**$ 428,800**	**$ 6,730,133**		
BNE	48194	Active	32G09			April 8, 2020	March 31, 2024			$307.00				
BNE	48195	Active	32G09			April 8, 2020	March 31, 2024			$307.00				
	2									**$614.00**				

Note that Exceeding Credits from claims within a 4.5km radius can be used for claims that have no current Exceeding Credits themselves, and all claims in the list are current as at the date of this data download from MERN (September 30, 2021).

4.3. The Québec Mining Act and Claims

The Québec Mining Act deals with the management of Mineral Resources and the granting of exploration rights for mineral substances during the exploration phase. It also deals with the granting of rights pertaining to the use of these substances during the mining phase. The act also establishes the rights and obligations of the holders of mining rights to ensure maximum development of Québec's Mineral Resources.

The claim is the only valid exploration right in Québec. The claim gives the holder an exclusive right to search for mineral substances in the public domain, with the exception of sand, gravel, clay and other loose deposits, on the land subjected to the claim. Since November 2000, exploration titles are obtained by map designation over predetermined parcels of land. This approach is quicker and simpler, rendering claims indisputable and protecting the investments made on a claim.

The term of a claim is two years, from the day the claim is registered, and it can be renewed indefinitely providing the holder meets all the conditions set out in the Mining Act, including the obligation to invest a minimum annual amount required in exploration work determined by regulation. The Act includes provisions to allow any amount disbursed to perform work in excess of the prescribed requirements to be applied to subsequent terms of the claim.

To satisfy government assessment requirements and thus maintain the claim(s) in good standing, minimum exploration expenditures must be incurred and filed 60 days prior to the anniversary date(s) of the claim(s). The report of work is due prior to 60 days of the anniversary date. In Québec, the amount of expenditure per claim varies according to the surface area of the claim, location (either north or south of 52° latitude) and the number of terms since its issuance, which escalates according to the following schedules:

Table 4-2 and Table 4-3 show the amount of assessment work to be carried out during each term of a claim.

Table 4-2: South of 52° of latitude

Term	Surface Area of Claim		
	< 25 ha	25 – 100 ha	> 100 ha
1 to 3	$500	$1,200	$1,800
4 to 6	$750	$1,800	$2,700
7 or more	$1,000	$2,500	$3,600

Table 4-3: North of 52° of latitude

Term	Surface Area of Claim		
	< 25 ha	25 – 45 ha	> 45 ha
1	$48	$120	$135
2	$160	$400	$450
3	$320	$800	$900
4	$480	$1,200	$1,350
5	$640	$1,600	$1,800
6	$750	$1,800	$1,800
7 or more	$1,000	$2,500	$2,500

Assessment work credits from another claim may be applied to the claim to be renewed, providing the renewed claim lies within a radius of 4.5 km from the centre of the claim with the excess work credits. The claim holder may apply amounts spent on work carried out on a mining lease or concession towards the renewal of a claim, provided that the work was performed during the term of the claim and that the amount does not exceed one quarter of the required amount for renewal. If the required work was not performed or was insufficient to cover the renewal of the claim, then the claim holder may pay a sum equivalent to the minimum cost of the work that should have been performed.

The cost of the renewal of a claim depends on the surface area of the claim, its location, and the date the application is received. If the application for renewal and fees are received prior to 60 days before the anniversary of the claims(s) the following renewal fees apply for claims north of 52° latitude: less than 25 ha = $26; 25 to 45 ha = $96; 45 to 50 ha = $107; over 50 ha = $120.

For claims south of 52° latitude the following renewal fees apply: less than 25 ha = $26; 25 to100 ha = $52; over 100 ha = $78. These renewal fees double if the application is received within 60 days or less of the anniversary date of the claim(s). The BlackRock Property is located south of the 52° latitude; therefore the latter fees apply on the project.

4.4. Permits

Several regulations are in place to protect the environment and the people residing in areas affected by the Project. In order to abide by the Government's laws, certain permits are required for the Project to start. To the Qualified Person's knowledge, all permits needed to make this project progress have been acquired or in the process to be. No unforeseen permitting issues were conveyed to the QP. A more detailed discussion on the permitting process can be found in Section 20 (Sections 20.1.4 and 20.2.3).

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1. Accessibility

The project area is accessible from the paved Highway 167, between Chibougamau and St-Félicien (Lac-St-Jean). At kilometre 200, access is provided by the forestry road No. 210 (known as "*Chemin de la mine Lemoine*" or "*Chemin Gagnon Frères*"). A network of forestry roads accesses the deposit at different locations from the Lemoine road. Some upgrading or maintenance will be needed on these roads for regular access. A distance of about 85 kilometres by road separates the centre of the Property from the Town of Chibougamau. The Property can also be reached from the north by way of another gravel road (locally known as Cigam road).

The former Lac Audet railroad siding of CNR Chibougamau-St-Félicien line passes about 12 km to the SW of the southwest extremity of the Property, following the forestry road. A deep-water port in Québec City, some 600 km southeast along the railroad, and the Saguenay Port in Grande-Anse, 400 km from the mine site, are available. A commercial airport is located between the towns of Chibougamau and Chapais, about 60 km from the Property. Most of the Property is not within the range of cellular phone towers.

Further details regarding project infrastructure can be found in Section 18 of the report.

5.2. Local Resources and Infrastructure

The cities of Chibougamau and Chapais, both copper and gold mining centers, have a combined population of 11,000 inhabitants, with the addition of the Cree community of Mistassini and Ouje-Bougoumou, with a population of about 3,000 inhabitants. Besides mining, the economy is based on lumbering and sawmills. Social, educational, commercial, medical and industrial services, a helicopter base, an airport and a seaplane base, as well as forestry and mining offices of the MRN (*Ministère des Ressources Naturelles*) are available at the town site. Chibougamau is a mining community and has abundant skilled manpower and equipment availability. It is well serviced by heavy equipment suppliers and maintenance providers.

As previously stated, the Chibougamau area is peppered with past operators and currently active operations. There are no major impediments to operating yearlong except for normal short-term slowdowns during inclement weather.

No infrastructure, with the exception of the logging roads, is present within the Property. A high-voltage (161 kV) power line is accessible close to the railroad. Further details regarding Project Infrastructure are available in Section 18 of the report.

Water is available locally. Other regional infrastructure is available in the nearby town of Chibougamau, including old mines and mills, 161 kV and 735 kV power stations, a hospital, an airport, various shops, road construction services, etc. Daily railroad freight service is available, linking the North American rail network, although the traffic density is low. The rail line will require some improvements in order to handle ore cars loaded with magnetite and ilmenite concentrates. Loading facilities are available in Chibougamau.

5.3. Climate and Physiography

A cold continental climate prevails in the Chibougamau area. It is characterized by mild summers (16°C July average) and cold winters (-20°C January average). The average annual precipitation is in the order of 919 mm of water, with prevailing winds from the west. Snow is present typically from late October to early May.

The deposit is located on a north-easterly trending hill culminating at an elevation of 535 m. It is limited to the north by the Lac Doré lowlands at an elevation of 410 m. A topographic high, reaching 120 m, is therefore present at the site. The mineralized zone extends parallel to the topography, as expressed by the crest of the elongated hill. The Property follows the height of land between the St-Lawrence River to the south and James Bay to the north. Drainage to the north of the watershed flows toward Chibougamau Lake, via the Villefagnan and Armitage rivers. Drainage to the south of the watershed flows toward Lac St-Jean, by way of the Boisvert River. The Property is well drained, with limited bogs and swamps near the Armitage, Bernadette, and Jean lakes, which are the only large bodies of water within the project boundaries.

The Property is covered by commercial taiga forest, dominated by black spruce and poplar. Most of the Property has been logged and reforested in the last three decades. Vegetation is typically second growth taiga forest. Wildlife consists of abundant black bears, moose, beavers, etc.

6. History

6.1. Previous Exploration Work

As mentioned in Girard (2008) from IOS Service Géoscientifique Inc., the Property being located in the vicinity of a historic mining district, abundant government and academic literature is available. More than 400 reports and maps are available for NTS 32G/16 and 32H/13 map-sheets. A thorough review of all literature is not considered relevant to the current report. The relevant governmental work is considered to be those of Allard (RP-567, 1967; RP-566, 1969; RP-589, 1970; and DPV-759, 1981) who mapped and described the magnetite series of the Lac Doré anorthositic complex, and from which Dr. Allard predicted the vanadium occurrence. A regional compilation of the complex was prepared by Daigneault and Allard (1990, MM-89-03).

Over time, the entire iron-titanium-vanadium mineralized belt located south-east of Lake Chibougamau, has been held and explored by numerous companies. The bulk part of the exploration has been carried out on the East and West Deposits (sometimes referred to as the Lake Doré Deposit), located to the northeast of the BlackRock Property. Due to the lack of detailed information, the total amount of exploration expenses exclusive to the BlackRock Property is unknown.

In 1947-1948, Dominion Gulf discovered the magnetite deposit after an aeromagnetic survey (GM-1028, GM-3873). In 1954, Dominion Gulf sent a 220 pound (99 kg) sample for mineralogical and metallurgical works (GM-3640). In 1956 (GM-4411, GM-4653), Dominion Gulf carried out geological mapping, trenching and stripping, and a ground magnetometer survey. A 1,400 pound (630.5 kg) sample was collected for metallurgical testing. Part of Dominion Gulf exploration work covered the area of the Southwest Zone.

In 1958, subsequent exploration work by Jalore Mining (a subsidiary of Jones and Laughlin Steel from Pittsburgh) and Continental Ore Corporation included a ground magnetic survey, six diamond drillholes, geological mapping and some metallurgical testing (GM-07301, GM-08572-A & B, GM-11061, GM-27165). Approximate location of the drillholes indicate they were drilled in the northeast extension of the Southwest Zone, on the adjacent property. The Property was dropped because of the high titanium content rendering it unsuitable as iron ore at that time. The core was not assayed for vanadium.

From 1957 to 1959, Trepan Mining Corporation carried out a limited amount of work over the Armitage extension (GM-06047, GM-06482, GM-10012). This included geological mapping, a ground magnetometer survey and three diamond drillholes. The exact location of drillholes is not available, although they are suspected of being near the former forestry road leading to the Armitage Lake. The core was assayed for iron and titanium only.

The vanadium content of the titanium rich magnetite layers was noted by Dr. Allard (1967, 1967b) (DP-076, RP-567), while working for the Québec Department of Natural Resources. The deposit was then staked on behalf of the Crown and from 1966 to 1975, the following work was completed:

- Geological mapping (Allard 1967 (DP-076); Allard and Caty, 1969 (RP-567));
- Ground magnetometer survey, lLine cutting and 19 exploratory diamond drillholes (Kish 1971, Avramtchev 1975), of which two are on the Southwest Deposit of BlackRock;
- Line cutting, geological mapping and surveying (Gobeil, 1976, DP 354);
- Bulk sampling: 30 tonnes, 200 tonnes, and 600 tonnes;
- Numerous metallurgical tests, both for alkali roasting and steel-slag smelting (Assad, 1967, 1968; Boulay & al., 1969; Cloutier & al. 1971; Castonguay, 1975a, 1975b; Canmet, 1976; CRM 1979; QIT 1978; CRIQ, Union Carbide, IRSID (France), Ontario Research Foundation 1975);
- Preliminary resource estimates on the Southwest, West and East Deposits (Assad 1968; Kish 1971; Cloutier & al. 1971; Avramtchev 1975). Details are found in Section 6.3.

The Lac Doré vanadium project was transferred from the MNRQ to SOQUEM, a Québec crown corporation, in 1977. SOQUEM did some geological and geophysical work until 1979 at which time a resource was established, followed by additional metallurgical testing until 1980. Work by SOQUEM was mainly carried out on the West and East Deposits, northeast of the Southwest Deposit. In 1981, SOQUEM abandoned the development program due to a weakening vanadium market.

From 1983 to 1989, the project was reviewed and evaluated for its economic potential by various groups on behalf of SOQUEM:

- 1983: CRM (Malensky and Castonguay, 1983);
- 1989: Hydro-Québec (Unsigned, 1989);
- 1989: Société Générale de Financement (Vallée, 1989);
- 1989: Hatch and Associates (Lachapelle, 1989).

In 1997, McKenzie Bay Resources Ltd. optioned the remaining SOQUEM's claims and staked new ones. The McKenzie Bay Property was covering all the known occurrences, including the Southwest and Armitage Deposits now belonging to BlackRock.

The Property was optioned to Cambior from 1998 to 2000. Cambior/McKenzie Bay conducted large geological mapping, stripping, sampling, drilling and then a new resource/reserve estimate on the East and West Deposits. Cambior dropped their option in 2000. In 2001-2002, McKenzie Bay contracted SNC-Lavalin in order to perform a scoping study including metallurgical testing at Lakefield Research Limited on the Lac Doré Deposit (East and West Deposit).

The bulk of McKenzie Bay's work from 1997 to 2002 was performed on the Lac Doré Deposit, outside the boundary of the contiguous BlackRock claims. More specifically, work performed on BlackRock's Property consisted of:

- Line cutting, a ground magnetometer survey;
- Stripping and detailed mapping and sampling of 11 trenches on the BlackRock Property itself;
- Ore microscopy and microprobe analyses (Lamontagne 1997, Lamontagne et Lavoie 1997, Bédard 1998);
- Drilling 11 holes totalling 1604.7 m on the BlackRock Property.

In 2006, government geologists compiled all available data over the region located south of Lake Chibougamau, including most of the BlackRock Property, at a scale of 1:50,000.

6.2. Historical Drilling

As previously mentioned, most operators concentrated their exploration efforts immediately to the northeast of the Southwest Deposit, outside BlackRock's claims. Notwithstanding, there were three phases of drilling completed on the BlackRock claims specifically:

1. From 1957 to 1959, Trepan Mining Corporation drilled three diamond drillholes totalling 454 m (GM-10012). Holes were assayed for iron and titanium. The exact hole locations are not available, but they were definitively drilled on the Armitage Deposit.
2. In 1970, of their 10-hole drilling program, the Ministry of Natural Resources drilled holes #9 and #10 on the Southwest Deposit, for a total of 318 m (DP 053). These holes were assayed for iron, titanium and vanadium.
3. In 2001, as part of their exploration program, McKenzie Bay drilled 11 holes directly on BlackRock Property. Four holes totalling 702 m were drilled on the Armitage Deposit. Three holes totalling 468 m were drilled on the Southwest Deposit. Finally, four exploration holes totalling 579 m were drilled in the extreme northeast corner of the Property. Holes were assayed for iron, titanium and vanadium.

6.3. Historical Resource Estimates

As briefly mentioned in the above subsections, there have been some resource estimates centred on the Lac Doré Deposit, mainly on the East and West Deposits. Few calculations were done on BlackRock's Southwest Deposit.

These historical resource/reserve calculations were done in several ways by using data collected from geological mapping, magnetometer and gravimetric surveys, trenching, diamond drilling, bulk sampling results, etc. None of these historical resource/reserve calculations were done using the actual NI 43-101 standards, they are reported in this report for information purposes and shall thus be used with much caution. None of these historical data, neither resource/reserve calculation was used or included by BlackRock for the resource/reserve calculation of the Southwest and Armitage Deposits in this NI 43-101 report. A qualified person has not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves; and, the issuer is not treating the historical estimate as current Mineral Resources or Mineral Reserves.

In 1968, on behalf of the MRNQ, Assad (DP-005, GM-23154) roughly reported a resource of 70 Mt of mineralization with no grade on the Southwest, West and East Deposits. It seems that this was simply based on the width and length of the magnetite mineralization down to a vertical depth of 61 m. A qualified person has not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves; and, the issuer is not treating the historical estimate as current Mineral Resources or Mineral Reserves.

In 1971, also on behalf of the MRNQ, Kish (DP-053) and Cloutier & al. (GM-30396) made resource/reserve calculations for the Southwest, West and East Deposits based on geological data, diamond drilling assays, Davis tube and density assays, and down to a depth of 152 m. It is the first time that resource/reserve calculations were specifically done on the Southwest Deposit on the BlackRock Deposit. The reported historical resources are:

Table 6-1: Reported resources

Deposit	Tons	Fe%	V_2O_5%
East Deposit	28 187 841	33.39	0.52
West Deposit	21 605 651	30.53	0.48
Southwest Deposit	22 490 406	29.19	0.48
Total and average	72 283 898	31.27	0.50

Note: A qualified person has not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves; and, the issuer is not treating the historical estimate as current Mineral Resources or Mineral Reserves.

In 1975, Avramtchev (DP-309) released an updated resource/reserve calculation for the East Deposit, based on new drillholes from the MRNQ. There was no change for the Southwest Deposit.

In 1980, after a new drilling and geophysical program, SOQUEM (GM-36918), reported a new resource/reserve calculation for the East Deposit. There was no change for the Southwest Deposit.

In 2002, the scoping study by SNC-Lavalin released a combined updated resource for the Southwest, West and East Deposits based on all the work done by McKenzie Bay and Cambior, two new holes and a bulk samples. This new resource was done using a block model and kriging as the interpolation method for the grade. SNC-Lavalin reported Measured and Indicated resources of:

- 102 million tonnes grading 35% magnetite (25.3% Fe_T), 17.4% ilmenite (9.2% TiO_2) and 0.50% V_2O_5.

It should be noted that resource estimates quoted in this aforementioned Feasibility Study and completed by SNC-Lavalin in 2002, should be considered as speculative and unreliable. A qualified person has not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves; and, the issuer is not treating the historical estimate as current Mineral Resources or Mineral Reserves.

All of the Mineral Resource calculations stated above are considered historical resource estimates. Most of the key assumptions, parameters and methods used are not fully known as original data, reports and other necessary information is not available, missing or no longer existing. As such, no qualified person has not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves; and, the issuer is not treating the historical estimate as current Mineral Resources or Mineral Reserves.

In 2008, BlackRock acquired the project area outside of the defined resources from McKenzie Bay. This area was then extensively drill tested on 200 m drill panels and 100 m holes spacing, with about 25% with infill drilling done on 100 m drill panels. Work was completed by 2011 on the Southwest and Armitage Deposits, which culminated in a NI 43-101 compliant Measured and Indicated resource of:

- 530 million tonnes grading 24.6% Fe_T, 7.2% TiO_2* and 0.42% V_2O_5*

  * average taken from the 292-tonne resource that falls within the pit design.

The above SGS 2011 historical resource estimates were never publicly released by BlackRock yet, they form the basis of any subsequent Mineral Resource estimate. The data, key assumptions, parameters and methods are fully known. The current resource estimate builds on this 2011 work.

6.4. Historical Technical and Environmental Studies

There was significant work completed by McKenzie Bay on the project area, but they were focused predominantly on vanadium production to the northeast of the Southwest Deposit. Therefore, most of the mining and processing plan does not apply to the current Feasibility Study. The environmental baseline work that had been done was more useful, and BlackRock engaged the same contractors to complete the studies on its behalf.

6.5. BlackRock Exploration and Development History

In late 2008, BlackRock carried out an airborne magnetometric survey covering the entire length of the Lac Doré Complex (Novatem, 2008). An interpretation report was produced in March 2009 by Gérard Lambert Geosciences Ltee (Lambert, 2009). The company has completed a compilation of all analytic data into a master database.

7. Geological Setting and Mineralization

The BlackRock Project is located within the Abitibi Terrain of the Superior Province within the Canadian Shield. It forms part of the Archean package of rocks (2.7 Ga) that comprise the core of the North American craton (Figure 7-1).

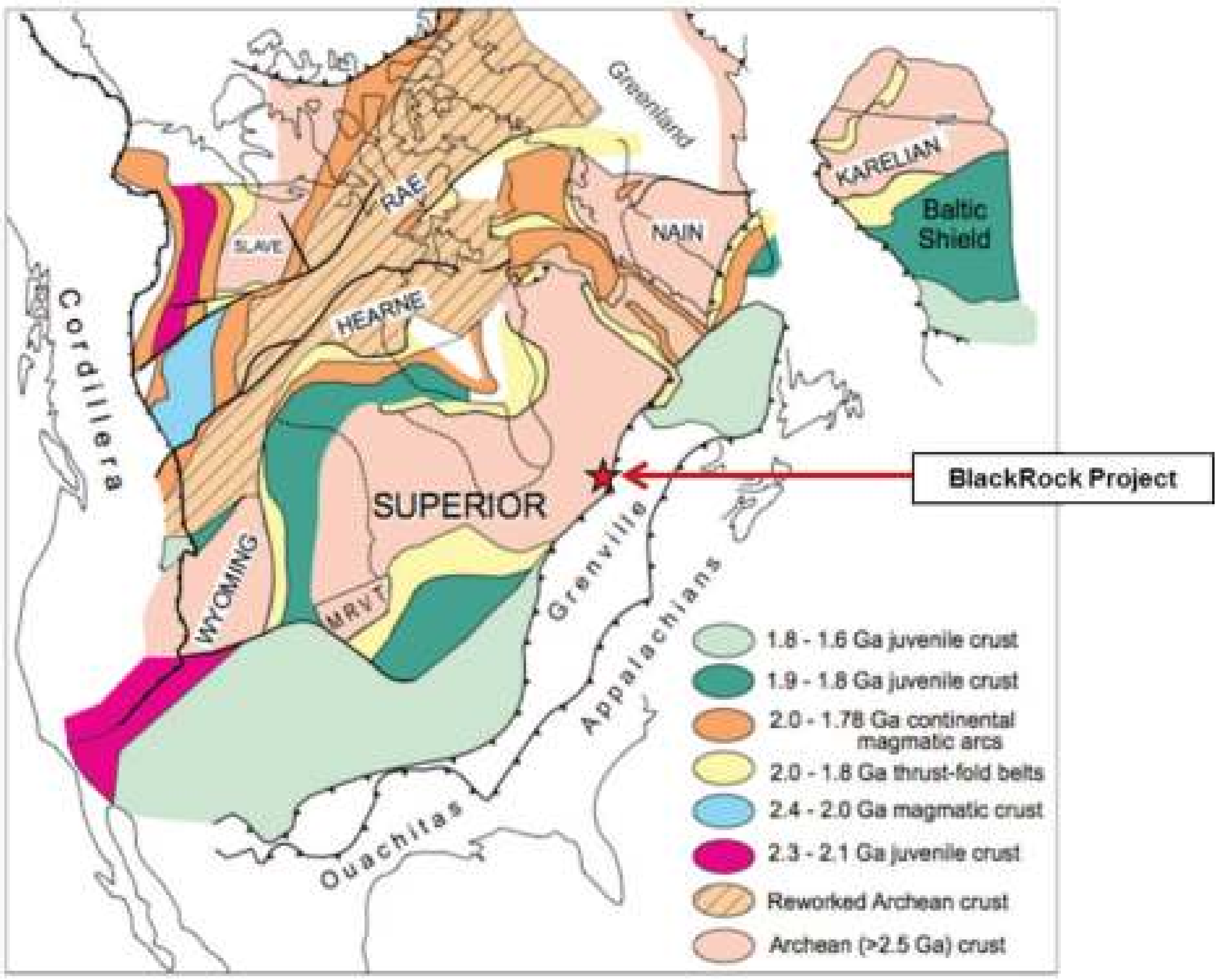


Figure 7-1: Cratonic map of North America

The BlackRock Project is shown by the red star on the edge of the Superior Province (After Percival 2007).

7.1. Regional Geology

The Chibougamau geological region is located near the contact between the East-West trending Archean volcanic and sedimentary rocks of the Superior Province and the late Proterozoic Grenville Province. More precisely, the Chibougamau area is at the eastern end of the Abitibi Greenstone Belt (AGB), a part of the Abitibi Sub-province of the larger Superior Province. Overall, rock units of the AGB have been metamorphosed to the greenschist facies and metamorphic grade increases to amphibolite facies near the Grenville Front.

In the Chibougamau region, Archean rocks have been divided into the volcanic Roy Group, at the base, overlain by volcano-sedimentary Opemisca Group. Contact between both groups corresponds to a major regional unconformity.

The Roy Group consists of five formations, the four lower ones forming two distinct volcanic cycles. Cycle 1, at the base, is composed of the basaltic units of Obatogamau Formation overlain by the felsic volcanites of the Waconichi Formation. Cycle 2 is composed of the basalt of Gilman Formation overlain by the felsic volcanoclastites of Blondeau Formation. Finally, at the top of the Roy Group, there is the felsic to intermediate volcaniclastites of the Bordeleau Formation.

The Opemica Group is composed of the three epiclastic sedimentary horizons. At the base, there is the Stella and Chebistuan Formations overlain by the Haüy Formation at the top.

Numerous synvolcanic mafic to ultramafic sills and various synvolcanic, syntectonic to post-tectonic granitic intrusion have intruded the volcanic and sedimentary sequences of the Chibougamau region. One of the most important of them is the anorthositic Lac Doré Complex (LDC), at the base of the Waconichi Formation that hosts numerous of the Copper-Gold (Cu-Au) mines of the Chibougamau mining camp and more importantly, the Vanadium-Titanomagnetite (VTM) mineralization of the BlackRock Project.

All rock units were affected by multiple deformation events as folding, faulting and shearing. Lithologies trend predominantly east-west and dip steeply to the north and the south. They are folded and they form a succession of E-W trending anticlines and synclines. The late Chibougamau pluton that occupies the core of the Chibougamau anticline has intruded and truncated the LDC.

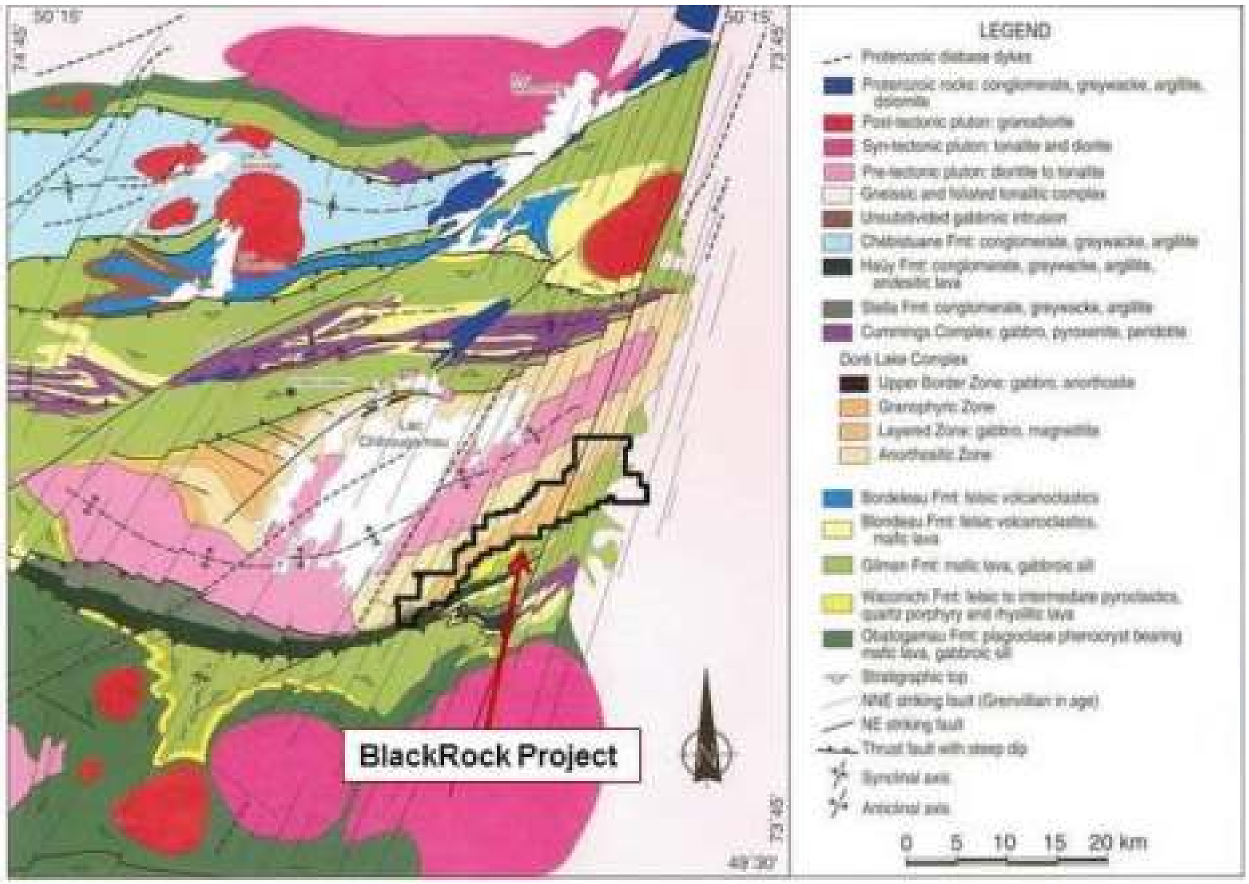


Figure 7-2: Regional map of the geologic framework surrounding the Lac Doré Complex.

The outline of BlackRock's 2017 claim position is shown for relative location within the district

Note the plunging anticline cored by Lac Chibougamau. The BlackRock Project lies on the southern limb of this anticline.

Faults and shear zones trend predominantly E-W and NE to NNE. NW faults are also reported. Large-scale synclines and anticlines are generally bound by regional synvolcanic/sedimentary and syntectonic E-W faults. Late NE-NNE faults dissect the area. They are either associated with or reactivated by the Grenvillian event.

On the mining side, from 1954 to 1990, the Chibougamau/Chapais region had been one of the most important mining districts in Québec. During that period, total combined mining production had reached 1.2 Mt of copper (Cu), 115,000 kg of gold (Au), 650,000 kg of silver (Ag), 115,000 kg of zinc (Zn) and 4,000 kg of lead (Pb).

Past economic copper, gold, zinc and silver mineralization of the previous mines can be classified under four main categories:

1. Magmatic-hydrothermal and porphyritic-type Cu-Au mineralization of the Lac Doré mining camp in Chibougamau;
2. Opemiska-type Cu veins of the Chapais region;
3. Cu-Zn-Au-Ag volcanic massive sulphide deposits of the Lemoine mine;
4. Archean mesothermal gold mineralization of the Joe Mann and Lac Shortt mines.

The VTM mineralization in the Lac Doré Complex on the BlackRock Property is one of the most important mineral occurrences in the Chibougamau region. It is a magmatic-type iron mineralization related to layered mafic intrusions. It was discovered in the 1950's and it has been sporadically explored for vanadium since then. The economic potential of the iron and titanium has been identified and delineated by BlackRock Metals over the last five years.

7.2. The Lac Doré Complex

As described by Girard and Allard (1998), the Lac Doré Complex (LDC) T is a differentiated ultramafic-mafic sill of the Archean age, which is observed in the heart of the Chibougamau anticline (Figure 7-2). Its flanks outcrop on the north and south shores of the Chibougamau Lake. It was emplaced during the arc-magmatic and collisional stages of orogenesis at 2.727 Ga. Shortly after emplacements, the LDC was folded into a broad anticline during continued compressive accretion of the Abitibi-Wawa Terrane between 2.698-2.690 Ga (Figure 7-2; Daigneault and Allard, 1990).

The project is located on the eastern edge of the Superior Province where it has been affected by low-grade thermal effects along the accretion contact with the much younger Grenville Province (1.1 Ga).striking

The LDC stratigraphy is basically composed of thick sequences of anorthosite that encase the copper-gold deposits. The anorthosite is overlain by a layered sequence that contains vanadiferous magnetite and ilmenite, layered ferrogabbros and ferropyroxenites. The layered sequence is finally surmounted by a granophyre and a contact zone with the felsic volcanic rocks of the Waconichi Formation.

The anorthosite zone is dominantly composed of anorthosite layers, with minor gabbro, anorthositic gabbro and gabbroic anorthosite. Gabbro and magnetite bearing gabbro increase within the top 150 m of the zone near the contact with the layered zone. The iron (magnetite) as well as the vanadium and titanium content increases accordingly.

The layered zone is the main host of the VTM Deposit. It is a rhythmic layered sequence dominated by pyroxenite and anorthosite beds. The thickness of the layered sequence ranges from 150 to 900 m and the thickness of individual beds ranges from a few centimetres to a few metres.

The Granophyre Zone of the LDC has been identified by previous regional mapping. It can be traced all along the southeastern boundary of the main block outside the limit of the proposed Armitage and Southwest Pits.

Within the LDC, the VTM mineralization occurs at the top of the anorthosite zone and the base of the Layered zone. Allard (1967) has previously defined three main VTM mineralized horizons called P1-P2-P3. Girard and Allard (1998) have partly revised this stratigraphy and have implemented a P0 unit stratigraphically below the former P1.

The layered sequence has been subdivided into various units, separated by discontinuous anorthosite screens. It is noted, from the base to the top of the mineralization:

- P0: Anorthosite sequence interspersed by thin layers of magnetite (10-15% magnetite);
- P1: Dominant anorthositic sequence interspersed with numerous layers of magnetite of metric thickness (15-40% magnetite). This sequence is considered a low-grade magnetite source but one that is of high-grade vanadium. This resource is probably economic.
- P2: Sequence dominated by layers of magnetite and magnetite ferrogabbro (>40% magnetite) with a high-grade of vanadium. This sequence constitutes the major part of the vanadiferous magnetite deposit;
- P3: Magnetite and ilmenite ferrogabbro sequence (20-40% magnetite).

Generally, the three main mineralized units, P1 to P3, are separated in the field by rocks carrying no, or very low amounts of magnetite. The magnetite is usually well layered with sharp contacts, but some is disseminated in the fringed host rocks. The limits for each unit are seldom clear but the gangue rock is distinctive.

Magnetite and ilmenite are the main iron and titanium economic minerals. As described by Allard (1967), magnetite mineralization occurs as an alternation of layers of solid titaniferous magnetite, magnetite rich gabbro, magnetite rich pyroxenite, gabbro and anorthositic gabbro. The solid magnetite layers range from centimetres to over a metre in thickness. The magnetite rich band is everywhere at the same stratigraphic horizon, but each magnetite layer is discontinuous and exhibits marked changes in thickness and character along strike.

As expressed by the ground magnetometer survey and the recent aeromagnetic survey These horizons extend, almost continuously, for 20 km in length.

7.3. Layered Mafic Intrusions

Layered Mafic Intrusions (LMI) are one of the most complex and well-studied geologic systems. This has been driven in part by economic interests seeking predictive models for locating valuable mineralization that can range from sulphides (PGMs, Ni, Co and Cu) to oxides (Cr, Fe, Ti and V). These deposits have also been exploited for use in ceramics, refractories, or in cases of high Anorthite (An) plagioclase like Lac Doré, for their aluminum resources.

The primary deposition of magnetite and ilmenite as ore-bearing minerals in these bodies is termed orthomagmatic; they represent a direct crystallization and deposition from the parent magma. Some of the processes responsible for this are directly analogous to sedimentary processes. Figure 7-3 shows a cross-section through a well-studied example of an LMI from the Northwest Territories of Canada. It is shown here as an illustration of a similarly sized body that is fully exposed and very well documented.

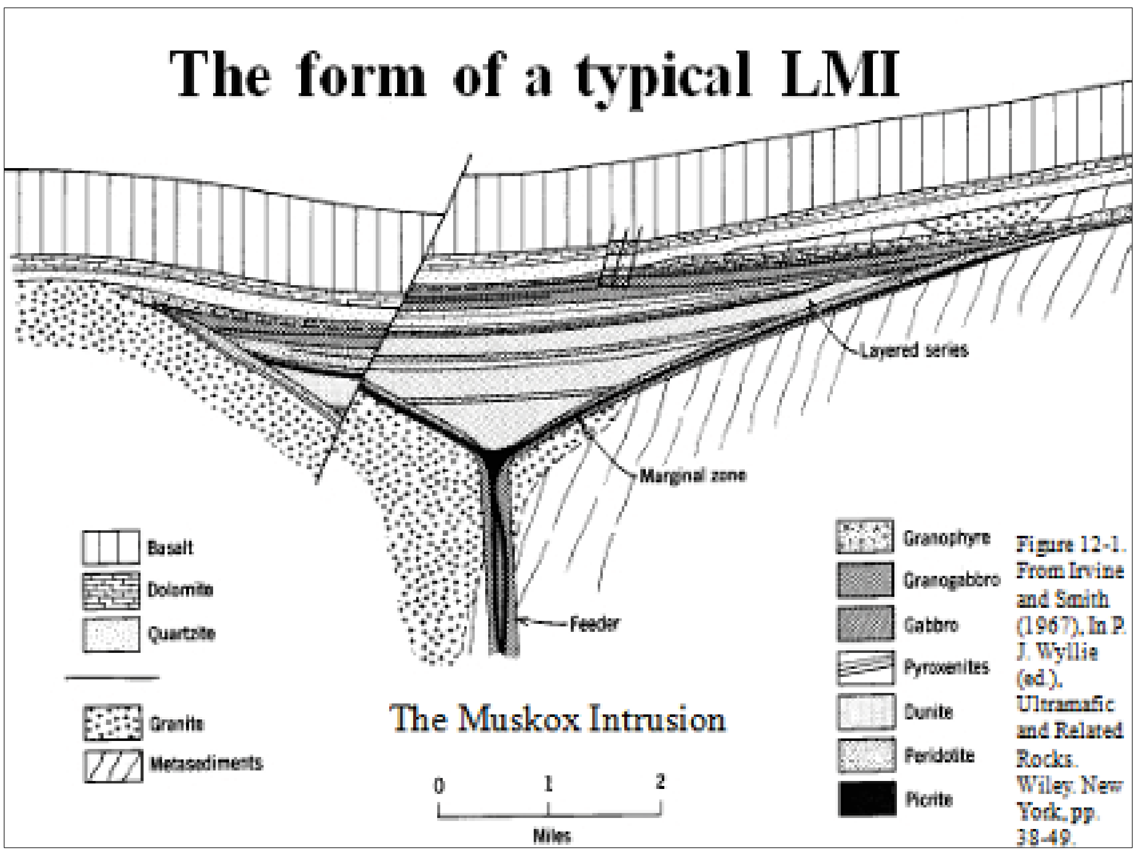


Figure 7-3: Cross-section of the Muskox Intrusion in the Northwest Territories of Canada

Note the basinal geometry and the predictable evolution of the composition in the layering. This example shows strong layering typical of these bodies and a geometry that is directly analogous to that seen in sedimentary basins, and in chemical sedimentary evaporate sequences in particular. Geological continuity can be extreme in these systems, with individual beds (time or event horizons) traceable over hundreds of kilometres in some cases.

Geochemistry has been demonstrated to work extremely well in fingerprinting and providing a record of magmatic conditions in these closed to semi-closed systems. There are over two dozen triggering and depositional mechanisms that have been proposed to explain the finely detailed relationships that have been deciphered through systematic chemical investigations.

7.4. Geology of the BlackRock Project

Based on surface mapping and drill core logging, BlackRock geologists have been able to identify the complete stratigraphy of the LDC, except the border zone. The geology of Property is dominated by the anorthositic Lac Doré Complex (LDC), with minor late gabbro, diorite and felsic intrusions and late Proterozoic diabase dykes (Figure 7-4).

All units form an unfolded homoclinal sequence striking roughly from N30° to N50° and dipping between 45° to 80° to the southeast. Stratigraphic tops face to the southeast. Figure 7-4 shows the overall geology of the BlackRock Project. Figure 7-5 and Figure 7-6 are detailed maps of the Southwest and Armitage Deposits, respectively.

On surface exposures and in drill cores, the anorthosite zone is dominantly composed of anorthosite layers, with minor gabbro, anorthositic gabbro and gabbroic anorthosite. As expected, and verified by drilling, gabbro and magnetite bearing gabbro increase within the top 150 m of the zone near the contact with the layered zone. The magnetite as well as the vanadium and titanium content increases accordingly.

The layered zone, is the main host of the VTM Deposit and is well exposed in the area of the Southwest Deposit. It is a rhythmic layered sequence dominated by pyroxenite and anorthosite beds. The thickness of the layered sequence ranges from 150 m to 900 m and the thickness of individual beds ranges from a few centimetres to a few metres.

The granophyre zone of the LDC has been identified by previous regional mapping. It can be traced all along the southeastern boundary of the main block outside the limit of the proposed Armitage and Southwest Pits.

Locally, the rocks were plastically deformed during regional metamorphism. There is also some post-genetic brittle fracturing and faulting. As a consequence, the units may be fairly difficult to distinguish in mapping. Although the favourable stratigraphy is relatively well exposed in trenches in the Southwest Deposit area, it is not the case over the remainder of the Property. In fact, because of the poor exposure, it has been impossible to distinguish the various magnetite bearing units over most of the Property. In the southwestern half of the Property in the Armitage Lake area, the favourable stratigraphy is almost entirely covered by overburden or under water.

Map key is provided on Figure 7-5 and Figure 7-6 for reference.

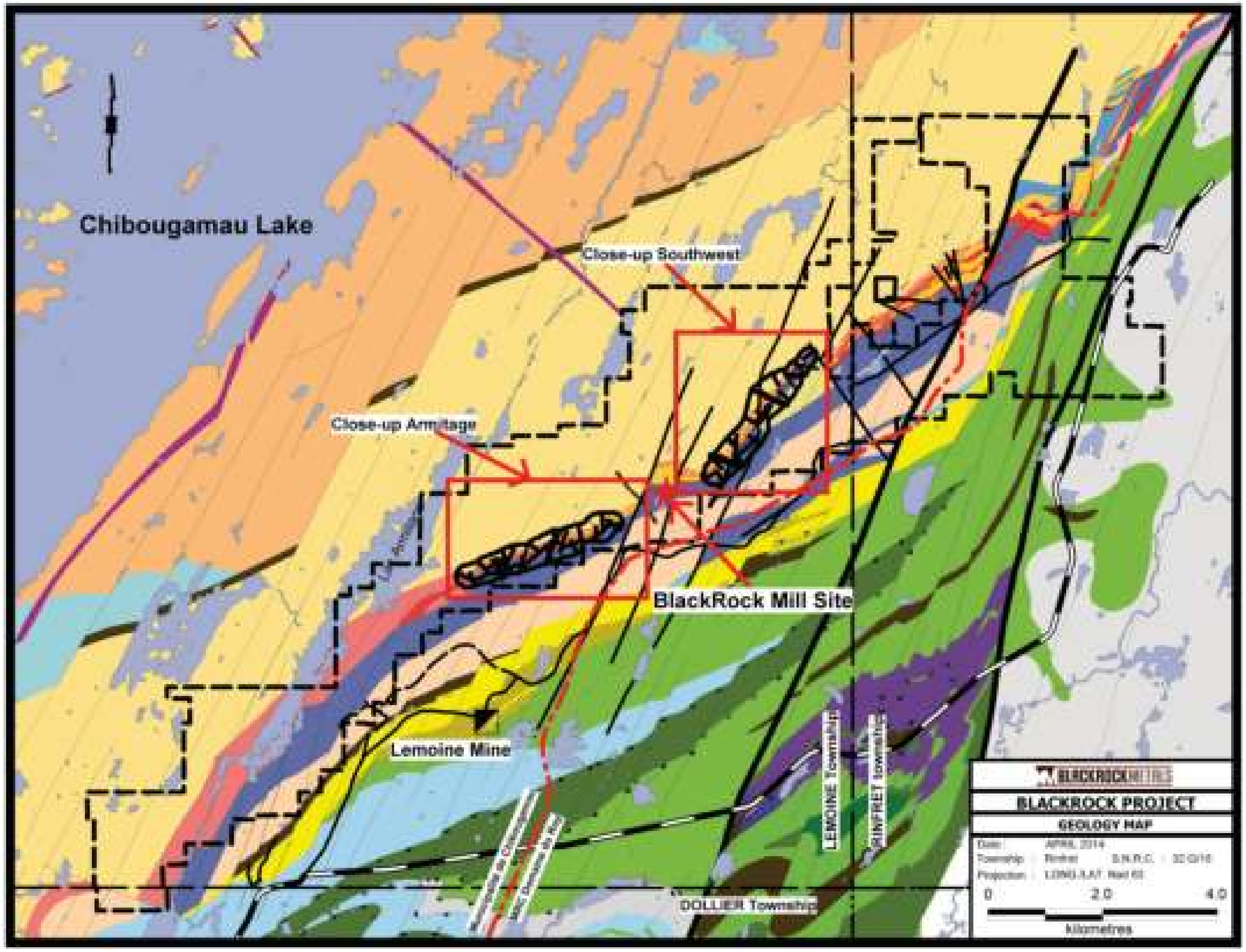


Figure 7-4: Geology of the BlackRock Property

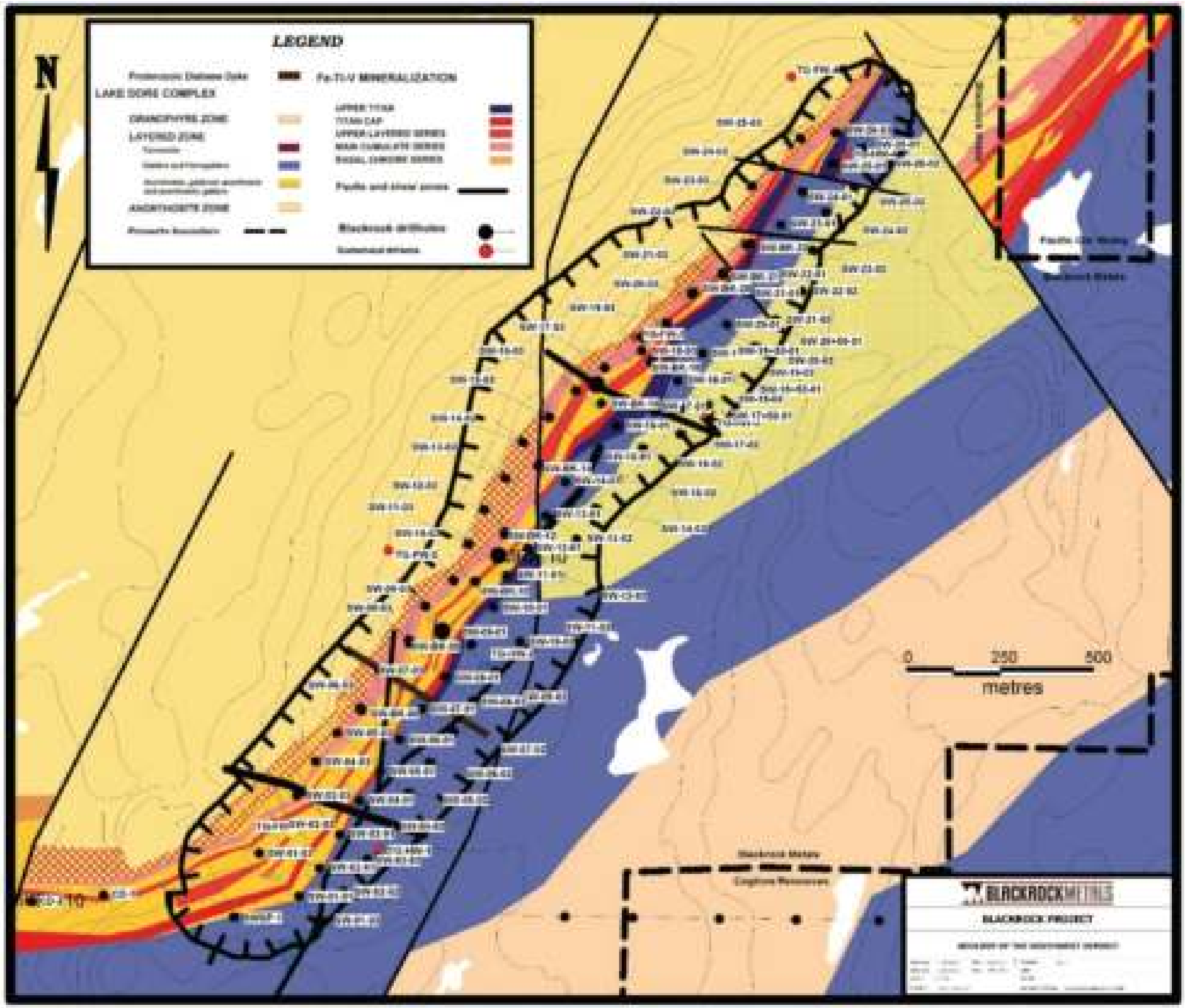


Figure 7-5: Geology of the Southwest Deposit

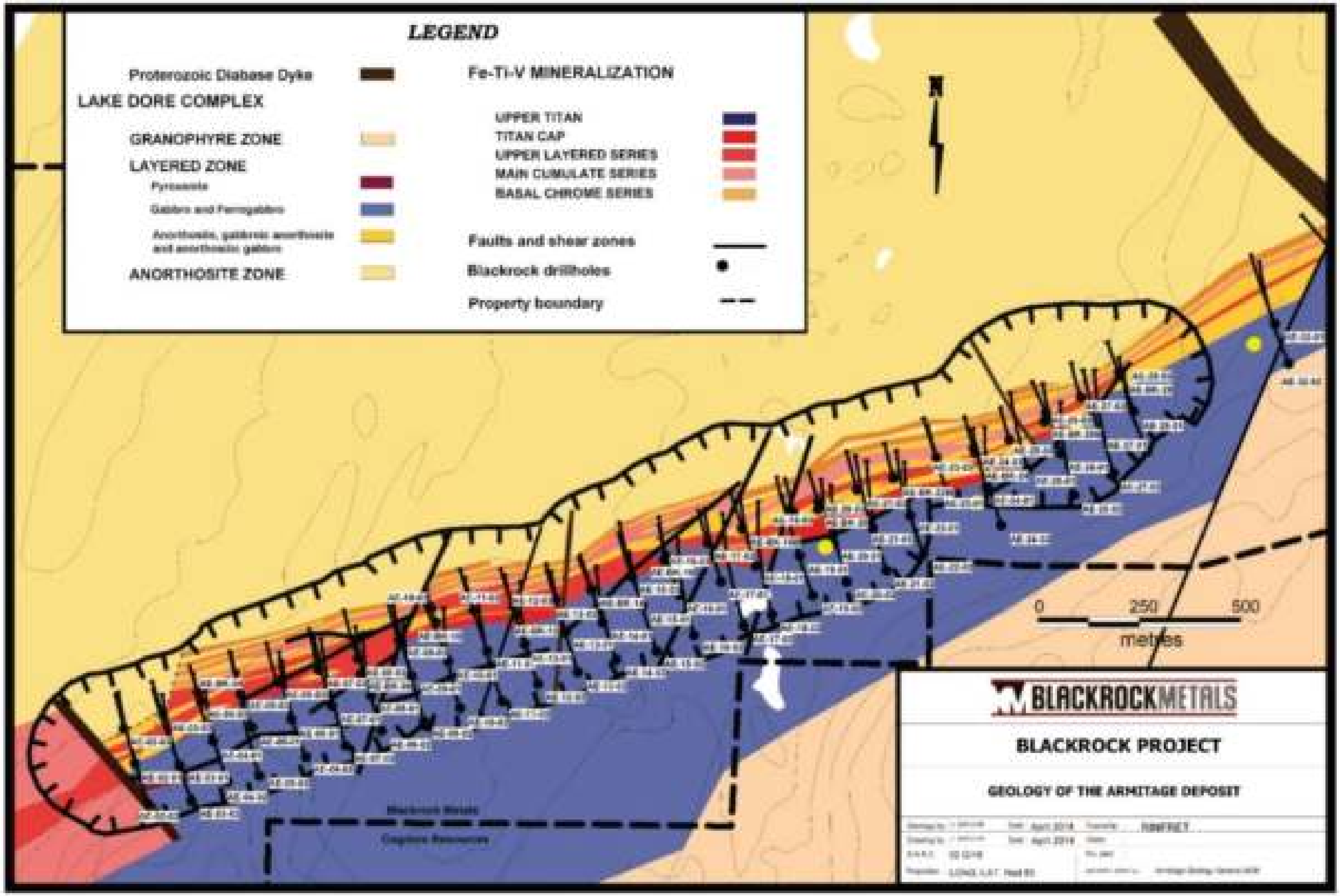


Figure 7-6: Geology of the Armitage Deposit

To the south, the situation is more complex. Directly to the south of the magnetite bearing units, we mostly observe gabbro and pyroxenite. The gabbro is the unit most closely related to the mineralization. A granophyre unit is observed in the southeast portion of the Property. The latter is coarse-grained and exhibits a very distinctive texture. It apparently extends all the way to the extreme south of the Property but more probably outside the Property. It is in contact with the felsic volcanic rocks of the Waconichi Formation hosting the Lemoine VMS mine to the southwest.

The rock exposure is similar for units to the north and south of the magnetite bearing stratigraphy. To the north, an anorthosite layer, locally interspersed with gabbro or anorthositic gabbro is observed. This unit extends to the northern limit of BlackRock's claims.

The reconnaissance mapping revealed the presence of rusty horizons within the Waconichi Formation. It was not established if the latter are related to the Lemoine horizon. More mapping would be necessary to establish that relationship. Given the frequency of NNE striking faults in the region though (close to the Grenville Front), correlations could be difficult to make.

7.4.1. Structural Geology

The proximity of the Grenville front produced late brittle faulted blocks that are well outlined on BlackRock's recent magnetic airborne survey. These faults generally run parallel to the Grenville Front in a north-northeast direction. There are also a number of northwest trending structures (sub-parallel to the drilling) that have been observed at the surface and in recent drilling. The Southwest Deposit is defined by two such structures, which separate it from the Armitage to the southwest, and the East and West Deposits (located outside the BlackRock claims) to the northeast.

Overall, all the rocks underlying the BlackRock project are massive, competent with minor shearing and fracturing. Bedding and layering are better defined within the layered zone. Geological interpretation and correlation allowed BlackRock to define three main orientations of shearing and faulting.

First, within ferrogabbro and pyroxenite, previous regional mapping and core logging have clearly demonstrated the occurrence of internal ductile shearing, and probably boudinage, sub-parallel to layering and foliation.

Second, NE-NNE and NW-WNW subvertical ductile shear zones and brittle faults have been interpreted on the project. The NE-NNE family is spatially related to the Grenville Front. Some of these structures are well defined on the airborne magnetometer survey and correspond closely to already known regional shear zones.

These NNE-SSW and WNW-ESE structures are characterized by a strong shearing and/or fracturing, usually with a strong to intense chlorite and/or sericite and/or carbonate alteration. Occasionally, quartz and quartz-carbonate veins with minor pyrite-chalcopyrite-pyrrhotite mineralization have been observed. Some of these faults and shear zones are filled by late gabbroic to felsic dykes and diabase. Within the proposed Armitage and Southwest Pits, horizontal displacements range from 5 to 20 m. More important shear zones mark the northeastern and southeastern limits of the Southwest zone.

Third, a major reverse NE-SW shear zone dipping between 45° to 65° to the northwest has been traced from drilling sections 100 to 900 in the southwest zone. Although this shear zone cuts the VTM mineralization at depth, the sense and extent of the displacement are not well defined.

A description of the structural features of the mineralization is detailed in Section 7.8.

7.5. Mineralogical Rock Composition and Chemistry

The deposit is notably fresh with respect to its sister deposits outside of Canada. There is little surface weathering. However, the gabbros and ferrogabbros associated with the magnetite-titanium layering have been affected by low-grade greenschist metamorphism over the entire Property explored to date. They are more properly meta-gabbros and meta-ferrogabbros, but they are referred to throughout this report by the protolith name.

The chemical mass balance of the rocks demonstrates the disequilibrium assemblage that has resulted from the cumulate fractionation process. The crystallization products have been segregated from the residual melt, and the resulting bulk chemistry represents a non-equilibrium grouping.

Table 7-1: Composite Average of all WRA for the Southwest and Armitage Deposits

Major Elements						
SiO_2	Fe_2O_3	Al_2O_3	CaO	TiO_2	MgO	Na_2O
33.73	28.67	15.17	7.71	5.05	3.7	1.91
Minor Elements				**Trace Elements**		
V_2O_5	S	MnO	K_2O	Cr_2O_3	P_2O_5	SrO
0.26	0.21	0.2	0.2	0.04	0.04	0.02

Note the non-equilibrium elemental proportions that have resulted from the gravity separation of the oxide phases and sulphide phases into cumulate layers that were isolated from the residual magma.

The elemental assemblage is dominated by silica, iron, aluminum and calcium. As a result, the deposit is mineralogically quite simple. There may have been a more complex mineralogy at the time of formation, but regional Greenschist alteration has annealed and homogenized to a limited number of silicate phases that accommodate variable ratios of metal cations in continuous solid solutions.

7.5.1. Gangue Mineralogy

Main gangue constituents are metamorphic minerals including chlorite, actinolite/grunerite and saussuritized anorthite. There are also remnants of the original mineralogy that probably consisted of orthopyroxene and clynopyroxene, anorthite plagioclase and minor amphibole.

7.5.2. Ore Mineralogy

Ore phase mineralogy is a very simple binary system consisting of titano-magnetite and ilmenite with minor magnetite whose relative ratios vary systematically with the cycling depositional mechanisms at the scale of the rhythmic layering, and also within the stratigraphic sequence as a whole on the scale of the modal layering.

7.5.3. Minor and Trace Phases

Minor phases that are important are epidote and base metal and iron sulphides. These phase have to be cleaned from the final concentrates to meet market specifications.

Trace amounts of rutile, titanite (sphene), albite, hematite and calcite have been noted from surface samples and in the core of shear zones where deformation and comminution of grains is highest.

7.5.4. Surface Oxidation

Since the main ore minerals are primary magmatic oxides, there is little opportunity for the deposit to be affected significantly by additional oxidation, with the exception of the magnetite potentially being transformed to hematite. All mineralogical studies to date suggest that there is no significant hematite alteration or other oxidation associated with surface weathering or supergene processes. The last glacial period is thought to have mechanically removed any near-surface oxidation that may have formed and weakened the resulting rock coherence prior to the glaciation event.

7.5.5. Metamorphism

Rocks of the Lac Doré Complex have been metamorphosed to the greenschist facies. It is generally an isochemical process and true alteration is limited. The original chemistry was primitive with low alkali metal content, and dominated by the elements that originally crystalized as anorthite, pyroxene/amphibole and oxide minerals. These have been homogenized by metamorphic reactions into the present chlorite-amphibole mineralogy (see Section 7.5.1).

7.5.6. Late-Stage Alteration

There is evidence from the work of Girard (2001) for an overprinted alteration reaction near structures, and to a lesser degree near the surface, that has preferentially attacked the titano-magnetite to form Fe-chlorite. The ilmenite phases appear to be refractory to these alteration events (see Section 9.6 of this report).

7.6. Types of Layering

Layering, or stratification, occurs in several forms within a typical LMI. A layer is defined as any sheet-like cumulate unit distinguished by its compositional and/or textural features. The mineralized layers at Lac Doré are characterized by homogeneous textures and uniform mineralogy throughout the mineralized sequence (Figure 7-7). There are many types of layering that have been identified in these systems, three of which are of primary importance to BlackRock's stratigraphic work at Lac Doré:

1. Rhythmic layering;
2. Modal layering;
3. Cryptic layering.

Figure 7-7: Examples of typical layering present at Lac Doré

The rhythmic nature of the modal layering is clearly seen in these photos. Cryptic layering can only be examined through chemical analysis and is not visually obvious.

The entire layered sequence displays various scales of rhythmic layering where the adcumulate, mesocumulate and orthocumulate magnetite layers alternate with less mineralized or barren anorthositic or gabbroic compositions. Although cumulate phases are the most visually obvious characteristic of the sequence, they are also one of the least useful in terms of defining the stratigraphic sequence. In practice the interlayered rocks within the mineralized sequence do not have strong identifying markers that define a fine-scale positioning.

Of greater utility is the presence of large-scale modal layering that separates the major lithological types within the ore sequence. This is characterized by variation in the relative proportions of constituent minerals in a layered sequence, and manifests as large-scale groupings of interlayer types. In the base of the section, anorthosite is the dominant interlayer with the magnetite cumulates. In the mid-section, this switches to gabbroic compositions and, in the upper portions there is a distinct break into pyroxenites as the interlayers. Magnetite and ilmenite cumulate phases are present throughout the section.

Finally, the single most important tool for defining stratigraphy at LDC is the presence of a well-developed cryptic layering (not obvious to the eye). Cryptic layering is characterized by the systematic variation in the chemical composition of certain minerals with stratigraphic height in a layered sequence. Figure 7-8 illustrates cryptic layering along the right-hand side of the stratigraphic columns for both the Bell River, as well as the Bushveld in terms of crystal chemistry and elemental ratios. With the availability of closed-system chemical evolution tied to fractionation a similar story has also emerged for the LDC. This has formed the basis for BlackRock's stratigraphic interpretation that will be discussed in detail in Section 7.8 of this report.

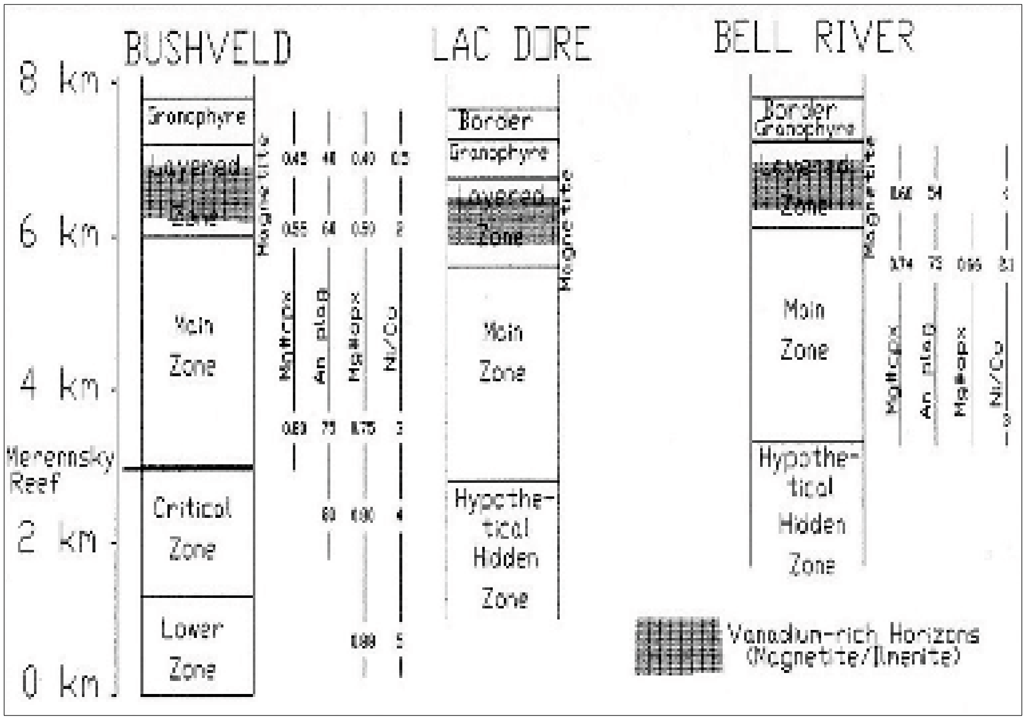


Figure 7-8: The Lac Doré Complex in comparison to the Bushveld and Bell River Systems

The gross layering is identical in thickness and order for the upper portions of each complex. Note the fractionation trends illustrated for Bushveld and Bell River are typical of this type of orthomagmatic system. Similar trends were the basis for BlackRock's stratigraphic interpretation of the Lac Doré sequence.

7.7. Stratigraphy of the Lac Doré Complex and the Layered Zone

BlackRock has adapted the stratigraphic model from Allard and Girard (1998) to the detailed dataset that it has generated over its deposits. Defining a local stratigraphy requires unique marker horizons that allow determination of the time-sequence of events (depositional history) and defining the timing of deposition. Knowing the "tops" and the order of deposition provides directionality and allows analysis of any perturbation to the original sequence. It can help to distinguish between brittle and ductile deformation styles, relative motion within the deposit, and pre-mineral versus post-mineral relationships. This in turn guides development of models regarding triggers and causes for depositing ore minerals. Used in concert, these windows into the primary deposition and the post mineral dislocation provide the basis of BlackRock's predictive models for exploration and development.

A generalized stratigraphic column is presented in Figure 7-9. Stratigraphic criteria for making specific unit assignments to composite intervals are outlined and the correlation to with the Allard and Girard (1998) stratigraphic nomenclature is provided. There is good general agreement between the two schemes as they are both based largely on natural breaks in lithology that are mappable on the surface and subsurface over the 12 km of the LDC that has been drilled to date by McKenzie Bay and BlackRock. For most of this report, the two members of Titan are not differentiated.

The first unit break is called the Basal Chrome Series (BCS) and consists of a section of the top of Allard's Anorthosite Zone that displays interbedded of magnetite cumulate with increasing frequency moving upwards. The break between the top of this unit and the beginning of the Middle Cumulate Horizon (MC or MCS) is marked by the appearance of three or more continuous composites with Satmagan values >10%. The MC marks the beginning of the Layered Zone, which is the main ore host in the LDC. The MC passes into the Upper Layered Series (ULS), which is marked by an interlayering of anorthositic gabbro to ferrogabbro with the magnetite cumulate beds. The ULS ends and the Titaniferous Capping Series (Titan) begins with the appearance of pyroxenite as the dominant interlayered material with the magnetite cumulate rocks that progressively fade passing into the upper Titan.

	Girard & Allard Stratigraphy	BLACKROCK	Dominant Lithology	Comments	Maximum Mineralized Thickness (m)
GRANOPHYRE ZONE			Soda Granophyre	High Iron contain but in silicate	
LAYERED ZONE	P3		Ferropyroxenite	Quartz Bearing	
			Ferropyroxenite	10-20% apatite	
		UPPER TITAN	Ferropyroxenite	10-20% ilmenite	
		TITAN CAP	Ferropyroxenite	Basal portion with magnetitite bands	55
	A2		Gabbroic Anorthosite	Minor Qtz ; 1 cm feldspar	
	P2	ULS	Ferrogabbro	Upper : Pyroxenite and magnetitite	60
				Lower : Gabbro anorthositic gabbro	
	A1		Gabbroic Anorthosite		
	P1	MCS	Magnetitite		45
ANORTHOSITE ZONE	P0	BCS	Anorthositisit Gabbro	At the top, 150 meters with magnetitite and magnetite rich layers	75
			Anorthosite		

Figure 7-9: BlackRock stratigraphy based on surface mapping and drilling

Thickness ranges are based on true thickness from model results.

7.8. Structural Geology of the Armitage and Southwest Deposits

Differentiation between brittle and ductile deformation styles on the Property is made difficult due to a lack of exposure and relatively coarse resolution in the data, but is the central question in approaching the overall framework for the geological model. Both styles are in evidence.

7.8.1. Geologic Evidence

Regional and local scale mapping on the project has been limited by lack of exposure. Trenching has augmented the natural exposures to a degree, but the trenches are limited in number and extent. The best continuity and coverage of lithological data is provided by diamond drilling on nominal 100 m x 100 m spacing, and logging of drill sections has provided the strongest tool to interpret the rocks in the subsurface directly.

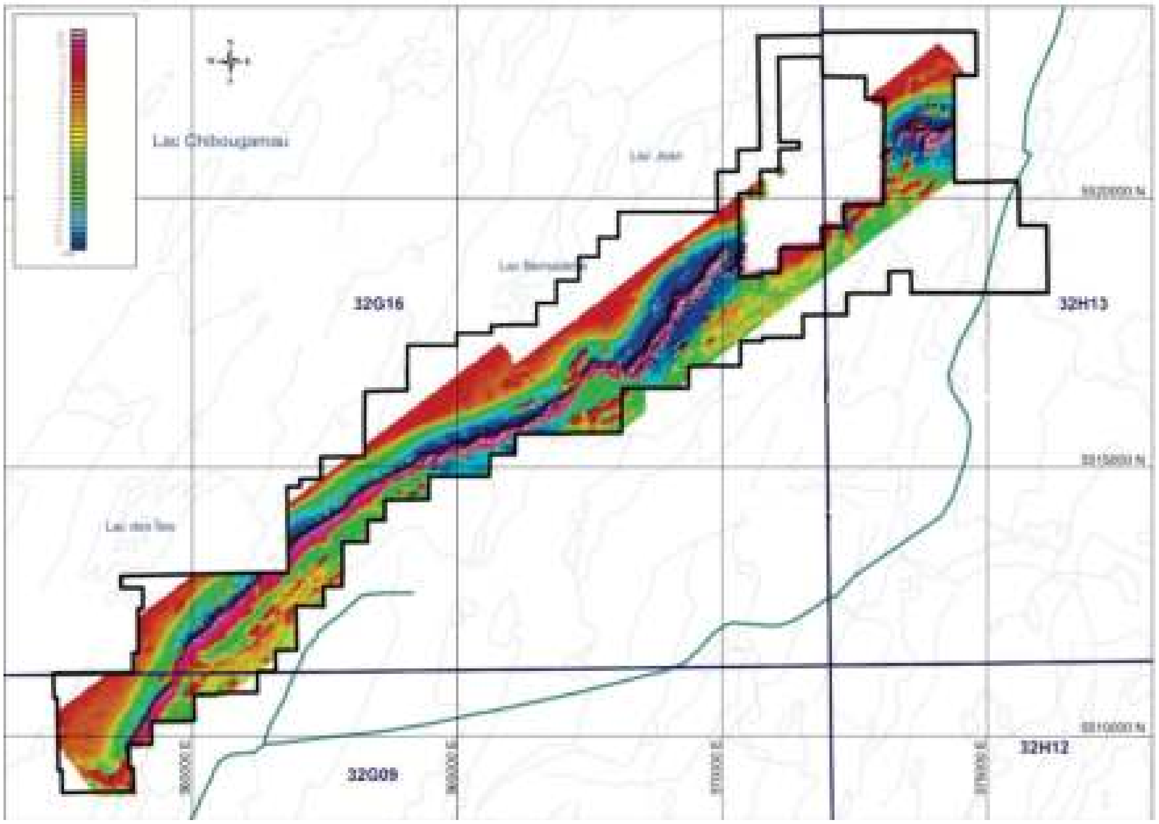
The large-scale context puts the Property on the steeply dipping southern limb of an anticline that has warped the LDC. As noted above, the sedimentary nature of deposition in the LDC also lends itself to ductile features developing in the crystal mush prior to complete lithification (crystallization of the melt). Other factors affecting the package are its deposition near the brittle/ductile transition, and having had a long time (>2.7 Ga) for deformation to occur.

7.8.2. Geophysical Evidence

Magnetite is the most abundant economic oxide mineral at Lac Doré, and BlackRock's VTM mineralization has a strong continuous magnetic signature as a result. This signature can be traced over the entire length of the Property. There are several small breaks in this signature that indicate the limits of each of the various sub-deposits within the overall 20-24km trend.

Ground magnetics provides a much more detailed picture of what his happening on the scale of 100's of metres. Figure 7-10 reflects this finer scale of detail, in this case in the Armitage Deposit, where many of the breaks in this map are coincident with proposed structures or domain bounding faults.

Geophysical methods provide consistent empirical coverage and indications of large and local-scale brittle dislocations. While the 100m spaced ground magnetic survey proved useful for local structural mapping in the Armitage, his data worked less well in the Southwest due to leveling issues in the datasets. However, coupled with the airborne magnetics and hyperspectral data from the core, they provide strong evidence for brittle faulting with offsets that measure 10s to 100s of metres.



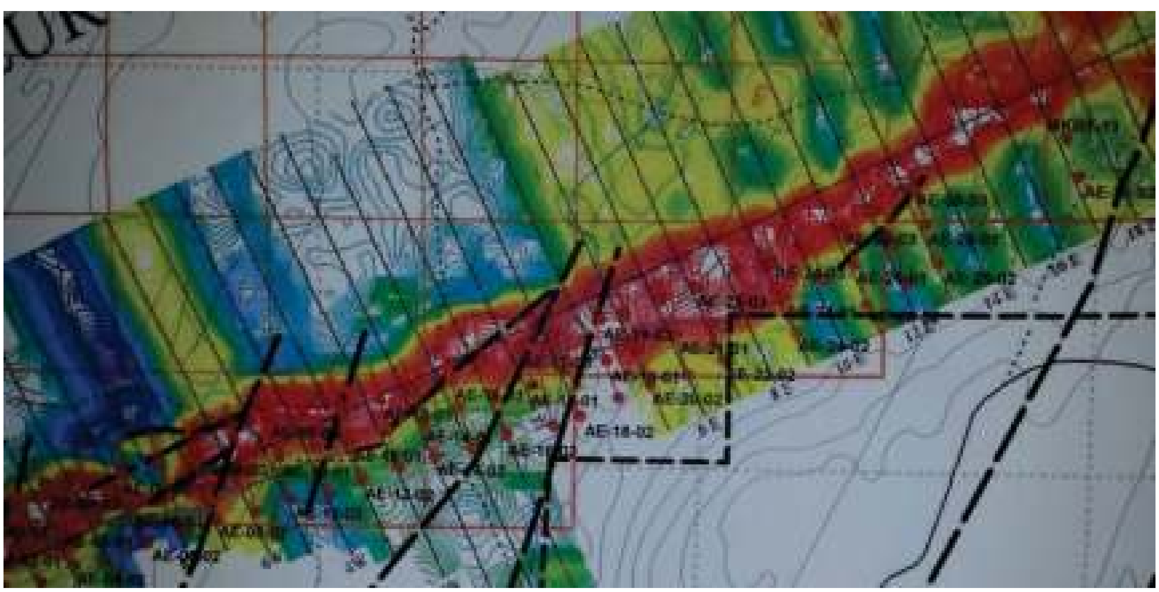


Figure 7-10: Upper map shows the aeromagnetic mat of the trend controlled by BlackRock in 2017 (in black)

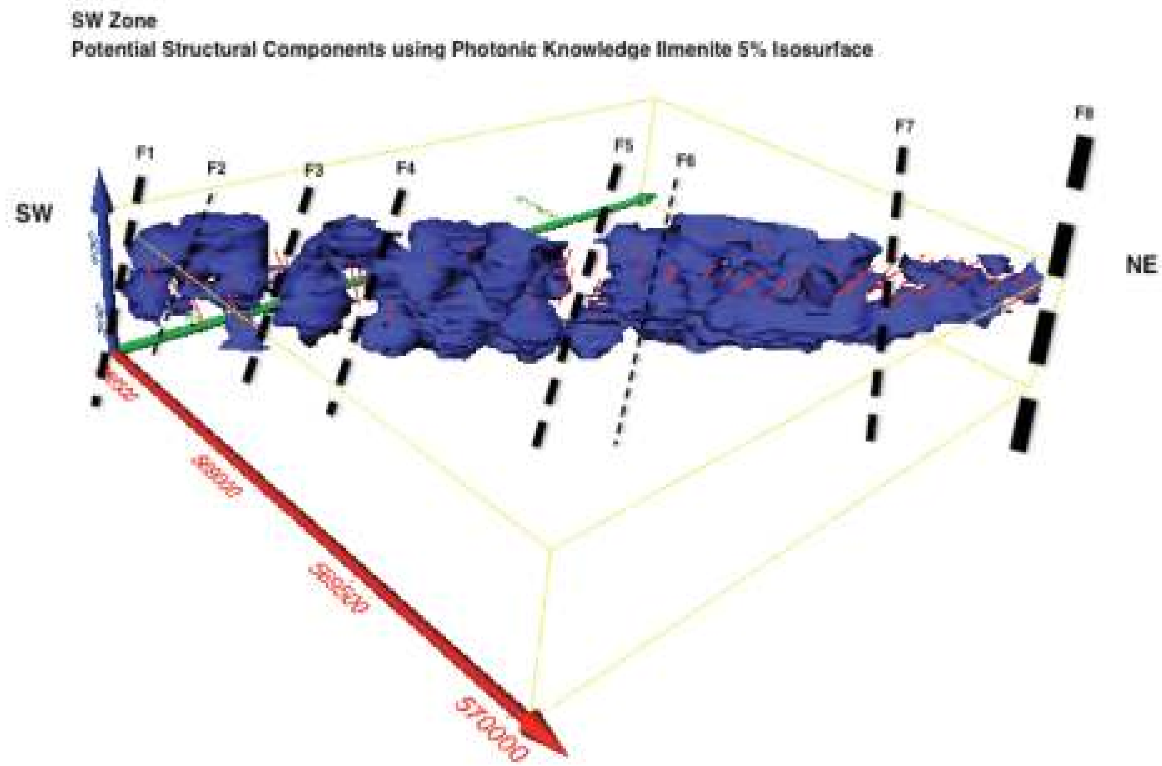
The “hole” in the data represents ground controlled by a third party. The lower image shows a provides detail over a portion of the ground magnetic map of the Armitage Deposit.

Many of the breaks in this map are coincident with proposed structures or domain bounding faults.

Geophysical methods provide consistent empirical coverage and indications of local-scale brittle dislocations. A 100 m spaced ground magnetic survey proved most useful for local structural mapping in the Armitage. This data worked less well in the Southwest due to leveling issues in the datasets. However, coupled with the airborne magnetics and hyperspectral data from the core, they provide strong evidence for brittle faulting with offsets that measure 10s to 100s of metres.

The results for the McKenzie Bay ground magnetic surveys over Armitage are shown in Figure 7-10 above. There are several dislocations that are easily seen despite the poor data leveling in these data. The line spacing is wide at 100 m, but the proximity to the ground provides greater detail than is seen in a comparable air magnetic survey flown on 50 m spacing over the same area. This map became available after the model for the Southwest Deposit was completed, but was one of the primary tools used to place a structural framework onto Armitage. It was used in concert with mapping, logging, geochemistry and hyperspectral imaging to define the location of the major structures that have affected the mineralization.

Figure 7-11 shows spatial distribution for one of the PK hyperspectral mineral assemblages for illustration purposes. The proposed structures are based on variance in the relative concentrations of the minerals within the mineral assemblage represented in the “Ilmenite” library. The traces of these breaks are resolved by empirically adjusting the threshold limits to optimize the image.



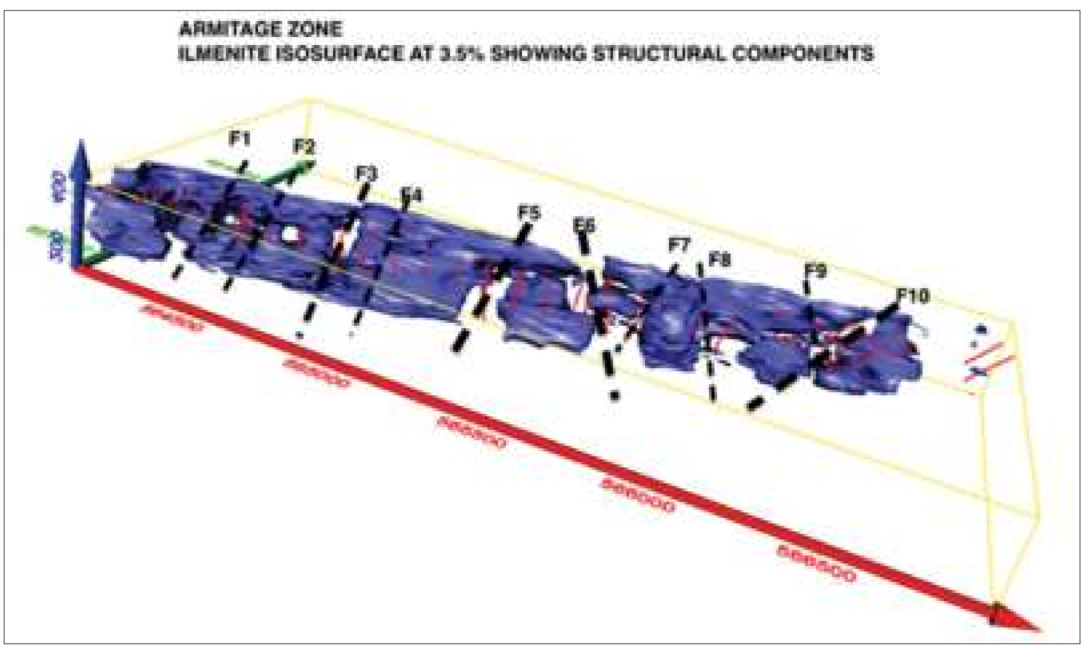


Figure 7-11: Hyperspectral images of the Southwest (top) and Armitage (bottom) Deposits

The hyperspectral data shows gaps formed where the ratios of the minerals in the assemblage displayed have been visualized by empirically adjusting the thresholding to achieve the image above. A similar image is seen for the Southwest Deposit.

It is thought that the hyperspectral signature of an originally uniform distribution has been shifted as a result of structural features that allowed fluids to penetrate the package (these can include CO_2 and H_2O, etc.). These changes are interpreted to be isochemical; there is no clear evidence for introduced metals or other elements in the resulting rocks, but the size and modal distribution of the minerals present have been affected.

7.8.3. Geochemical Evidence

Geochemical methods confirm that the dominant deformation style on the deposit scale is brittle faulting. An inflection in the slope of these elemental ratios could indicate recumbent folding or overturned beds. None was detected. The stratigraphic units in all drill panels in both deposits appear to be planer and not folded to any significant degree.

The timing of the brittle faulting has also been examined. Local variation in thickness and grade of the cumulate beds could be caused by offset along magmatic feeder structures or may have formed physical traps along the floor of the magma chamber. One example of such an anomaly could be the near doubling of the aggregate thickness of the MC+ULS in Armitage below drill panel 1100.

However, the majority of the faulting identified in this study appears post mineral in nature.

7.8.4. Interpretative Summary

BlackRock has used all three sources of evidence (geology, geophysics and geochemistry) to infer the location and orientation of significant structures that affect the mineralized stratigraphy. Summary tables of these structures are provided in Table 7-2 for Southwest and Table 7-3 for Armitage. Conclusions of the model interpretation are as follows:

1. Lac Doré is on the southern limb of a broad (20 km) anticline whose axis is sub-parallel to the main direction of convergence/accretion in the pre-Cambrian.
2. Brittle post-mineral faulting is the main deformation style on the scale of the deposits and mine planning.
3. Only major domain bounding faults were imposed for modelling and resource calculation.
4. The absolute location of each of the faults is limited by exposure, geophysical data quality and drill spacing. They are generally located to ±20 m in most cases.

5. They rarely have a strong indication of dip, and most structures have been modeled as vertical unless otherwise controlled.
6. The azimuths are constrained by drill control and are good to about ±30 degrees. Most faults are defined by one known point; it takes three to fully define a plane.

The absolute location and attitudes of the faults are not considered material for resource modelling; the volumes represented will remain essentially identical even if the attitudes are reinterpreted. The location and attitude do become important for short term mine planning, and pre-stripping of top soil prior to initiation of mining will produce the resolution of exposure required to constrain these features more precisely.

Table 7-2: Southwest domain summary

Domain Name	Block Model Domain	Approximate Model Strike	Accepted Model Dip and Structural Domain Segments	X-Section Dip	Drill Panel	Interpretation Support	Throw, Type
			SW End			PK, Map/Sect	
D1	1	40	45	45	100		
D1	1	40	45	45	200		
D1	1	40	45	45	300		
D1	1	40	45	45	400		
		315	DB-1			PK, Dip	
		215	Southern Bounding			Map/Sect, 3-D	+100 m ?, REV possible
D2	2	40	70	70	500		
D2	2	40	70	70	600		
		315	DB-2			PK, Dip	
D3	3	40	55	55	700		
D3	3	40	55	55	800		
D3	3	40	55	55	900		
D3	3	40	55	58	1000		
		315	DB-3			PK, Dip	
D4	4	40	72	70	1100		
D4	4	40	72	70	1200		
D4	4	40	72	75	1300		
D4		N-S	DB-4			Map/Sect	
D4, D5R	5	40	72	70	1400		
D4, D5L, D5R	5	40	72	74	1500		
			DB-5			Map/Sect	
D5L, D5in, D5R	5	40	72	72	1600		
D5in, D5R	5	40	72	75	1700		
		275	DB-6			Map/Sect, 3-D, Dip	100 m, DEXTRAL SLIP
D6	6	40	83	83	1800		
D6	6	40	83	81	1900		
D6	6	40	83	83	2000		
		315	OFFSET			PK, Map/Sect, 3-D	20 m DEXTRAL SLIP
D6	6	40	83	83	2100		
D6	6	40	83	86	2200		
		275	DB-7			Map/Sect, Dip	50 m SINESTRAL SLIP
D7	7	40	70	75	2300		
D7	7	40	70	70	2400		
D7	7	40	70	67	2500		
D7	7	40	70	80	2600		
			NE End			PK, Map/Sect	

Mag = Ground Magnetic break (Dave Caldwell Geophysisist)
PK = Interpreted PK break (Mike Allen Geologist)
Dip = Model Dip (Domain) change,implication from model (Dave Caldwell Geologist)
Map/Sect = Proposed structure from cross section,bench and surface geology (Daniel Bernard, Charles Perry Geologists)
3-D = Proposed Structure to solve geometric issue (Dave Caldwell Geologist)

Table 7-3: Armitage domain summary

Domain Name	Block Model Domain	Approximate Model Strike	Accepted Model Dip and Structural Domain Segments	X-Section Dip	Drill Panel	Interpretation Support	Throw, Type
				-90	Open Projection		200 meter projection of Section 200 polygons to west
D1	1	73	80	80	200		
D1	1	73	80	78	300		
DB1		130		-90		Mag, Dip, Map/Sect, 3-D	
Clipper		80		80 to S	200 & 300	Map/Sect, 3-D	
D2	2	68	60	60	300		
D2	2	68	60	60	400		
D2	2	68	60	60	500		
F1		180		-90		Mag, Dip, 3-D	
D3	3	70	70	70	500		
D3	3	70	70	70	600		
D3	3	70	70	70	700		
D3	3	70	70	67	800		
F2		142		-90		Mag, PK, 3-D	
D4	4	76	70	70	900		
F3		162		-90		Mag, PK, 3-D	
D5	5	87	70	65	1000		
D5	5	87	70	66	1100		
D5	5	87	70	66	1200		
D5	5	87	70	70	1300		
F4		150		-90		Mag, 3-D	
D6	6	71	70	70	1400		
F5		162		-90		Mag, PK, 3-D	
D7	7	68	70	65	1500		
D7	7	68	70	70	1600		
F6		124		-90		PK, 3-D	
D8	8	80	70	70	1700		
D8	8	80	70	75	1800		
D8	8	80	70	65/82	1900		
D8	8	80	70	65/82	2000		
F7		163		-90		Mag, Dip, PK, 3-D	
D9	9	67	70	65	2100		
D9	9	67	70	75	2200		
F8		164		-90		Mag, Dip, PK, 3-D	
D10	10	75	70	70	2300		
D10	10	75	70	70	2400		
D10	10 & 12	75	70	70	2500		
F9		155		80 to NE		Mag, Dip, PK, 3-D	
F10		213	70	80 to NW	2100 THROUGH 2800	PK, Map/Sect, 3-D	
D11	11 & 13	70	85	70	2500		
D11	11 & 13	70	85	80	2600		
D11	13	70	85	90	2700		
D11	13	70	85	90	2800		
Terminator				-90		Mag, 3-D	Projection of the 2800 polygons to mag break to the east

Mag = Ground Magnetic break (Dave Caldwell Geophysisist)
PK = Interpreted PK break (Mike Allen Geologist)
Dip = Model Dip (Domain) change,implication from model (Dave Caldwell Geologist)
Map/Sect = Proposed structure from cross section,bench and surface geology (Daniel Bernard, Charles Perry Geologists)
3-D = Proposed Structure to solve geometric issue (Dave Caldwell Geologist)

These data were provided to SGS as an outline for constraining the block model that was produced. As noted in the "Interpretation Support" column, there are two to four independent lines of evidence for each modeled structure or domain break.

7.9. Geological Modelling

Although fundamentally igneous in origin, the rocks and the depositional processes in the LDC are actually a variant of a sedimentary system that has been locally modified by discordant and concordant intrusive bodies. This conceptual frame forms the foundation for all of the interpretive work and the resulting wireframe geologic solids.

7.9.1. Assumptions

The following assumptions were utilized in construction of the deposit models for Southwest and Armitage:

1. The beds were assumed to be originally largely flat-lying and laterally extensive. None of the workers on the LDC since the mid-1960s have described the edges of the deposit. BlackRock has not seen any "edge effects" in the data and the top, bottom and sides of the chamber are assumed to be well away from the focus of the models.
2. LDC represents a closed chemical system, with a straightforward fractionation sequence being recorded in the rocks as they were laid down in stratigraphic order.
3. The flat lying rocks were uniformly affected by a broad regional anticline, and are now tilted nearly on-end forming and holding a topographic high in the Southwest deposit.
4. Despite its great age, it was assumed from the results of the air magnetic survey that the body is largely intact except for a limited number of major discontinuities that define the individual zones. In practice, offsets tend to be 20-50 m or less in the zones and 500->1000 m between zones.
5. Post mineral alteration is also closed system and manifests as isochemical metamorphic products.

This last point is one of the key pieces of understanding that has been confirmed from work to date, and shapes much of the thinking about the modelling of the deposit. The bulk WRA chemistry stays remarkably constant everywhere we currently have data, but the modal mineral distribution does shift significantly and this is essentially what is being mapped through the stratigraphic work, as well as our structural interpretations.

7.9.2. Modelling Methodology

The geologic model for this study was built using a fundamentally simple set of guidelines:

1. Modelling was done on drill panels that displayed TiO_2, PK data, Satmagan and bench geology down the drill spines.
2. Full assay printouts were printed and used for discriminating stratigraphic calls.
3. Stratigraphic unit polygons were drawn to composite boundaries. The geological calls from bench logging were found to be generally less than 50% reliable, but did prove useful in cases where the chemistry was ambiguous.
4. In a general sense, a natural 2% Sat cut-off for the lower end of the grade range was used; the elemental ratios were always checked to verify that they were essentially in line with the position in the fractionation sequence, or to understand why they might diverge.
5. To simplify the model, natural breaks were taken for stratigraphic unit boundaries whenever convenient. In a continuous fractionation sequence, the breakouts are somewhat arbitrary, but the location of the lithology breaks are thought to represent breaks in time or mode that formed zones of relative weakness that may have allowed preferential failure to occur.
6. On occasion, internal waste was allowed where it simplified the solid geometry. These undifferentiated intrusive phases generally have low-grades, and in most cases only single 3 m composites were allowed (very rarely two intervals taken).
7. Shapes were carried through from hole-to-hole down to single 3 m interval widths where the continuity appeared obvious. There are numerous examples of very narrow intervals carrying through 2-3 drill intercepts on single drill panels; similar continuity is seen when connecting drill panels into solids.
8. Intercepts that were present as 1-2 composite intervals but isolated (fault slivers, etc.), or otherwise outside of the well-defined stratigraphic units (ratios) were “orphaned” and left out of the final interpreted polygons. Many of these orphans have moderate to high-grades.
9. The MC is distinguished in Southwest Zone by 3+ continuous intervals of Sat% over 10%. In Armitage, this rule did not work due to a more even distribution of grades.
10. The ULS was distinguished by the first breakdown of the MC rule as we moved up the drillholes. In Armitage the relative position in ratio space was discriminator.
11. As a first rule, the simplest interpretation was preferred. For instance, folding is present at various scales and could be a player at the deposit scale, but this was ruled out based on cryptic fractionation layering, which greatly simplified the resulting models.
12. The simple straight-line interpretation left many of the contacts with one or two points of control.

13. To the extent possible, interpretations were driven from places that had three points of contact control into those that had less.
14. Natural breaks in grade were thought to represent potential contacts and were often used to define the limits of individual depositional units. These can be syn-depositional, or present as dykes or bedding conformable sills.
15. After the first pass modelling was completed, fractionation trends were plotted for each drillhole as a cross-check on the stratigraphic calls.

Evidence from plotting unclassed intervals on the elemental fractionation curves suggests that there are at least five or six distinct families of intrusive. Many of these are also visually distinct but have been inconsistently identified on the logging table. These chemical subgroups can be further defined and used in conjunction with reclogging to refine future geologic interpretations.

7.9.3. Results of the Southwest Deposit Model

Figure 7-12 presents a typical section through the Southwest Deposit to give a sense of what the stratigraphy looks like in this plane. Further detail and examples of the geologic interpretation as it relates to constraining the block models will be provided in Section 14 of this report. Generally speaking, the interpretation finds a regular and predictable layering of the sequence with very little structural disturbance evident on the cross-sections. All four mineralized stratigraphic beds are present on all drill panels with rare exception. Where there is a unit missing, it is either because the drill testing and assaying was incomplete, or interpreted to be faulted out of the plane. There were also instances of an ore zone repeating in a section. This was interpreted as being caused by faulting across the plane of the drill panel.

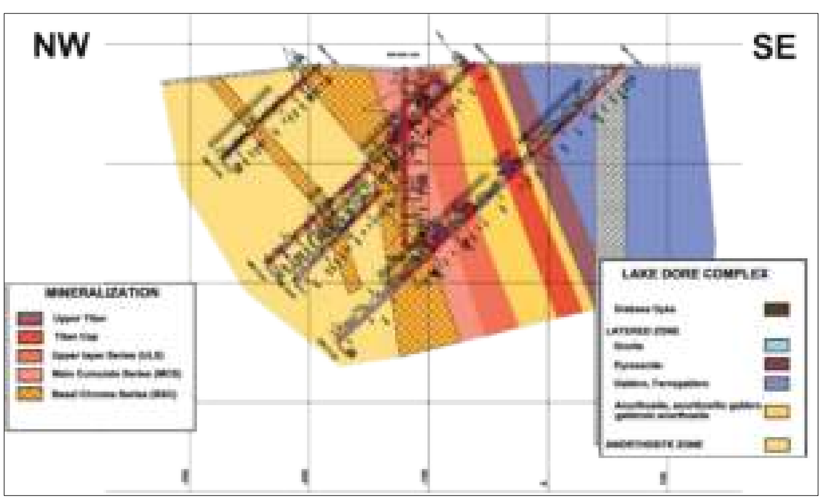


Figure 7-12: Typical section of the Southwest Deposit looking northeast

Note the mineralization forms a part of the Layered Zone, with the lower section (BCS to the left-hand side and progressively moving upward to the Titan (and upper Titan)). Histograms on the drill spines represent TiO_2, line plot portrays the PK results. Satmagan readings are shown by the text entries on the outside of the histograms that are too small to read in this view.

The next series of plots show the average Satmagan values by down hole composite position within the stratigraphic sequence for the northeast end of the Southwest (Figure 7-13). To prepare these plots the first down hole intercept for each unit in each hole was taken to represent the top of that stratigraphic unit and labeled composite #1. Each subsequent sample was sequentially numbered until the base of that unit was reached. The resulting data were then sorted by drillholes and then drill panels, and averages for each composite number in each stratigraphic unit tabulated and arranged in a continuous series. Interlayers of barren anorthosite were omitted from this analysis. When viewed vertically with the BCS at the base, this becomes a profile for oxide deposition over time that can be placed alongside the stratigraphic column.

Note that the relative ratios between Satmagan and TiO_2 are largest in the MC and ULS and taper down to 1:1 or less on either end. The number of composites in each unit varies significantly across both deposits. This has produced edge effects where the higher numbered composites tend to average progressively fewer values resulting in correspondingly high variance on the bottom edges of the units, and the core of the trends are smoothed and good representations of average values. Also note that this work confirms that the top contact of the Titan and the bottom contact of the BCS have not been fully drill tested or assayed for either deposit at this point. In contrast, the MC and ULS have assay defined top and bottom contacts in essentially all drill panels over both deposits.

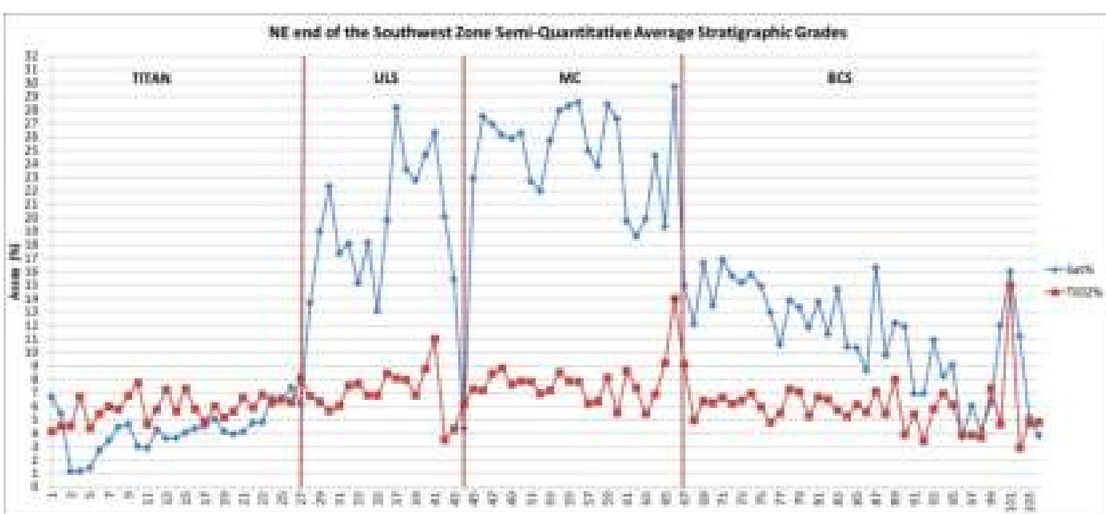


Figure 7-13: Average Satmagan values by down-hole composite position within the stratigraphic sequence for the northeast end of the southwest zone

Idealized section through the mineralized units averaged composite values on all drilling between Southwest drill panel 1400 and 2600. Horizontal axis is the relative composite position down hole. Note the sharp contacts between BCS and MC, and ULS and Titan.

The resulting plot is considered semi-quantitative and representative of the general trends in the data. The mineralized package is comprised of interlayered oxide rich and moderately graded bands that were deposited in a rhythmic way. There are baseline shoulders in both the Titan and the BCS where the lowest values are found. There is also strong evidence in these plots of two types of deposition. One is a steady state background deposition that is always present, has relatively low variance and has a ratio of around 1:1 Satmagan: TiO_2. The second is dominated by magnetite and the ratio increases up to >3:1 with the addition of magnetite cumulate cycles that overprint on the background crystallization.

The cumulate events represent the bulk of the contained metal in the deposits. Their frequency and thickness steadily ramp up through the BCS until there is a sustained jump in grade that signals the entry into the MC. The ratio and overall grades of Satmagan and TiO_2 then generally decrease slightly during deposition of the ULS. The contact with Titan is generally marked reduction in overall grades, and the frequency and thickness of cumulate bands fall off significantly.

The average grades in these figures can also be used to track individual higher-grade bands that appear to be relatively wide in the MC horizon over the surrounding units and may represent the intensity of deposition more than the length of time. As the data passes into the ULS, the frequency of the oscillation increases, which may indicate entry into a cyclic chemical process that was controlling partitioning.

There are pockets of high-grade that formed locally in the upper BCS and lower portion of Titan that may indicate sporadic local triggering of oxide deposition that became more prevalent and eventually formed high-grade blankets once the conditions in the chamber allowed. Local pockets of high-grade cumulate are responsible for a slight ramping of average grades near the contacts with the MC and ULS respectively.

7.9.4. Results of the Armitage Deposit Model

The Armitage Deposit shows similar stratigraphic beds to those in the Southwest Deposit (Figure 7-14). There are, however, subtle differences in the character and tenor of the deposit. A first impression from the plots is that it displays remarkably consistent grades of both Satmagan and TiO_2 mineralization compared to the Southwest (Figure 7-15). The average grades in most of the stratigraphic package are similar to what is seen in the southwest end of the Southwest Deposit with the exception of the TiO_2 in Titan. Here, the average in Southwest is about 6% compared to the 4% seen in Armitage, which is a significant difference.

A second striking difference between the two deposits is the relative consistency and lack of dramatic threshold steps in grades in the Armitage. There is a general smoothing of the grades over BCS, MC and ULS to the point where it is more subjective where the contacts for these horizons lie. Recall that the MC mineralization was well defined as 3+ composite intervals of >10 Satmagan in the Southwest, but in parts of the Armitage there are long runs of Satmagan >10, and other ratio criteria and natural breaks in the mineralization have to be employed to make the identification.

The overall sense is that the Armitage mineralization shows stronger continuity and lower-grade variance than the Southwest. There is a general trend noted of decreasing SAT% going from the NE to the SW along the entire trend at a rate of about 0.5% SAT per km. It is thought that this might indicate some influence from the heat generated at the Grenville Front during thrusting and accretionary docking of that terrane, or it may be related to proximity to magmatic injection.

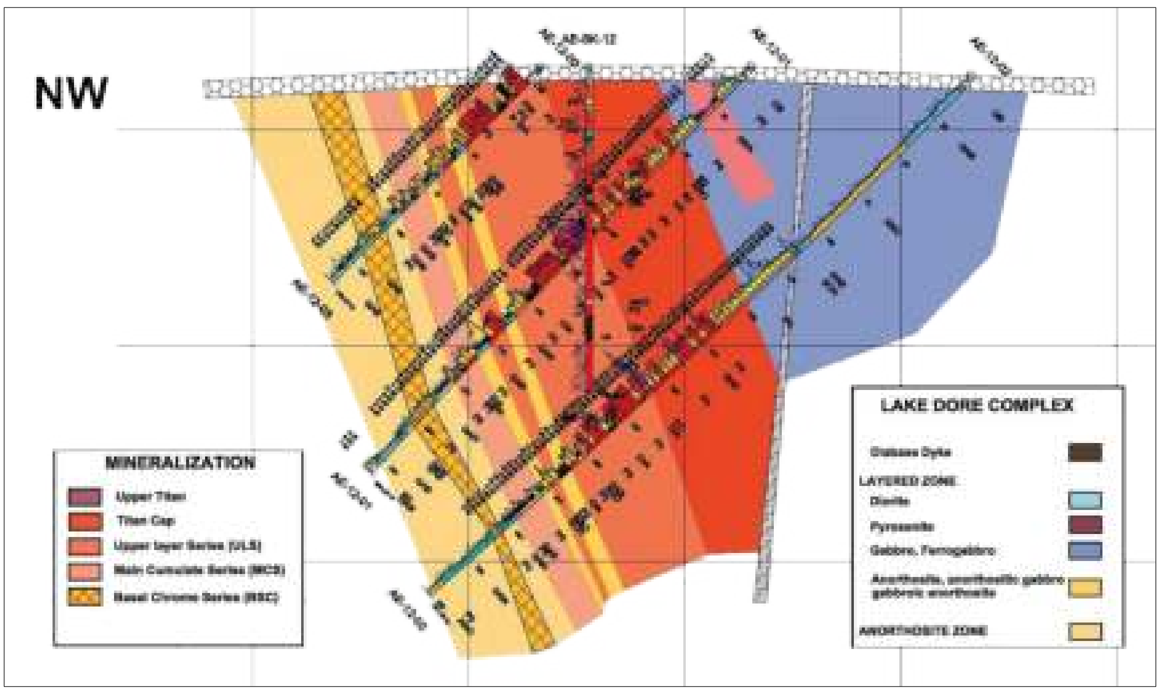


Figure 7-14: Typical section of the Armitage Deposit looking northeast

Note the similarity with the Southwest section presented in Figure 7-12.

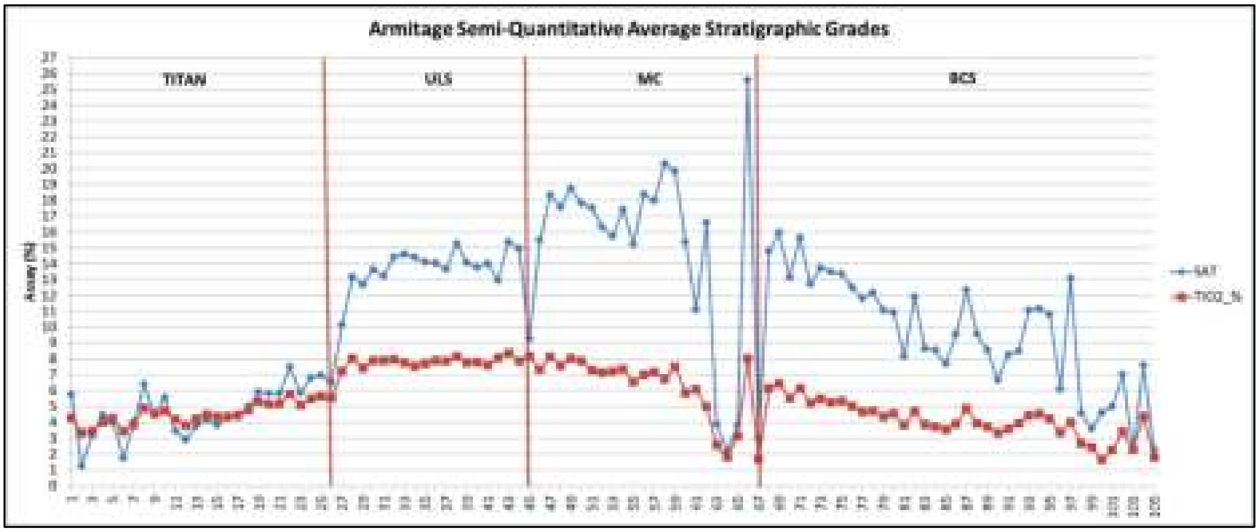


Figure 7-15: Idealized section through the mineralized units averaging composite values on all drilling in Armitage

Horizontal axis is the relative composite position down hole. Note the gradational contact between BCS and MC, and the relatively sharp contact between ULS and Titan. Also note that the average grades for the BCS, MC and ULS units are quite similar, and that the overall variance in grades is lower than seen in Southwest.

7.10. Variation in Thickness

One of the basic tenants BlackRock has employed is that the geologic model is governed by sedimentary processes, and that this resulted in a steady rain of mineralization onto the floor of the magma chamber. As a first approximation, the un-deformed thickness is assumed to have been relatively constant.

The following graphs (Figure 7-16) summarize work undertaken to measure the true model thickness of the four stratigraphic units in each of the deposits, and to determine how they vary along strike. It provides an independent test of the results of the modelling as they relate to the stratigraphic relations pre-deformation, and the structural effects that have been overprinted on them.

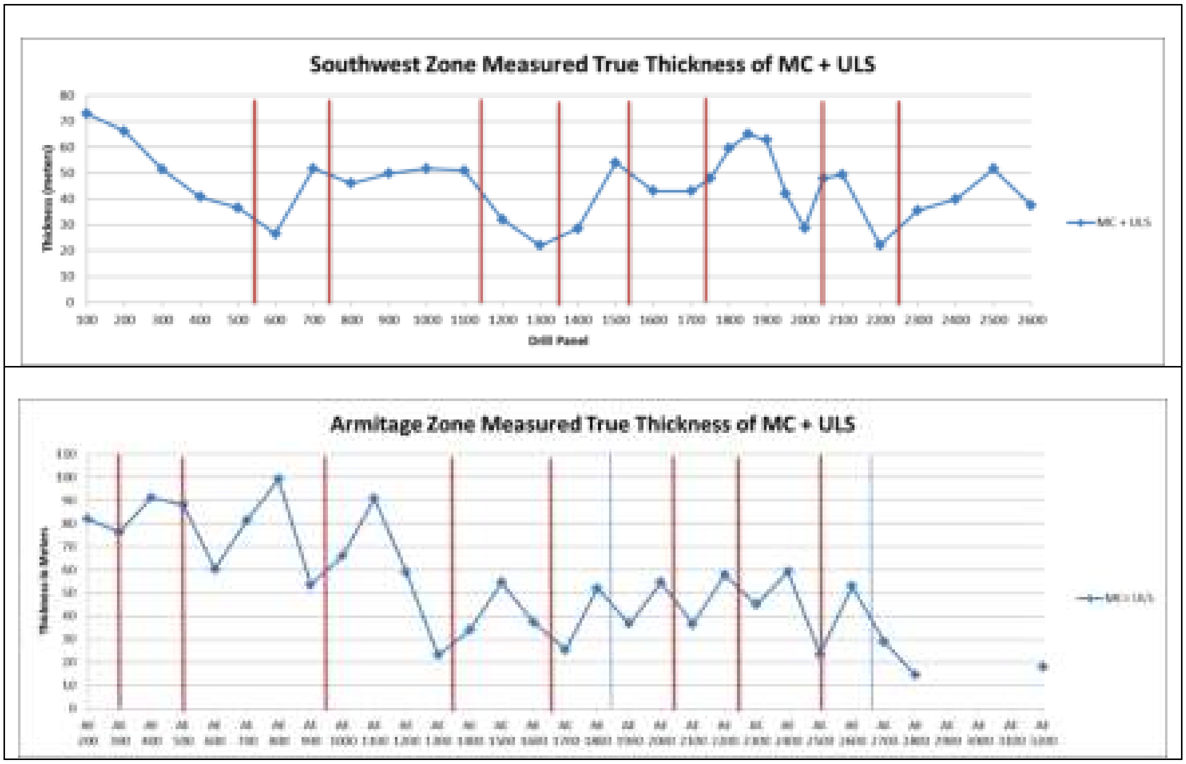


Figure 7-16: Graphs of the average thickness of the MC+ULS on each drill panel in the Southwest (upper graph) and Armitage (lower graph)

Domain breaks are shown in red, with additional suspected structures in light blue. Horizontal axis represents the drill panel numbers.

The following assumptions were used to derive the data presented in these plots:

1. All measurements were made at the 400 m elevation.
2. True thickness was scaled from the modeled polygons for each drill panel for each of the deposits.
3. Measurement was made perpendicular to the boundaries of the polygons. Where the top and bottom contacts of a polygon were not parallel, a distance was measured to an average position that approximated the optimal 90-degree angle.
4. Where there were multiple polygons with the same stratigraphic designator, the aggregate of the measurements of those polygons was recorded.
5. Where there were structural or dyke gaps or repeats in sections, data adjacent to the compromised 400 m level was used. This happened in a limited number of instances.

6. Of the four stratigraphic units, only the MC and ULS have seen their top and bottom contacts consistently defined in all drill panels.
7. The top of Titan is poorly constrained at this time: The bottom of BCS is not well drilled everywhere either.
8. As a result of the analysis, the plots utilize an aggregate thickness of MC+ULS.

7.10.1. Southwest Thickness Variation

The Southwest plot shows a fairly broad variation with several areas of relatively equal thickness punctuated by zones where the aggregate thickness drops over 1-3 drill panels. Note the area between 1700-2100 where there are additional data points provided by the 50 m offset sections (i.e., 1700+50, 1800+15, etc.). In these areas the thickness data has twice the resolution but the thickness values fall essentially into place on the trend lines between the wider spaced 100 m sections.

7.10.2. Armitage Thickness Variation

The Armitage plot reveals a simpler and more frequent and regular variation of thickness. There are two significant observations of which to take note. First, the average aggregate thickness effectively doubles between drill panels 1200-1300 with a range of 20-60 m on the east side, and 55-100 m on the west side of this line. Second, and coincident with the increased thickness, is an increase in the peak-to-peak wavelength of highs, with true length of about 300 m on the west side where maximum thickness is 100 m, to about 200 m on the east where maximum thickness is about 60 m.

7.10.3. Depositional Variation

The coincidence of the change in wavelength with the change in thickness of the relatively rigid cumulate series is strongly suggestive of a pinch and swell, or boudinage, style of deformation (Ramburg, 1955). If this is the case, then the un-deformed thickness of the MC+ULS would be roughly equal to the thickest portions of the plot with the attenuation localizing in the necks of the boudins.

With this assumption, the original thickness of the units would have been about roughly 55 m for the blanket of mineralization from drill panel 1200 in the Armitage all the way to 2600 in the far northwest end of the Southwest (with a gap where there is no cumulate mineralization identified between the two zones). Below 1200 in the Armitage and all the way to the end of the drilling on 200, the thickness increases to about 90 m. There is a 400 m span that forms a gap where the nature of the increase is obscured by the structural attenuation zone, and it is unclear if this was originally an abrupt structurally controlled thickening or a gradational increase.

7.11. Comments on Section 7

One possible explanation for the apparent deformation that is outlined in these data is that the relatively competent beds comprised of oxide cumulate may have formed pinch and swell or boudinage structures during lateral extension under compressional stress. The direction of compression was probably normal to the accretionary sutures that are so prominent in the cratonic map of North America (Figure 7-1), which is coincidentally normal to the inclined bedding of the cumulate layers.

The photograph presented in Figure 7-17 shows typical examples of this type of deformation. This could help us to explain:

1. Bedding parallel shear fabrics that are commonly noted on drill logs, as well as some of the "clipper faults" we have used in our 3D models.
2. The large number of domains and bounding faults but the limited number of offsetting faults. Things appeared jostled, not displaced and this is consistent with the bounding faults being non-penetrative structures.
3. Systematic variation in thickness of the layered zone.
4. Systematic variation in grades and ratio values across the deposits.

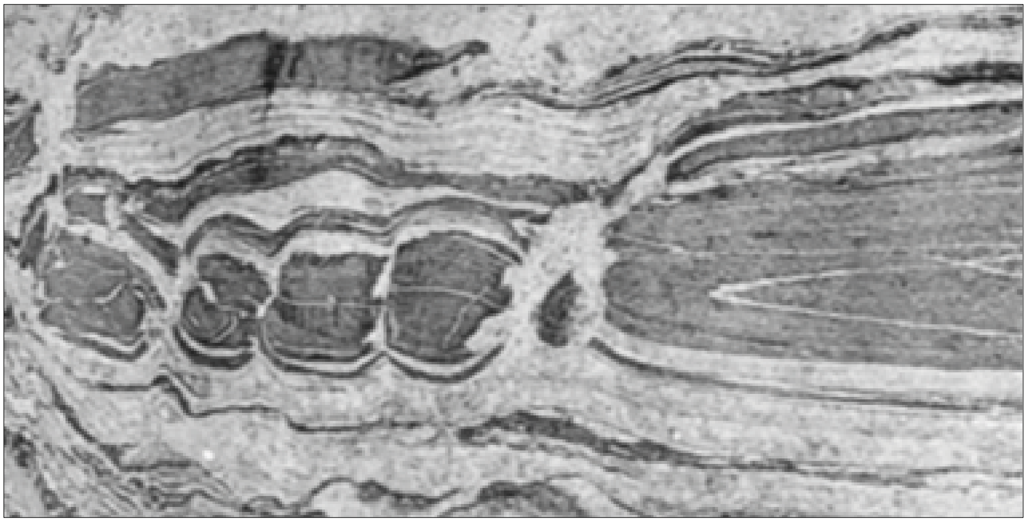


Figure 7-17: A typical pinch and swell or boudinage structure (modified from Ramburg, 1955)

Note the variable small-scale rotation of the typically barrel-shaped segments, which can be separated by thin non-penetrative structures, or alternately large areas of extension where the more fluid layers have flowed into the space between the separating boudins.

This could also explain why there is a change of strike between the Armitage and the Southwest. Figure 7-18 shows the ground magnetic map to demonstrate the hinged rotation of the Southwest Zone out of line with the remainder of the LDC.

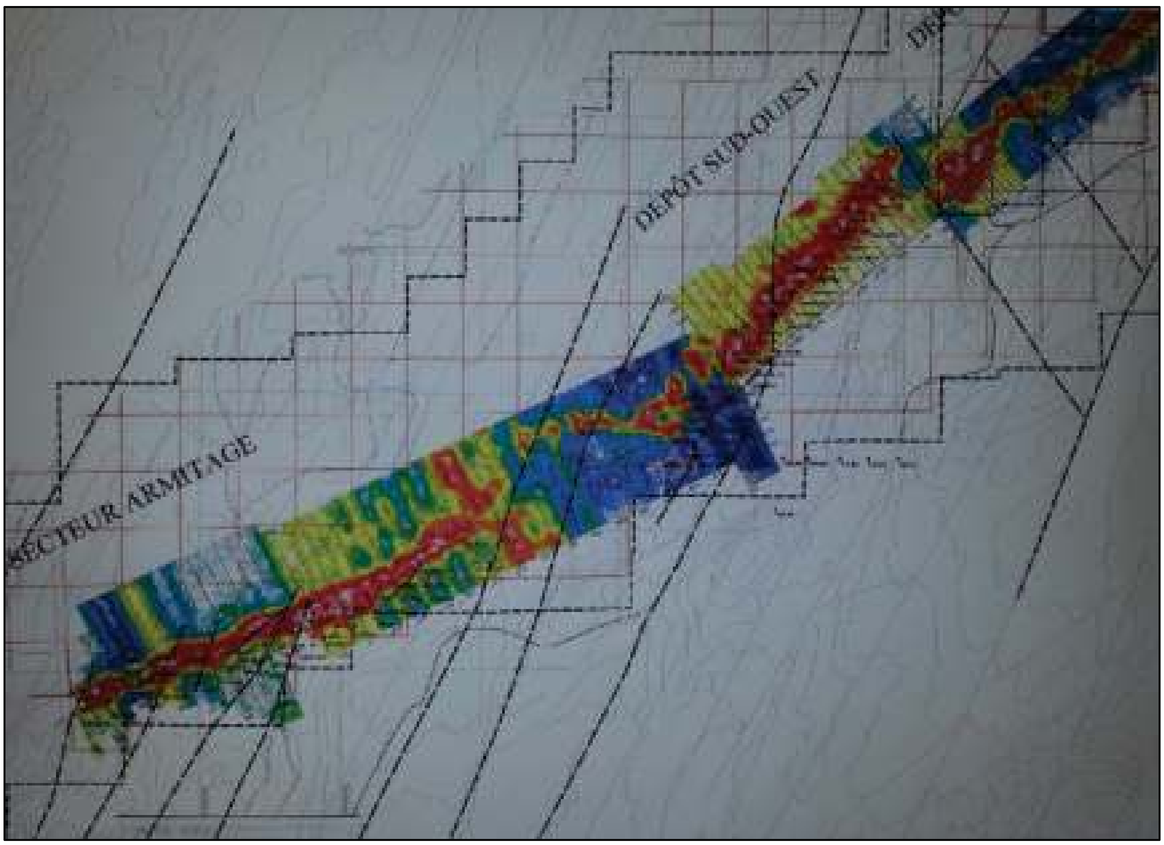


Figure 7-18: Ground magnetic map showing the general trends of the Southwest and Armitage Deposits

Note the apparent hinge rotation of the Southwest Deposit out of line with the rest of the magnetic trend.

The variable leveling quality of the underlying magnetic data is also demonstrated here. Note the background level is variable across the survey area in blocks that probably represent different acquisition days. In the southwest end of the Southwest Deposit the magnetic signature is very obscure as a result and never reaches above the blue level. This may also be due in part to the 45-degree dip of the stratigraphy.

The BlackRock Deposit is unusual in its relatively simple mineralogy and clean mineral chemistry. This leads to superior quality in both magnetite and ilmenite concentrates that are produced, BlackRock believes that this is directly related to the timing of emplacement and the condition of both the parent magma as well as the host rocks to the intrusion.

BlackRock's Deposit formed at a favorable time of crustal evolution, at 2.7Ga (Figure 7-19). The backdrop for this figure is work done mapping orogenic events worldwide through the full span of the Earth's history. The proxy for these orogenic and continent building events is the distribution of zircon formation through time (Bradley, D.C, 2011).

A second line of evidence that supports BlackRock being formed as part of the very first rock cycle marking the birth of modern plate tectonics. Shirley et al. (2012) looked at age dates for various mineral inclusions found in diamonds, and more specifically those derived from reworked material. Again, there was a very large sample size, and they specifically looked at the inclusion of eclogitic minerals as an indicator of the first occurrence volcanic rocks that had been subducted to a sufficient depth to undergo this level of metamorphism. That study found that the oldest eclogite inclusions dated to 3Ga which is coincident with the beginning of the orogenic events that produced the zircons at the time of the assembly of the Superior Craton.

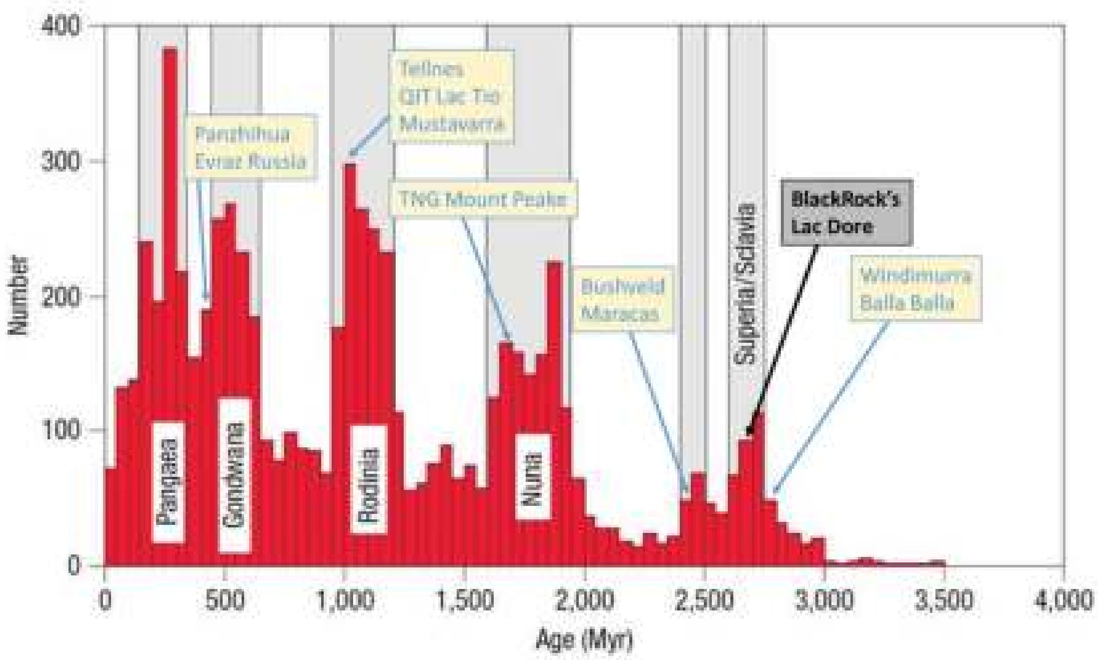


Figure 7-19: Location of BlackRock's Deposit in time, modified from Bradley, 2011

As a result of being emplaced at the very earliest stages of the rock cycle that is one of the hallmarks of plate tectonics, BlackRock benefited from limited crustal "distillation" products to contaminate the melt. This further benefited through its emplacement into an oceanic crust that shared a very similar chemistry on the edge of the Superior Craton; while there was significant country rock assimilation during this event, no contamination with deleterious elements took place.

Panzhihua provides a marked contrast and opposite end-member in time. It is hosted in evolved silicic continental rocks and was intruded only 258Ma during a continent-continent collisional event. It shows significant quality degradation and in comparison, with the pristine condition of BlackRock's mineral concentrates as a result.

A second factor in that is key to the simple processing and mineral separation at BlackRock is the lack of any primary or secondary hematite. Deep tropical weathering profiles have adversely affected many of the Brazilian and Australian VTM's. As noted, BlackRock has no significant alteration displayed in the ore zones, and is a pure end-member titanomagnetite. Hosted in a northern latitude and being subjected to the grinding force produced by three kilometres of ice scouring off any weathering products has removed any alteration that might have been present prior to the last glaciation. The Laurentide Ice Sheet also acted to keep the newly exposed primary minerals fresh and under ice until the final withdrawal of the continental glaciers 11,700 years ago.

8. Deposit Types

The geological setting and mineralization encountered on the BlackRock Property, located in the Lac Doré Complex (LDC), indicates some similarities with other world-class magmatic Fe-Ti-V oxide deposits associated with layered mafic intrusive rocks (see Section 7-4). On the other hand, the age of the LDC is different, and represents a major defining parameter on the crystallizing conditions of its mineralogy and explains why its mineralogy is relatively simple in comparison to most other orthomagmatic oxide deposits in the world.

This family of deposits includes the world class Bushveld complex in South Africa (producing district), the Panzhihua layered intrusion in China (producing district), the Windimurra complex in Australia (producing district), the Maracas Deposit in Brazil (in development), the Skaergaard complex in Greenland (advanced exploration) and the Bell River Archean complex in the Matagami region of Québec (prospect).

Large known ilmenite and titaniferous magnetite deposits are hosted in massive and layered intrusive complexes. Titaniferous magnetite is associated with gabbro and leucogabbro intrusions, whereas when ilmenite is the major mineral phase, it is associated with proterozoic anorthosite. As a result, the iron- and titanium-rich deposits can be classified as two subtypes of the magmatic Fe-Ti-V oxide deposits, on the basis of the principal ore minerals and the petrology of the host intrusions. The proportions of the principal ore minerals vary from ilmenite-dominant in anorthosite host rocks to titaniferous magnetite-dominant in gabbro and leucogabbro host rocks. The classification is based on the document: Gross G. A. et al., 1998; "Magmatic Ti-Fe-V Oxide Deposits".

This classification subdivides the layered ultramafics of Type 26 into the following two subtypes:

Subtype 26.1: These deposits consist mainly of ilmenite and hemo-ilmenite with minor titaniferous magnetite, and form massive irregular discordant intrusions or layered bodies, hosted in massive proterozoic anorthosites. Important examples are Lac Tio, exploited by QIT (Lac Allard 1.160 Ga), Degrosbois, Lac des Pins Rouges, St-Urbain, and Ivry (Morin anorthosite) in Québec, Canada; Tellnes (0.925 Ga) in Norway; Sybille (1.434 Ga) in the USA; and the Ilmen Mountains in the former U.S.S.R. The age range is restricted mainly to the Mesoproterozoic (1.600 – 1.000 Ga).

Subtype 26.2: These deposits consist mainly of titaniferous magnetite and ilmenite mineral assemblages hosted in layered and/or massive intrusions of leucogabbro, gabbro, norite, and rocks of intermediate composition. Examples include the Lac Doré Complex (BlackRock, 2.727 Ga), Sept-Îles (540 Ma), Magpie Mountain, St. Charles, Kiglapait, Newboro Lake, and Lodestone Mountain in Canada; Emeishan (Panzhihua 258 Ma) in China; Smaalands-Taberg in Sweden; the Bushveld Igneous Complex (2.054 Ga) in South Africa; Kachkanar and Kusinskoye in the former U.S.S.R.; Tahawus and Iron Mountain (1.500 Ga) in the United States; Windimurra (2.813 Ga). The time range of Subtype 26.2 deposits is unrestricted, ranging from the Archean (3.800 Ga) into the Phanerozoic (258 Ma).

Deposits of both subtypes include irregular discordant masses in layered or massive intrusions, and concordant oxide-rich layers produced during fractional crystallization. The latter is typical of the BlackRock Project that exhibits the principal ore minerals of "reduced" oxides of iron in ilmenite, magnetite, and titaniferous magnetite. There is a lack of extra oxygen to fully oxidize the ferrous (Fe2+) oxides in magnetite and ilmenite to form the ferric (Fe3+) oxides in hematite and hemo-ilmenite.

The term "titaniferous magnetite" refers to crystal aggregates and exsolution intergrowths consisting of ilmenite, magnetite and titanomagnetite (a solid solution of Fe_3O_4 - Fe_2TiO_4).

These subtypes reflect differences in age, geological context and mineralogy and, when taken into consideration, explain why one deposit cannot be directly compared with another.

Deposits of both subtypes provide resources of titanium, vanadium, and iron. Some deposits contain important quantities of apatite (Gross, 1967a; von Gruenewaldt, 1993).

8.1. Geological Setting

Deposits of Subtype 26.1 are hosted worldwide in anorthosite as the Lac Tio (Rio Tinto QIT) in the Grenville Province in Québec and the Tellnes in the Rogaland Anorthite Province of Norway. Most of the deposits form discordant dykes, sills, and stock-like masses in the host anorthositic rocks. Others are layered concentrations of Fe-Ti oxides within anorthosite or gabbro, concordant to layering in the host and to the internal fabric of late stage intrusions.

Subtype 26.2 deposits are hosted worldwide in mafic layered and massive intrusions. The layered deposits generally form concordant, laterally continuous magnetite-rich layers measuring centimetres to metres thick. Deposits in massive intrusions usually consist of disseminated titaniferous magnetite. Deposits of Subtype 26.2 also include massive discordant stock-like bodies of Fe-Ti oxide in layered deposits, as at Newboro Lake in Canada. The host intrusive complexes are typically differentiated and include gabbro, leucogabbro, diorite, diabase, gabbro-diorite, and quartz monzonite.

Concentrations of metallic oxide minerals in both Subtypes 26.1 and 26.2 are conspicuously developed in four styles:

1. Disseminated syngenetic metal oxides in the host rocks;
2. Irregular to conformable auto intrusions that have sharp to indistinct or gradational borders with earlier phases of the host anorthosite and gabbro, and were emplaced during the lithification and cooling of the host intrusive rocks;
3. Late stage dykes and intrusions transecting the lithified host anorthosite and gabbro complexes;
4. In the skarn rock and alteration zones at the contact of the host intrusions and wall rocks.

8.2. Form of Deposits and Relation to Host Rock

Generalizations on the form and relationships of these deposits to host rocks are tenuous because of the many variations from deposit to deposit in the host rocks, mineralogy, and geological settings. Both subtypes of Fe-Ti oxide deposits occur in two general forms: massive lenses, dykes, sills, and irregular intrusions; and stratiform, layered, concordant, or irregular bodies. The Fe-Ti oxide minerals may be disseminated and interstitial to the silicate minerals or occur as massive aggregates separated from them. Deposits of Subtype 26.2 are predominantly stratiform and layered as the Lac Doré Complex. In some cases, as for Tahawus and Iron Mountain, attributes of both forms are combined in a single intrusive complex.

Ilmenite deposits of Subtype 26.1 are typically massive discordant intrusive bodies in anorthositic host rocks but some also occur as conformable layers within late stage gabbroic, troctolitic, and dioritic intrusions in anorthosite. Some of the Fe-Ti oxide masses, especially along their borders with the host rocks, have local fragmented or brecciated structures, show evidence of plucking and stoping of the enclosing rocks, and contain abundant xenoliths of anorthosite and xenocrysts of plagioclase derived from anorthosite. Both massive and disseminated ores are found within a single intrusion. The massive discordant intrusions of Fe-Ti oxide range in shape from sinuous dyke-like forms to irregular equi-dimensional masses.

Layered stratiform deposits of Subtype 26.2 hosted in gabbro and leucogabbro usually contain layers of disseminated titaniferous magnetite which alternate with layers of feldspar and mafic silicate minerals. Individual layers range in thickness from centimetres to tens of metres. Lateral continuity of oxide-rich layers in large intrusions can be in the order of several thousands to millions of metres. For instance, the Lac Doré complex is over 20 km long and the Emeishan deposit (Panzhihua) occupies an area of 660,000 km^2.

8.3. Ore Mineralogy, Composition, and Texture

The proportions of the common ore minerals, ilmenite, hemo-ilmenite, titaniferous magnetite, titanomagnetite, and magnetite vary greatly from one deposit to another. The complex exsolution textures and mineral relationships that indicate mineral paragenesis and sequence of crystallization vary greatly and appear to be distinctive for individual deposits.

The principal ore minerals in deposits of Subtype 26.1 are ilmenite, hemo-ilmenite and their exsolution intergrowths, and titanomagnetite. They are associated with plagioclase, pyroxene, olivine, garnet, biotite, apatite, ulvospinel, quartz, hornblende, rutile, and pyrrhotite, which are present in varying proportions. Hemo-ilmenite, the principal ore mineral at the Lac Tio and Tellnes deposits, is hosted in anorthosite and is typically equigranular with coarse exsolution lamellae of magnetite that constitute as much as 30 mole per cent of the crystal grains. A second set of very fine exsolution lamellae of ilmenite is commonly developed within the broad haematite lamellae. The relicts of earlier titanomagnetite crystals can he recognized where the diagnostic trellis lamellae of ilmenite are still preserved along the {111} planes of the host magnetite.

Some parts of the Lac Allard ilmenite Deposits contain 8 to 10% fluorapatite (Gross, 1967a). Ilmenite-apatite occurrences (nelsonites) have been reported in many anorthosites (Kolker, 1982). Some of the anorthosite-hosted Fe-Ti oxide deposits contain minor rutile, sapphirine, corundum, sillimanite, and graphite (Ashwal, 1993).

In the Subtype 26.2, the principal ore minerals in deposits of are titanomagnetite, and other varieties of titaniferous magnetite and ilmenite that occur as discrete crystals and as exsolution intergrowths in various proportions in magnetite. They are associated with plagioclase (commonly labradorite), olivine, pyroxene, and small amounts of apatite, titanite (sphene), rutile, spinel, biotite, pyrite, chalcopyrite, and pyrrhotite.

It is important to note that the composition of the country rock that the ultramafic body has intruded may affect the quality of the ore body in terms of picking-up of detrimental elements. For instance, the Lac Doré Deposit has intruded into proto-lithosphere with relatively few contaminants. In contrast, the Panzhihua Deposit is intruded into lithospheric on lithospheric plates as the Indian plate pushed into Eurasian plate. The parental melts are thought to have been derived from a mantle plume contaminated by interaction at relatively shallow depths with an enriched lithospheric mantle (Song et al., 2001).

The Lac Doré Deposit is distinctive with its lack of major contaminants. This may partly be explained by the Archean age of the deposit, by relatively low oxygen fugacity in the melt compared to the latter Proterozoic intrusions of the Subtype 26.1 that exhibit hemo-ilmenite; i.e. the magnetite component has been oxidized to hematite.

8.4. Metamorphism

This has quite an effect on individual deposits even though of the same subtype. As previously mentioned, the Lac Doré of Archean age (2.727 Ga) has undergone greenschist facies metamorphism, so that the original mineralogy of mafic silicates of olivine and pyroxenes was transformed to softer chlorite minerals, the plagioclases have undergone saussuritization to clay minerals, all producing a softer meta-ultramafic rock. (See Section 9.6) In contrast, the Panzhihua ore body (Emeishan) being considerably younger at 258 Ma, the mineralogy remains relatively fresh and unaltered, requiring a good deal more energy to crush and liberate the ore minerals.

8.5. Definitive Characteristics of the Ore

- Massive and layered ilmenite and hemo-ilmenite deposits (Subtype 26.1) are hosted in anorthosite. Layered and massive concentrations of titanomagnetite, titaniferous magnetite, magnetite, and ilmenite (Subtype 26.2) are hosted in differentiated mafic layered and massive intrusions;
- Subtype 26.1 deposits are massive irregular to tabular bodies with disseminated masses of coarse-grained ilmenite containing blades of exsolved hematite, pure ilmenite, and titaniferous magnetite. All hosted in massive or layered anorthosite and leucogabbro intrusive complexes, stocks, and sills;
- Typical Subtype 26.1 deposits contain from 20 to 40% titanium and 25 to 45% iron with Fe/Ti ratios of about 2:1;
- Subtype 26.2 deposits consist of layered disseminated concentrations and massive irregular to tabular intrusions of titaniferous magnetite, titanomagnetite, magnetite, and ilmenite. These minerals are distributed as discrete grains, and as granular and exsolution intergrowths. The host silicate phases include gabbro, gabbroic anorthosite, and other differentiated intrusive complexes ranging in composition from gabbro, through norite, quartz monzonite, to syenite;
- The iron content in Subtype 26.2 deposits ranges from 20 to 45%; Ti ranges from 2 to 20%; Fe:Ti ratios vary from 40:1 to 2:1 and are commonly about 5:1. The content of P205 varies to a maximum of about 8% and the content of V, Cu, Ni, Cr, and Mn may vary greatly, but the average for each element is about 0.25% or less;
- As a group, Subtype 26.2 deposits vary greatly in composition, mineralogy, and physical characteristics, but individual deposits are fairly uniform;
- The mafic-hosted titanium-iron deposits of both subtypes vary greatly in character and composition depending on the kinds of associated host intrusions, the stage of differentiation and oxygen potential in the magma from which they were derived, tectonic setting, and mobilization of elements during metamorphism (cf. Yoder, 1968);

- They are important as sources of titanium oxide and high quality iron metal that are recovered as co-products, and as resources of iron ore concentrate in which the titanium content can be reduced to 1% or less.

8.6. Genetic Models for Mafic Intrusion-Hosted Titanium-Iron Deposits

The titanium-iron deposits that are associated with Proterozoic anorthosites and layered mafic intrusions are clearly late products of the crystallization history of individual intrusions. Brecciation of ore-hosting anorthosite and truncation of structural elements in anorthosite are clear evidence for late intrusion of the ore-forming magmas in many Subtype 26.1 deposits. Conformable layers in small intrusions in anorthosite and in large mafic layered intrusions throughout the world indicate an origin by crystal settling and accumulation on the floors of magma chambers for Subtype 26.2 deposits and parts of Subtype 26.1 deposits.

Both subtypes of deposits require extensive periods of prior plagioclase crystallization to concentrate Fe and Ti in residual magmas, and variations in the oxidation state of the magmas (monitored by the intensive parameter – oxygen fugacity) to promote the formation of the titanium-iron deposits. Hemo-ilmenite deposits (Subtype 26.1) require relatively more oxidizing conditions of formation compared to the more reduced titanomagnetite deposits (Subtype 26.2).

Evidence is lacking for the presence of hydrous fluids during formation of the Ti-Fe deposits, although CO2-dominant fluids were likely present. The preserved primary mineral assemblages are typified by anhydrous mineralogies. Hydrous minerals are always late and volumetrically minor (<<1%) or definitely related to cross-cutting monzonitic or granitic intrusions. The presence of grain-boundary graphite and CO2-rich inclusions in apatite from anorthosite indicates that the very small amounts of fluids associated with anorthosite were probably CO2-dominated.

The origin of conformable Fe-Ti oxide-rich layers in layered intrusions is more straightforward than that for the massive discordant intrusions. The conformable layers represent the overproduction of Fe-Ti oxides in a progressively crystallizing magma, mainly in response to local variations in oxygen fugacity (Morse, 1980). Prior to the cumulus arrival of magnetite and/or ilmenite in a magma, protracted crystallization of plagioclase will enrich the residual magma in Fe, Ti, (and V), and increase the density of this residual melt. The prominent titanomagnetite layers in the Kiglapait intrusion of Labrador (Morse, 1969, 1980) and the Bushveld Igneous Complex of South Africa (Willemse, 1970; Reynolds, 1985) occur relatively high in the stratigraphic sections of these intrusions, and require crystallization from magnetite-supersaturated liquids.

The compositions of Fe-Ti oxides in both hemo-ilmenite-rich and titanomagnetite-rich ores can undergo substantial modification during cooling by both intra- and intercrystalline reaction and exchange. During slow cooling, the titanium component in titanomagnetite may be exsolved by oxidation to form either discrete lamellae of ilmenite in magnetite, or granular exsolutions of ilmenite around magnetite grains, a process called oxy-exsolution (Buddington and Lindsley, 1964):

$6Fe_2TiO_4$	+	$O_2 = 6FeTiO_3$	+	$2Fe_3O_4$
in magnetite		ilmenite		magnetite

This reaction may be facilitated by the presence of a CO_2-rich fluid and can occur to very low temperatures (400-500°C). As a result, titanomagnetite grains can purge themselves entirely of the original titanium component and the resultant ore mineralogy and texture is one of interlocking discrete grains of magnetite and ilmenite. In addition, at relatively high temperatures, exchange of titanium and iron between individual grains of magnetite and ilmenite can occur according to the following equilibrium reaction, which proceeds to the right with decreasing temperature:

Fe_2TiO_4	+	Fe_2O_3	=	$FeTiO_3$	+	Fe_3O_4
in magnetite		in ilmenite		ilmenite		magnetite

This produces magnetite and ilmenite grains that will approach their end-member compositions as cooling proceeds.

Oxidation of ilmenite-rich deposits can result in the alteration of ilmenite to rutile. Associated alteration of silicate and Fe-Ti oxide minerals always postdates the formation of the deposits.

Table 8-1: Comparison of deposit types

Deposit Type Category		26.1	26.1	26.2	26.2
Mine		Tellness	Lac Tio	BlackRock	Emeishan
Supplier		Titania	QIT	BlackRock	Panzhihua
Country		Norway	Canada	Canada	China
Geological Province/Event		Rogaland	Grenville	Superior	Emeishan
Age Ma		925	1160	2727	258
Host Rock		Anorthosite	Anorthosite	Gabbro	Gabbro
Ore type		Ilmenite	Ilmenite	Ilmenite	Ilmenite
Reference		Titania	QIT	TZMI	TZMI
Active Ingredient	TiO_2	44.00	35.00	47.70	47.40
Iron Ferric	Fe_2O_3	13.00	25.20	4.86	5.60
Iron Ferrous Reactive	FeO	34.40	27.50	42.30	34.30
Vanadium	V_2O_5	0.17	0.27	0.26	0.10
Calcium	CaO	0.24	0.90	1.05	1.30
Magnesium	MgO	4.50	3.10	0.39	5.30
Chromium	Cr_2O_3	0.08	0.10	0.01	0.02
Manganese	MnO	0.31	0.20	0.93	0.65
Alumina	Al_2O_3	0.60	3.50	1.51	1.20
Silica	SiO_2	2.61	4.30	2.21	3.00
Arsenic	As_2O_3	0.00	0.00	0.00	0.00
Niobium	Nb_2O_3	0.01	0.00	0.00	0.03
Phosphorus	P_2O_5	0.02	0.00	0.02	0.02
Sulphur	S	0.00	0.00	0.13	-
Zirconia	ZrO_2	0.00	0.00	0.02	0.07
Radioactive (U +THPPM)		<0.8	<0.5	-	-
Subtotal		99.94	100.07	101.39	98.99

* Arsenic, Niobium, Phosphorous, Zirconia and Radioactive Elements (U+Th) are in traces amounts or below detection limits

9. Exploration

Since 2008, BlackRock has carried out extensive exploration work over its property and more specifically over the Southwest and Armitage Pits. The work has consisted of the following programs:

- Airborne magnetic survey and digital topography of the entire magnetite bearing envelope (17 km X 1 km);
- Compilation of historical work;
- Mapping of the entire magnetite bearing envelope (17 km X 500 m);
- Access road clean-ups over the Southwest and Armitage Pits;
- Trenching (650 m, three trenches in the Southwest Deposit);
- Surveying precise locations of all trenches (Southwest Deposit);
- Systematic 3 m composite channel sampling of the Southwest Deposit trenches (544 samples in 10 trenches spaced 200 m apart);
- Diamond drilling of the Southwest Deposit on nominal 100 m hole spacing and 100 m drill panel spacing over 2,500 m of strike;
- Diamond drilling of the Armitage Pit on nominal 100 m hole spacing and 100 m drill panel spacing over 2,600 m of strike;
- Mineralogical and metallurgical testing at Corem and SGS: Satmagan, thin sections, polished sections, DTA, WRA, pycnometre and specific gravity.

A low altitude helicopter-borne magnetometer survey, at 50 m spacing, totalling 1,995 km of lines and covering the entire Property, was flown in October 2008 (Novatem, 2008). It outlined with precision the magnetite-bearing horizon that hosts the mineralization and clearly indicated its excellent continuity over the entire length of the Property. Gérard Lambert Géosciences Ltee produced an interpretation report, in the spring of 2009 (Section 9.5.1).

In 2009, BlackRock completed an extensive compilation of historical work performed on and near the Property.

9.1. Grids and Surveys

A local reference grid was established for each of the deposits independently. These grids are oriented perpendicular to the long axis of mineralization, and all drill panels labels were established relative to the distance from the southwest end of each grid where an origin (0,0) was set.

A model grid space was also established for the block models for each deposit. All surveyed information was rotated into the model space to facilitate the work, and the results rotated back into UTM space upon completion.

All surveyed location information for samples and drillholes is reported using the Universal Transverse Mercator (UTM) geographic coordinate system NAD83, Zone 18 grid. All facilities planning and engineering are also carried out in UTM space.

9.2. Geological Mapping

From 1966 through 2001, Gilles Allard and few other geologists have undertaken geological mapping over the Property and at regional scale. This continuous work has established the broader geological framework of the area and has helped to improve subsequent geological interpretations and models.

During the summer of 2009, BlackRock carried out a detailed surface mapping program over the magnetite bearing units of the Lac Doré Complex. To assist in the mapping and to find all access trails for future work, MIR Télédétection was employed to produce a satellite terrain image.

In addition, BlackRock geologists carried out some reconnaissance mapping over certain portions of the Property to assess its base metal potential.

Finally, projections of subsurface geology, as inferred from cross-sectional interpretations, were projected to plan view to produce a near-surface map of the deposits along their strike lengths. This work was used to constrain the 3-D block models that were produced for resource estimation in this study.

9.3. Geochemical Sampling

BlackRock has undertaken systematic geochemical sampling of the trenches in the Southwest Deposit and on the drill core for both the Southwest and the Armitage Pits. In trenches, all sampling has consisted of channel samples of nominal 3 m lengths not bounded on geological contact. In boreholes, samples consisted of split or sawn core and, as for the surface channels, they consisted in systematic composites of nominal 3 m lengths that were not broken on geological contact. These programs are described in detail in Section 11.5 of this report.

9.4. Geophysical Surveys

9.4.1. Airborne Magnetometric Survey

In November 2008, BlackRock carried out an airborne magnetometric survey covering the length of the Lac Doré Complex using the survey company Novatem. The survey lines were spaced every 50 m, the navigation system of the aircraft was controlled by a continuous GPS monitoring and positioning system to a precision of ± 5 m on the survey data points. The coordinate system was the UTM NAD83, Zone 18. Novatem flew the survey at 60 to 65 m above the ground at a speed of 100 km/h.

In May 2009, Gerard Lambert, P. Eng., Consulting Geophysicist, issued a report interpreting the Novatem survey entitled, "Report on Airborne Magnetometer Interpretation". The main feature of the survey was the presence of a 20 km long near continuous, northeast to southwest magnetic trend. This ribbon-like feature had a uniform strike, with local dislocations. Lambert interpreted the magnetic anomaly as a thick tabular body likely composed of multiple sheets and dipping sub vertically.

In Figure 9-1, the "purple" band shows the presence of the magnetite rich magnetic trend.

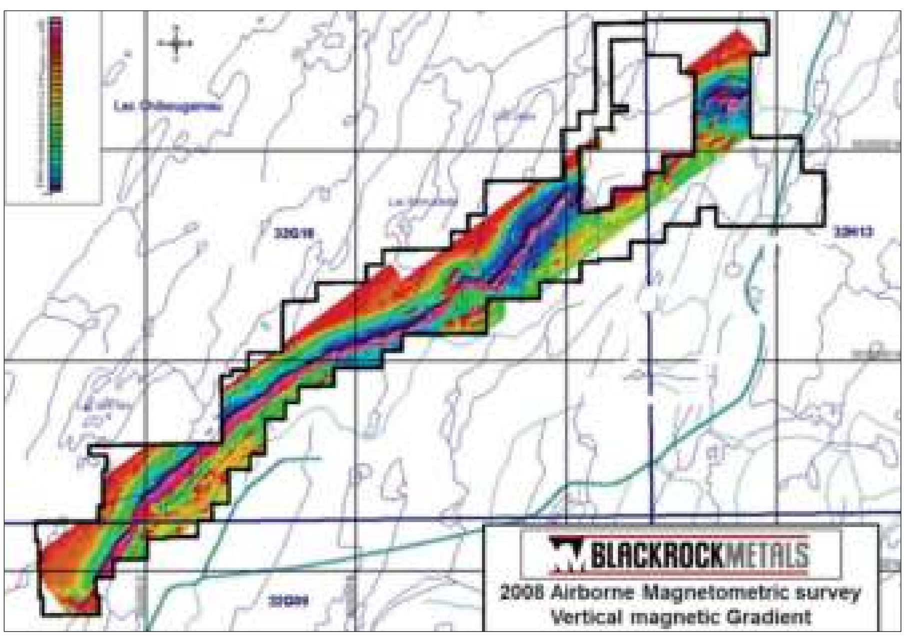


Figure 9-1: Airborne magnetometric survey showing the vertical gradient over 2017 claim outline.

Hot colors are magnetic highs; cold blue colors represent magnetic lows. Third parties currently control the blank area in the northeast corner.

9.4.2. Hyperspectral Survey

BlackRock used a full hyperspectral scan of the core by Photonic Knowledge Inc. (PK). BlackRock scanned its own entire drill core, and McKenzie Bay's available one, therefore covering over 95% of the core that has been drilled on the Property. This program of work produced a "virtual core shack", a library log of high resolution pictures of the core.

The hyperspectral scan also produced a semi-quantitative mineralogy of the core. Interpretation of PK data by BlackRock geologists, in conjunction with 3D software, proved to be useful in distinguishing structural elements of the deposit (Figure 9-2). It was possible to identify structural breaks, such as potential faults and displacements. These provided indicative data when the stratigraphy and rock unit relationships were being worked out. This was especially useful in helping to define the components and domains for the resource model.

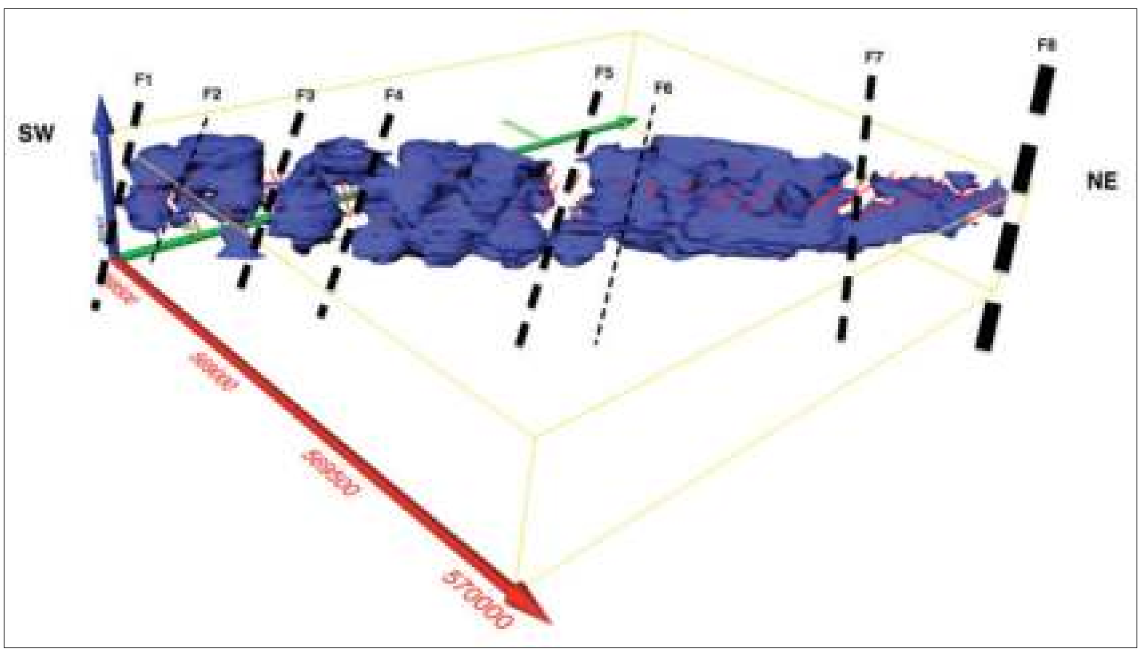


Figure 9-2: Southwest Deposit Potential Structural Components (black dashed lines) Derived from the Ilmenite 5% Isosurface (blue)

9.4.2.1. Background to the Hyperspectral Scan

The hyperspectral scan covers the electromagnetic spectrum from 400 nm to 1,000 nm, unlike a normal camera that "sees' in the visible spectral range of 400 nm to 700 nm. The result is that the hyperspectral data can distinguish the ligands, an ion or molecule attached to a metal atom by coordinate bonding. Thus, it can distinguish minerals such as ilmenite ($TiFeO_3$), magnetite (Fe_3O_4) and hematite (Fe_2O_3).

This analytical technique is non-invasive, and very fast in comparison to the more traditional whole rock analyses (WRA) and use of the Satmagan, which require destruction of the core to provide analytical data.

In the production of the spectral library, PK took numerous Scanning Electron Micro-photographs, thin sections and polished sections with probe analyses, that helped to define the ore mineralogy.

9.4.2.2. Hyperspectral Scan Methodology

The core was scanned by PK operators prior to sampling. The operators set up a mobile trailer at the core shack in Chibougamau so that, as the geologist logged the core, PK operators could scan it before sampling. It was found that the best operating conditions for scanning were on the complete core.

Before scanning, the core was dried and stored in a temporary heated shelter to ensure the core temperature was above 0°C. Debris, mud and grease had been previously removed from the core.

Six core boxes were sequentially loaded into the scanner at a time. The operators were able to scan approximately 800 m of core per day in 18 m core batches. The PK team had to calibrate the scanner before each shift. The primary data could be seen in real time on site, but the data was taken back to the PK main office for further processing. The data was processed into high resolution .jpg files and as semi-quantitative data on Excel spreadsheets. The latter was used as an aid for stratigraphic logging and compiling 3D visual software.

9.4.3. Ground Magnetometer Survey

Few ground magnetometer surveys had been done prior BlackRock's acquisition of the Property. They were discontinuous and incomplete and BlackRock did not integrate them into its data. As mentioned, it has used it to improved geological interpretation (Section 7.6.2).

BlackRock has not carried out any ground magnetometer survey.

9.5. Trenches and Channel Sampling

During the 2009 field season, three trenches totalling 650 m, were excavated in the Southwest deposit. They added to the original 1997 McKenzie Bay's seven trenches (Figure 9-3). These old trenches were cleaned over a width of 1-2 m, re-channeled and assayed. Previous assay results were not used. All trenches covered only the magnetite mineralized areas.

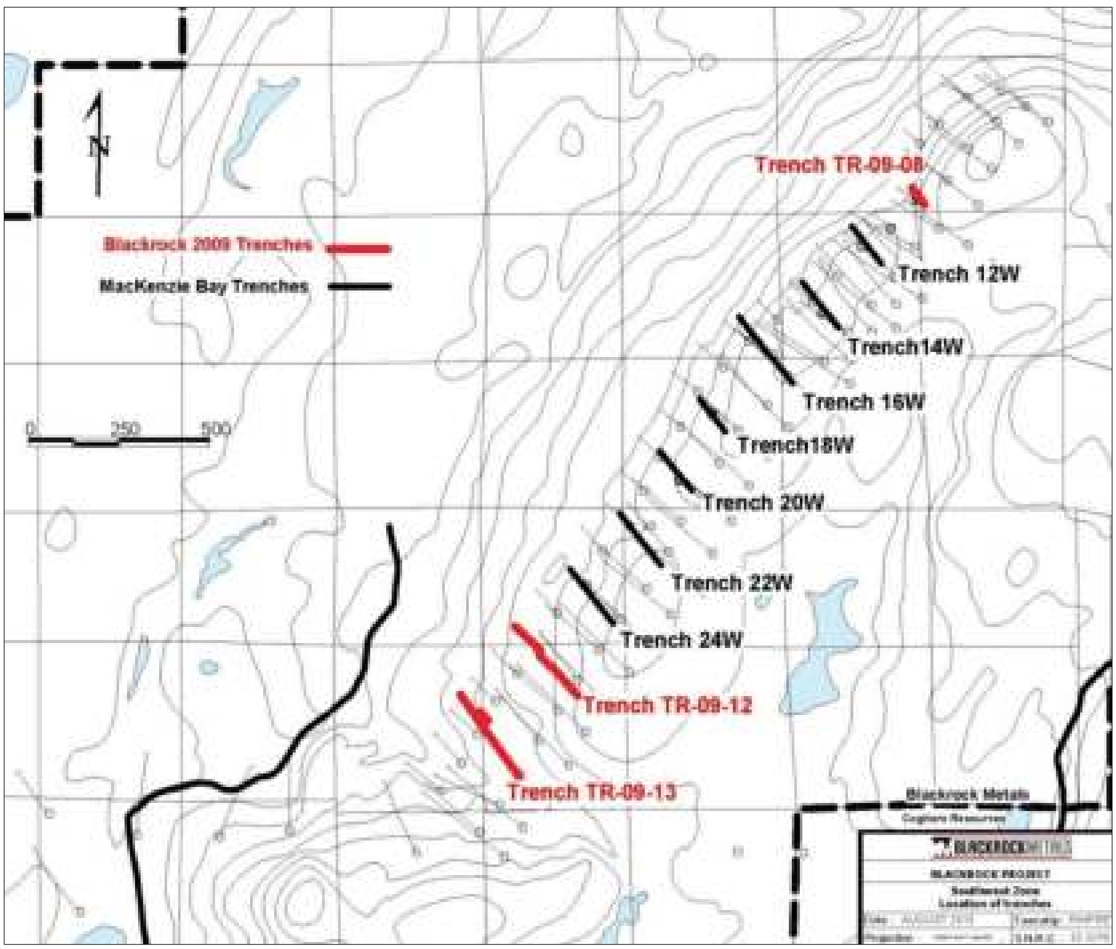


Figure 9-3: Location of BlackRock and McKenzie Bay trenches

In all trenches, channels consisted of two shallow parallel cut lines. Channels were done perpendicular through the mineralized zones and ranged from 60 to 240 metres in length. Channel sampling was performed using a rock saw and samples were collected using a rock hammer and a chisel. Sample dimensions were 3 m x 5 cm x 5 cm. Trench samples were labeled with distance from the origin, and distance off the baseline reference. No independent sampling was performed in the trenches.

Overall, 544 samples were collected for a total of 1,638 m of sampling. 357 channel samples were collected from McKenzie Bay trenches and 187 channel samples from new BlackRock ones.

All trenches and samples were surveyed by Paul Roy, professional surveyor. There were 675 channels surveyed, totalling 1,450 survey points. Both ends of all channels were surveyed with some locally intermediate points. The X-Y-Z coordinates were measured using real-time with a Leica GNSS GS15 receiver (sub-centimetre accuracy). Survey points were in the UTM coordinate system NAD83, Zone 18.

Figure 9-4: Typical channel in a magnetite-rich mineralized horizon

9.5.1. Trench Sample Preparation

All samples were prepared at the TJCM preparation laboratory in Chibougamau. Entire samples were crushed to greater than 70% passing ten mesh, split off up to 250 g, pulverized to greater than 85% passing 200 mesh and homogenized. One hundred-gram portion of the pulp was processed by ALS Chemex in Vancouver for multi-element analysis, whereas the remainder of the pulp was stored at the TJCM facility.

All samples were assayed for major oxides (WRA) including total iron, titanium dioxide, vanadium, sulphur, phosphorous, alumina and silica. Assays were performed by lithium metaborate fusion to dissolve resistive minerals followed by X-Ray fluorescence spectrometry (ME XRF11 package) for characterization of iron ores. BlackRock implemented a quality control program and 19 blanks and 10 duplicates were introduced with the channel samples for assay. No rock witness samples were kept.

The trench assay data was a guide to determine the stratigraphy of the Layered Zone of the Lake Dore Complex and develop the stratigraphic units for the resource model. On the other hand, the assays were not used in the current resource study because new drill sampling designed to allow projection of mineralization from surface was deemed to be more consistent and less prone to potential bias.

9.6. Petrology, Mineralogy and Research Studies

The mineralogy of the deposit has been examined by Corem, Panzhihua (PAN), Photonic Knowledge (PK) and IOS Services Géoscientifiques inc. (IOS). Corem and Panzhihua had examined crushed Run of Life (ROM) samples in terms of mineral liberation. PK and IOS had looked at whole rock samples.

PK analyzed 20 polished sections with a Scanning Electron Microscopy (SEM) and probe work to determine their "library" for hyperspectral scans. The scans covered the length and breadth of the deposits. This chapter is a summary of the findings of the above.

9.6.1. Previous IOS Study

In 2001, IOS had provided the ore petrography and 660 microprobe analyses for McKenzie Bay, the previous owner of the project. McKenzie Bay was focused on vanadium recovery. The bulk of the analyses was taken along strike to the north-east, beyond the BlackRock Project. The report does confirm continuation of geochemical signatures found in the Armitage and the Southwest Deposits of BlackRock, i.e. rock unit WRA signatures are confirmed over a total of 13 km of strike examined so far. For example:

- Vanadium grades in titanomagnetite steadily decrease from the basal P0 to the top P3 layers. (Geology section - Basal Chrome Series to the Titan Cap).

9.6.2. Ore Mineralogy

The main oxide ore minerals are titano-magnetite, titano-magnetite with ilmenite exsolution, ilmenite and minor magnetite. There is some substitution of Vanadium (V) for Iron (Fe) in both the titano-magnetite and the ilmenite. The V is more strongly partitioned to the titano-magnetite with greater than 1% concentrations commonly occurring.

Concentration in the ilmenite phase is generally in the 0.2-0.4% range. The titano-magnetite contains between 3-6% Ti substitutions for Fe in the crystal lattice. Exsolved ilmenite appears in the titano-magnetite as thin lamellae and as blebs.

The IOS report (Girard, 2001; p. 40) categorizes ilmenite into four habits – α, β, ɣ, δ – and suggests that these habits may be the result of “different moments in the crystallization process and different re-equilibrium reactions between the different oxides”.

The size range of the exsolved ilmenite ranges from <1 µm to > 70 µm. Ilmenite is also found in euhedral contact with titano-magnetite and as discrete crystals in the gangue (chlorite) ground mass.

Fine relict ilmenite (1< µm-10 µm) is frequently found in chlorite filled fractures, with the titano-magnetite replaced by iron silicates. IOS noted relict exsolved ilmenite lamellae in chlorite pseudomorphs of titano-magnetite.

Alteration of titano-magnetite to silicate and fine relict ilmenite are probably the major sources of unrecoverable iron and titanium. Minor amounts of sphene may account for other losses of TiO_2.

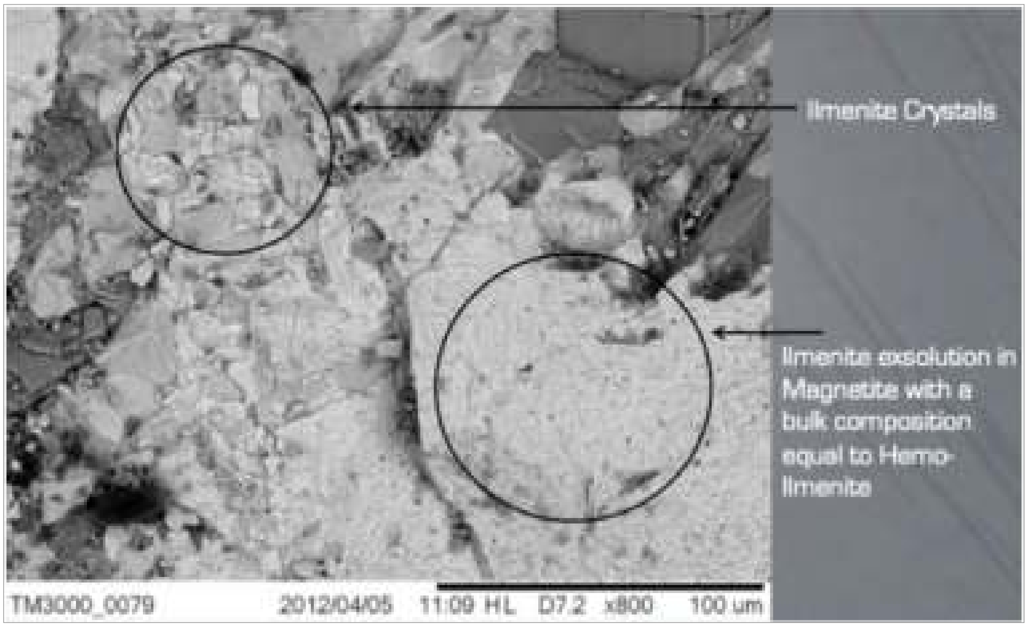


Figure 9-5: Ilmenite adjacent to magnetite with fine exsolved ilmenite (PK)

Figure 9-6: Two forms of ilmenite exsolution, lamellae (smaller crystal on left) and blobs (large crystal) in titano-magnetite

Note the preferential irregular replacement of titano-magnetite by silicates causing the pock-marked appearance of the titano-magnetite (Pan).

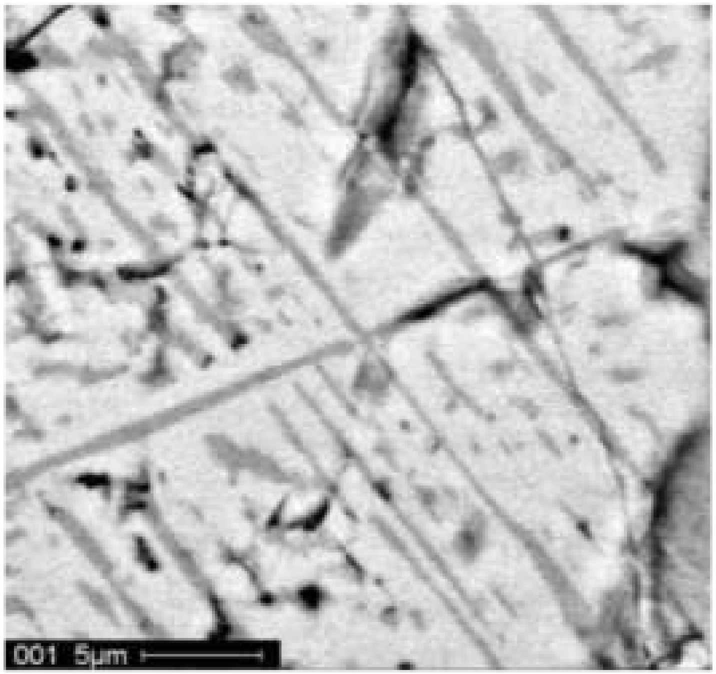


Figure 9-7: Very fine regular exsolved ilmenite lamellae arranged on the 111 crystal axes, with irregular exsolved ilmenite in titano-magnetite (PK)

Gangue minerals are chlorite, altered anorthite and epidote. Minor amounts of massive (non-fibrous) actinolite and iron rich species of hornblende have been noted. Calcite is common often in vein material sometimes associated with quartz and epidote. Sulphides in accessory quantities are pyrite, pyrrhotite, chalcopyrite and sphalerite. Trace amounts of albite and sphene have been noted in some mineralogical analyses (PK and IOS).

Most of the gangue minerals pseudomorph the original mineral fabric so, for example, in the original gabbro, pyroxene is replaced by chlorite and actinolite, plagioclase is heavily saussuritized or converted to epidote in part (a reflection of the green schist facies).

The epidote is iron rich and paramagnetic, and is also picked up with ilmenite in magnetic separation, requiring the use of gravity separation and flotation to remove it from the ilmenite concentrate.

Crystal boundaries between the oxide minerals and the gangue minerals can be very irregular, particularly between titano-magnetite, magnetite and chlorite. In the IOS study, chlorite was observed with a decussate structure in magnetite. In fractures and shear zones titano-magnetite/magnetite is absent, being replaced by chlorite or other iron silicates leaving fine relict ilmenite in part. Magnetite without ilmenite may appear in calcite veining, sometimes with pyrrhotite suggesting remobilization.

Surface exposures can show traces of rutile, anatase, hematite, and goethite due to late-stage oxidation effects of surface weathering.

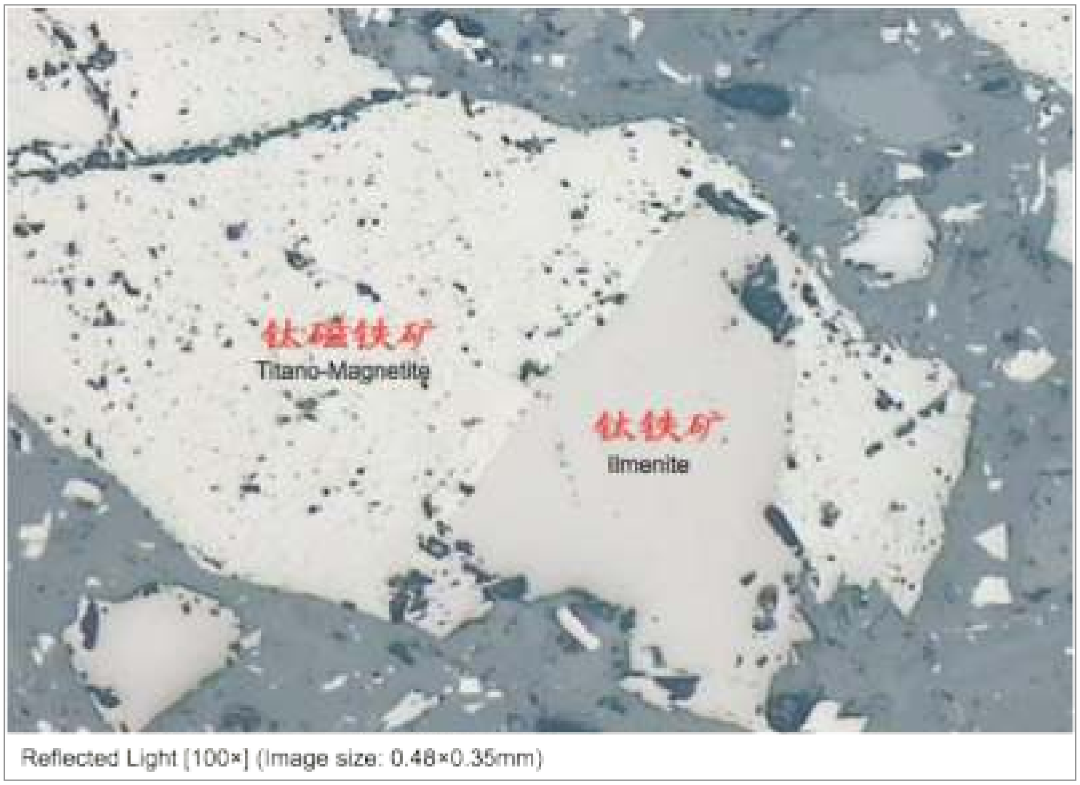


Figure 9-8: Euhedral ilmenite with titano-magnetite (pock marked) (Pan)

Figure 9-9: Ilmenite in chlorite gangue (PK)

Figure 9-10: Ilmenite particle

Note the clean, relatively unaltered ilmenite compared to magnetite (Pan).

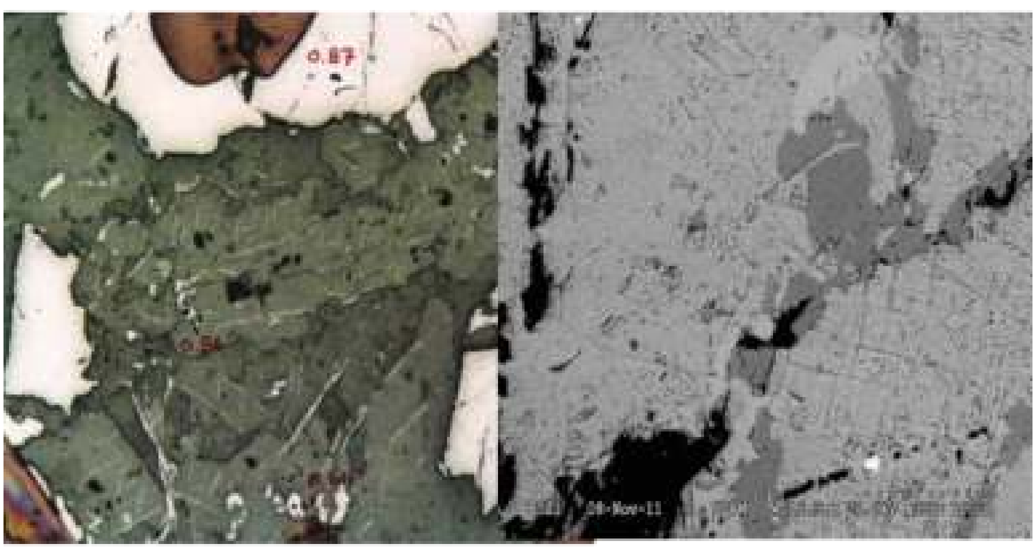


Figure 9-11: Left side chlorite pseudomorphing titano-magnetite

Note Relict Ilmenite Lamellae (IOS). Right Side, Unaltered Titano-Magnetite with Ilmenite Lamellae (PK)

Figure 9-12: Decussate Structure of Chlorite (dirty green) along Alteration Fractures in Magnetite (pale grey).

Note Residual Ilmenite in the Chlorite (IOS).

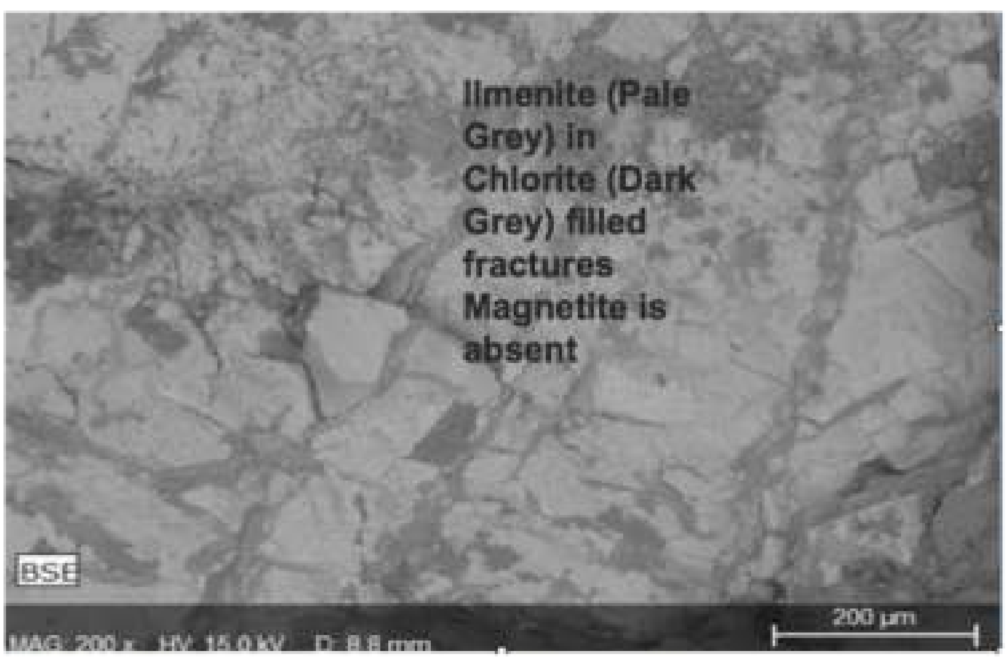


Figure 9-13: Alteration of magnetite along fractures (PK)

9.7. Exploration Potential

Finally, note the gap in drilling between 2800 and 3200. This area was not drilled because the mineralization was thought to be too narrow to be economic. In light of the pinch and swell pattern displayed in the rest of the deposit, this area becomes prospective. This area is also coincident with an increase in width and intensity of the ground magnetic signature that was not available at the time this area was drilled.

Similarly, the increase of thickness west of AE-1200 makes the next 3 km of magnetic anomaly stretching to the west prospective for containing similar quality resources.

10. Drilling

10.1. Historical Drilling

Thirteen holes were previously drilled on the Property, two by the MRN in 1970, and 11 by McKenzie Bay in 2001, for a total of 1,923 m (Section 6.2).

As previously mentioned, BlackRock secured the McKenzie Bay core in a secure site. None of the MRN core has been recovered.

10.2. BlackRock Drilling Campaigns

From 2010 to 2012, BlackRock completed three drilling programs over the Southwest Pit and two on the Armitage Pit, in all 219-diamond drillholes, totalling 49,680 m, were drilled (Table 10-1). Six holes were drilled on the neighbouring Cogitore Resources Inc. property as part of a condemnation drilling program for waste facilities, but they are not included in these statistics. All drilling was performed at the NQ size, except metallurgical (bulk) drillholes.

Table 10-1: Summary of BlackRock drilling on the BlackRock Property

Southwest Deposit		No. Holes	Meterage
	Resource drilling	83	20,534
	Metallurgical (Bulk sample, HQ Size)	20	2,985
	Geotechnical	8	1,833
	Condemnation	6	1,740
		117	**27,092**
Armitage Deposit		**No. Holes**	**Meterage**
	Resource drilling	81	19,573
	Metallurgical (Bulk sample, HQ Size)	21	3,015
		102	**22,588**
Grand Total	**Armitage and Southwest**	**219**	**49,680**
	Resource Drilling	164	40,107
	Metallurgical (Bulk)	41	6,000
Extra Drilling	Cogitore Resources Inc. property	6	1,644.5

10.2.1. Southwest Deposit Drilling Campaigns

The 2010 drilling program over the Southwest Deposit (Figure 10-1; Table 10-2) consisted of 49 NQ drillholes totalling 12,429 m under the supervision of Mr. Pierre O'Dowd, P.GEO. The program consisted of two holes per section spaced at 200 m (14 sections, 100N to 2600N).

The 2011 program (Table 10-3), under the supervision of Mr. Charles Perry P.GEO., consisted of in-fill drilling at 100 m spacing and a third hole on every section from 1600N to 2600N. Holes SW-17+50-01, SW-18+50-01, SW-19+50-01, SW-20+50-01 were also added, at a 50 m standoff between sections 1700N and 2100N.

Finally, hole SW-01-03 was drilled during the 2012 drilling program, because the drill site was not accessible in the previous drill program (Table 10-2).

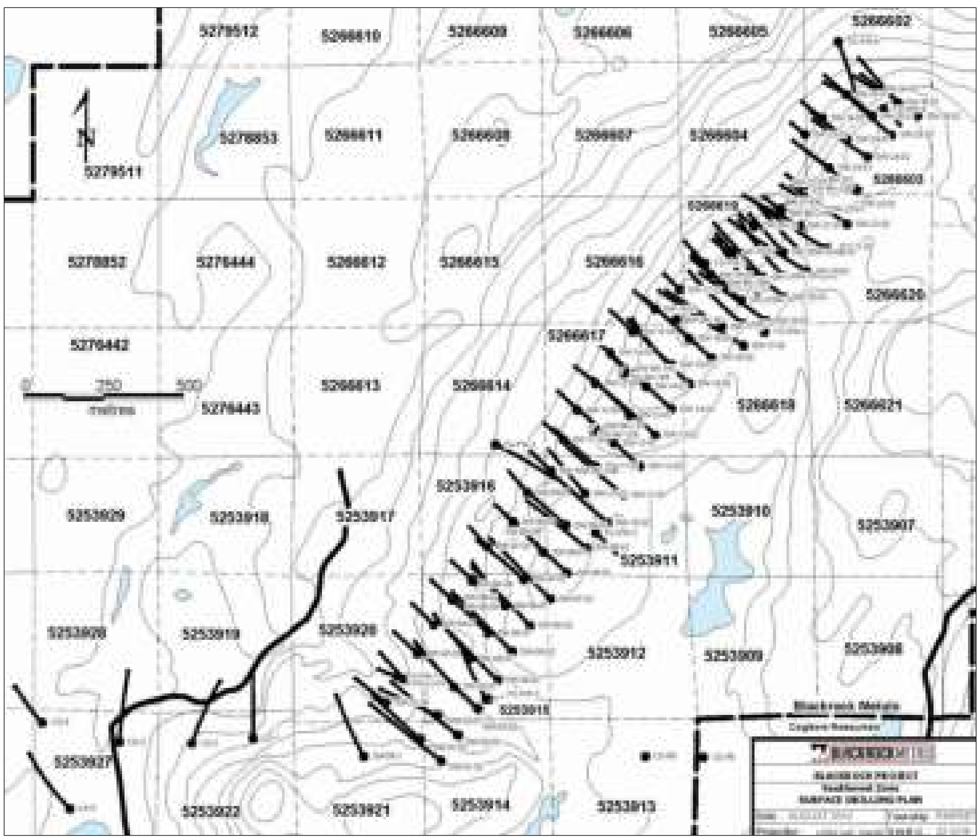


Figure 10-1: Location of diamond drillholes on the Southwest Deposit

Table 10-2: 2010 Drilling program – Southwest Deposit

DDH	Start Date	End Date	EASTING UTM NAD83 Zone 18	NORTHING UTM NAD83 Zone 18	Casing Elevation (m)	Azimuth	Plunge	Length (m)
SW-01-01	2010-02-06	2010-02-08	568497.30	5516795.60	462.47	311.35	-46.00	285
SW-01-02	2010-02-08	2010-02-11	568566.00	5516729.30	469.52	305.51	-46.00	387
SW-02-01	2010-02-11	2010-02-13	568550.80	5516870.90	449.82	293.01	-44.00	273
SW-02-02	2010-02-13	2010-02-16	568615.80	5516809.80	455.09	306.35	-47.46	378
SW-04-01	2010-02-01	2010-02-03	568655.10	5517049.60	466.93	318.18	-46.06	255
SW-04-02	2010-02-03	2010-02-06	568739.76	5516983.00	472.71	313.01	-45.00	366
SW-06-01	2010-01-27	2010-01-29	568760.40	5517215.30	479.47	321.18	-46.00	243
SW-06-02	2010-01-29	2010-02-01	568839.80	5517154.50	487.75	315.68	-45.50	354
SW-08-01	2010-01-23	2010-01-25	568878.60	5517383.40	490.78	313.68	-45.00	252
SW-08-02	2010-01-25	2010-01-27	568954.20	5517319.70	495.00	311.18	-44.97	351
SW-10-01	2010-01-21	2010-01-23	569006.50	5517569.30	487.51	318.35	-46.00	258
SW-10-02	2010-01-18	2010-01-21	569078.10	5517479.30	490.67	309.68	-45.00	327
SW-12-01	2010-01-16	2010-01-18	569095.00	5517727.30	492.35	312.18	-44.69	231
SW-12-02	2010-01-15	2010-01-16	569177.10	5517656.10	486.86	311.51	-43.81	336
SW-14-01	2010-01-14	2010-01-15	569194.34	5517905.00	502.05	309.85	-43.00	195
SW-14-02	2010-01-12	2010-01-14	569275.30	5517836.70	497.70	310.68	-42.28	336
SW-16-01	2010-02-16	2010-02-18	569327.70	5518057.10	511.63	311.35	-45.34	270
SW-16-02	2010-02-18	2010-02-21	569397.80	5517997.80	501.97	309.01	-45.00	381
SW-16-03	2010-03-10	2010-03-11	569219.60	5518148.20	500.27	308.01	-51.00	135
SW-17-01	2010-02-26	2010-06-29	569429.40	5518094.80	509.62	301.51	-46.00	297
SW-17-02	2010-02-28	2010-06-30	569498.30	5518036.70	497.55	299.85	-45.50	444
SW-17-03	2010-03-12	2010-03-12	569295.20	5518209.50	506.50	314.68	-51.00	108
SW-18-01	2010-02-21	2010-02-23	569489.20	5518178.40	509.03	310.18	-44.50	291
SW-18-02	2010-02-23	2010-02-26	569568.10	5518115.60	500.87	304.18	-46.00	369
SW-18-03	2010-03-12	2010-03-13	569392.00	5518257.10	515.10	308.35	-51.00	126
SW-19-01	2010-02-23	2010-02-25	569555.30	5518251.70	507.12	316.35	-43.50	312
SW-19-02	2010-02-25	2010-03-01	569623.30	5518190.50	502.50	307.18	-45.00	405
SW-19-03	2010-03-13	2010-03-13	569460.40	5518330.50	513.52	315.01	-50.00	114
SW-20-01	2010-02-19	2010-02-20	569619.20	5518329.40	508.02	317.35	-45.00	219
SW-20-02	2010-02-20	2010-02-23	569693.70	5518272.50	506.17	298.51	-45.00	405
SW-20-03	2010-03-12	2010-03-12	569525.40	5518408.80	515.22	309.01	-50.00	105
SW-21-01	2010-03-01	2010-03-02	569669.60	5518413.70	507.42	295.85	-44.50	276
SW-21-02	2010-03-03	2010-03-06	569750.70	5518344.30	512.78	300.01	-45.50	336
SW-21-03	2010-03-11	2010-03-12	569606.80	5518464.60	509.07	305.18	-51.00	105
SW-22-01	2010-02-16	2010-02-17	569736.40	5518464.20	522.10	299.85	-47.00	207
SW-22-02	2010-02-17	2010-02-19	569818.40	5518420.80	515.58	303.35	-44.00	291
SW-22-03	2010-03-10	2010-03-11	569671.30	5518542.20	505.80	302.68	-56.00	102

DDH	Start Date	End Date	EASTING UTM NAD83 Zone 18	NORTHING UTM NAD83 Zone 18	Casing Elevation (m)	Azimuth	Plunge	Length (m)
SW-23-01	2010-03-06	2010-03-08	569759.70	5518595.90	522.75	305.35	-46.00	204
SW-23-02	2010-03-08	2010-03-10	569844.60	5518529.80	525.04	306.18	-45.50	285
SW-23-03	2010-03-09	2010-03-10	569681.80	5518698.70	486.30	309.01	-56.00	102
SW-24-01	2010-02-14	2010-02-16	569814.40	5518685.40	527.75	306.51	-46.50	204
SW-24-02	2010-02-12	2010-02-14	569879.60	5518631.20	530.19	301.51	-45.50	288
SW-24-03	2010-03-08	2010-03-09	569736.50	5518750.30	489.37	307.85	-46.50	111
SW-25-01	2010-03-03	2010-03-04	569893.80	5518759.80	525.37	310.85	-47.50	243
SW-25-02	2010-03-04	2010-03-07	569952.90	5518699.50	527.40	310.68	-45.00	369
SW-25-03	2010-03-08	2010-03-08	569810.50	5518826.10	490.42	297.35	-45.00	108
SW-26-01	2010-02-06	2010-02-10	569970.70	5518802.40	517.29	317.35	-44.50	258
SW-26-02	2010-02-10	2010-02-12	570035.50	5518761.90	516.04	304.35	-45.00	300
SW-26-03	2010-03-07	2010-03-07	569901.90	5518842.50	501.55	298.35	-46.00	114

Table 10-3: 2011-2012 Drilling Program – Southwest Deposit

DDH	Start Date	End Date	EASTING UTM NAD83 Zone 18	NORTHING UTM NAD83 Zone 18	Casing Elevation (m)	Azimuth	Plunge	Length (m)
SW-01-03	2012-02-07	2012-02-08	568389.80	5516907.80	440.76	307.18	-45.00	174
SW-02-03	2011-12-14	2011-12-15	568444.90	5516983.10	441.82	316.51	-45.00	156
SW-03-01	2011-12-11	2011-12-13	568603.80	5516959.80	456.10	315.18	-46.17	273
SW-03-02	2011-12-07	2011-12-11	568677.90	5516895.00	455.14	311.51	-45.30	369
SW-03-03	2011-12-13	2011-12-14	568487.40	5517062.40	447.38	312.35	-45.49	144
SW-04-03	2011-12-06	2011-12-07	568536.90	5517153.70	457.17	313.85	-45.67	132
SW-05-01	2011-12-04	2011-12-06	568708.40	5517130.20	473.26	312.01	-45.91	258
SW-05-02	2011-12-01	2011-12-04	568784.80	5517072.80	480.93	313.18	-45.74	375
SW-05-03	2011-12-06	2011-12-06	568594.00	5517231.00	465.74	312.18	-45.66	129
SW-06-03	2011-11-30	2011-12-01	568655.20	5517296.70	470.79	310.68	-42.94	117
SW-07-01	2011-11-25	2011-11-26	568817.50	5517298.00	487.54	311.01	-44.98	264
SW-07-02	2011-11-26	2011-11-29	568902.20	5517238.60	491.48	313.01	-44.85	360
SW-07-03	2011-11-29	2011-11-30	568702.90	5517398.70	472.41	315.68	-46.70	123
SW-08-03	2011-11-24	2011-11-25	568782.70	5517478.50	476.23	310.35	-45.10	111
SW-09-01	2011-11-20	2011-11-22	568947.60	5517469.30	496.80	309.18	-44.95	263
SW-09-02	2011-11-22	2011-11-24	569013.20	5517399.00	490.60	307.68	-44.60	330
SW-09-03	2011-11-19	2011-11-20	568825.10	5517569.70	477.46	318.01	-43.59	123
SW-10-03	2011-11-14	2011-11-15	568899.30	5517638.90	477.30	310.68	-45.45	129
SW-11-01	2011-11-15	2011-11-17	569040.10	5517654.90	484.12	309.35	-44.49	243
SW-11-02	2011-11-17	2011-11-19	569127.70	5517566.80	482.83	306.35	-45.55	333
SW-11-03	2011-11-13	2011-11-14	568937.50	5517736.00	482.81	308.35	-44.08	114
SW-12-03	2011-11-12	2011-11-15	568979.40	5517828.70	483.89	313.85	-44.93	123
SW-13-01	2011-11-07	2011-11-09	569143.50	5517814.50	496.80	313.68	-44.76	228
SW-13-02	2011-11-09	2011-11-12	569225.30	5517753.10	493.43	313.51	-44.85	345

DDH	Start Date	End Date	EASTING UTM NAD83 Zone 18	NORTHING UTM NAD83 Zone 18	Casing Elevation (m)	Azimuth	Plunge	Length (m)
SW-13-03	2011-11-05	2011-11-06	569035.00	5517913.60	486.55	309.68	-42.76	114
SW-14-03	2011-11-04	2011-11-05	569080.70	5518008.90	486.53	309.35	-45.21	132
SW-15-01	2011-03-28	2011-03-30	569273.70	5517975.20	505.81	307.51	-44.92	285
SW-15-02	2011-03-30	2011-04-02	569329.00	5517916.40	500.07	309.68	-44.26	387
SW-15-03	2011-11-03	2011-11-04	569150.00	5518076.40	494.85	312.51	-45.29	138
SW-17+50-01	2011-03-06	2011-03-08	569496.10	5518104.80	501.10	308.68	-44.09	318
SW-18+50-01	2011-03-02	2011-03-05	569557.40	5518181.20	503.77	313.51	-41.56	327
SW-19+50-01	2011-02-27	2011-03-02	569623.10	5518255.60	504.71	311.85	-43.90	309
SW-20+50-01	2011-02-23	2011-02-26	569677.80	5518330.90	509.03	310.35	-44.79	294
SW-BK-06	2012-04-03	2012-04-05	568655.70	5517291.30	470.69	0	-90.00	147
SW-BK-08	2012-04-02	2012-04-03	568782.20	5517471.40	476.46	0	-90.00	144
SW-BK-10A	2012-03-30	2012-03-31	568954.70	5517638.50	480.08	0	-90.00	150
SW-BK-10B	2012-03-31	2012-04-02	568955.50	5517638.40	480.06	0	-90.00	150
SW-BK-12A	2012-03-27	2012-03-28	569034.00	5517767.00	491.00	0	-90.00	150
SW-BK-12B	2012-03-28	2012-03-30	569032.00	5517764.00	491.00	0	-90.00	150
SW-BK-12C	2012-04-05	2012-04-06	569036.00	5517770.00	491.00	0	-90.00	150
SW-BK-14A	2012-03-26	2012-03-27	569122.00	5517949.00	495.00	0	-90.00	150
SW-BK-14B	2012-04-06	2012-04-08	569120.00	5517946.00	495.00	0	-90.00	147
SW-BK-14C	2012-04-08	2012-04-09	569124.00	5517952.00	495.00	0	-90.00	150
SW-BK-16	2012-03-24	2012-03-26	569287.00	5518116.00	508.00	0	-90.00	150
SW-BK-18A	2012-03-23	2012-03-24	569419.00	5518230.00	512.00	0	-90.00	150
SW-BK-18B	2012-04-09	2012-04-10	569417.00	5518227.00	512.00	0	-90.00	150
SW-BK-18C	2012-04-10	2012-04-11	569421.00	5518233.00	512.00	0	-90.00	150
SW-BK-20	2012-03-21	2012-03-23	569526.00	5518414.00	515.00	0	-90.00	150
SW-BK-21A	2012-03-20	2012-03-21	569607.00	5518465.00	510.00	0	-90.00	147
SW-BK-21B	2012-04-11	2012-04-13	569605.00	5518462.00	510.00	0	-90.00	150
SW-BK-22A	2012-03-19	2012-03-20	569676.00	5518543.00	506.00	0	-90.00	150
SW-BK-22B	2012-04-13	2012-04-14	569672.00	5518540.00	509.00	0	-90.00	150
SW-BK-22C	2012-04-14	2012-04-15	569678.00	5518546.00	509.00	0	-90.00	150
SWSP-1	2012-02-08	2012-02-10	568323.10	5516738.30	470.13	329.85	-45.81	303
TG-FW-1	2011-03-25	2011-03-27	568425.90	5516990.40	439.87	284.18	-70.64	189
TG-FW-2	2011-03-18	2011-03-21	568723.20	5517717.70	460.58	114.02	-43.10	285
TG-FW-3	2011-03-11	2011-03-13	569374.10	5518292.20	508.68	312.68	-69.82	261
TG-FW-4	2011-03-14	2011-03-16	569782.70	5518990.40	470.19	156.52	-42.62	222
TG-HW-1	2011-03-23	2011-03-24	568705.50	5516921.80	460.11	326.35	-46.14	273
TG-HW-2	2011-03-21	2011-03-22	569042.10	5517446.90	486.62	131.02	-54.48	159
TG-HW-3	2011-03-08	2011-03-10	569563.60	5518079.90	498.99	315.18	-84.63	240
TG-HW-4	2011-03-16	2011-03-17	569928.30	5518782.30	519.50	126.18	-69.10	204
CD-1	2012-02-11	2012-02-14	567975.00	5516791.60	429.63	358.18	-45.00	309
CD-2	2012-02-14	2012-02-16	567782.90	5516772.80	430.58	17.18	-42.61	300
CD-3	2012-02-17	2012-02-20	567556.00	5516774.70	414.47	360.35	-44.51	306
CD-4	2012-02-20	2012-02-21	567313.40	5516831.60	414.76	317.18	-46.92	210
CD-5	2012-02-21	2012-02-23	567402.20	5516564.00	414.09	314.18	-42.22	309
CD-6	2012-02-23	2012-02-25	567152.10	5516180.30	427.92	310.51	-44.11	306

In the same campaign, 20 HQ size holes were drilled vertically to a depth of 150 m through the mineralized zones of the Southwest Deposit to obtain bulk samples for metallurgical testing at COREM. These vertical holes were not checked for deviation by the Deviflex, and they were not sampled on individual 3 m composites. All mineralized material was comingled into the ≈20 tonne bulk sample that was used for testing and optimization in a pilot plant.

Details for the six condemnation drillholes, CD-1 to CD-6, can be found in Section 10.11 of this report.

10.2.2. Armitage Deposit Drilling Campaigns

The 2010 drilling program on the Armitage Pit, supervised by Mr. Pierre O'Dowd, P.Geo., totalled 28 drillholes for 8,093 m (Figure 10-2; Table 10-4). The drilling program consisted of two holes per section spaced by 200 m (14 sections) plus one section at 400 m distance.

Subsequently in 2012, under the supervision of Mr. Charles Perry P.Geo., infill drilling spaced the holes at 100 m, with the addition of a third hole to each section.

Twenty HQ diameter holes were drilled vertically to a depth of 150 m to collect bulk samples for metallurgical testing at COREM. These holes were not checked for deviation by the Deviflex instrument.

All boreholes were drilled on NQ size, except for the metallurgical (bulk sampling) boreholes, which were on HQ core size.

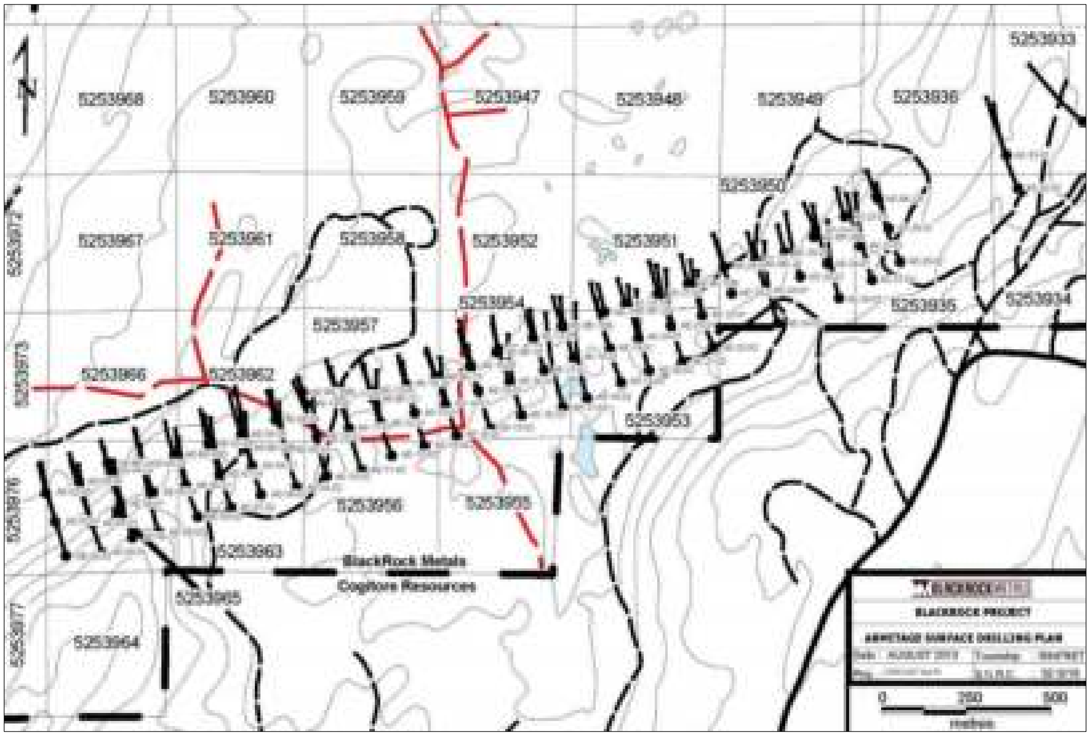


Figure 10-2: Location of diamond drillholes on the Armitage Deposit

Table 10-4: 2010 Drilling Program – Armitage Deposit

DDH	Start Date	End Date	EASTING UTM NAD83 Zone 18	NORTHING UTM NAD83 Zone 18	Casing Elevation (m)	Azimuth	Plunge	Length (m)
AE-04-01	2010-08-03	2010-08-05	45.00	5515025.80	438.80	334.51	-44.89	234
AE-04-02	2010-08-06	2010-08-08	564365.40	5514925.70	459.56	337.51	-45.69	327
AE-06-01	2010-08-14	2010-08-15	564520.70	5515081.30	443.50	341.01	-46.93	234
AE-06-02	2010-08-15	2010-08-18	564557.70	5514988.80	443.14	335.51	-45.26	333
AE-08-01	2010-07-30	2010-08-01	564716.00	5515140.90	435.55	338.01	-46.49	240
AE-08-02	2010-08-01	2010-08-03	564743.00	5515056.20	434.69	338.18	-45.43	339
AE-10-01	2010-08-09	2010-08-10	564905.10	5515228.00	433.42	335.85	-45.00	249
AE-10-02	2010-08-10	2010-08-13	564935.40	5515113.40	431.73	340.18	-46.77	351
AE-12-01	2010-07-26	2010-07-27	565093.00	5515271.30	429.39	338.51	-45.34	258
AE-12-02	2010-07-27	2010-07-29	565126.90	5515175.30	424.94	336.34	-45.27	351
AE-14-01	2010-08-18	2010-08-20	565283.80	5515326.80	422.98	340.01	-45.00	249
AE-14-02	2010-08-20	2010-08-23	565319.00	5515236.10	419.43	337.34	-46.26	351
AE-16-01	2010-07-21	2010-07-23	565471.30	5515397.50	418.82	337.18	-44.09	249
AE-16-02	2010-07-23	2010-07-26	565508.90	5515298.40	416.83	340.18	-45.30	357
AE-18-01	2010-08-23	2010-08-25	565659.20	5515469.50	415.20	337.85	-44.85	252
AE-18-02	2010-08-25	2010-08-28	565700.20	5515351.90	415.86	341.68	-42.11	363
AE-20-01	2010-07-17	2010-07-19	565849.70	5515530.40	419.64	340.85	-47.43	231
AE-20-02	2010-07-19	2010-07-21	565883.30	5515434.30	420.07	339.18	-45.59	342
AE-22-01	2010-08-28	2010-08-29	566037.60	5515598.30	425.67	340.18	-45.47	231
AE-22-02	2010-08-30	2010-09-01	566071.20	5515505.00	429.38	336.18	-45.57	324
AE-24-01	2010-07-13	2010-07-14	566224.00	5515670.60	429.90	340.18	-46.92	213
AE-24-02	2010-07-14	2010-07-17	566261.50	5515577.40	431.44	338.51	-44.76	339
AE-26-02	2010-09-01	2010-09-03	566440.60	5515653.50	434.50	340.68	-46.11	324
AE-26-01	2010-09-04	2010-09-05	566405.50	5515755.40	434.56	339.01	-44.00	240
AE-28-01	2010-07-08	2010-07-10	566584.50	5515859.90	438.50	340.51	-46.65	189
AE-28-02	2010-07-10	2010-07-13	566619.30	5515762.50	434.01	334.51	-44.81	317
AE-32-01	2010-07-04	2010-07-06	566928.70	5516079.10	436.06	339.18	-45.95	252
AE-32-02	2010-07-06	2010-07-08	566964.20	5515982.10	437.95	337.85	-44.76	354

Table 10-5: 2012 Drilling Program – Armitage Deposit

DDH	Start Date	End Date	EASTING UTM NAD83 Zone 18	NORTHING UTM NAD83 Zone 18	Casing Elevation (m)	Azimuth	Plunge	Length (m)
AE-02-01	2012-06-09	2012-06-10	564136.50	5514969.20	425.30	342.85	-45.66	255
AE-02-02	2012-06-10	2012-06-13	564169.30	5514869.90	442.82	342.51	-45.76	354
AE-02-03	2012-06-08	2012-06-09	564108.10	5515053.20	420.20	343.51	-45.18	126
AE-03-01	2012-06-04	2012-06-06	564247.40	5514964.30	441.86	336.68	-43.91	270
AE-03-02	2012-06-01	2012-06-04	564276.30	5514878.60	459.48	338.51	-43.23	369
AE-03-03	2012-06-07	2012-06-08	564205.10	5515090.00	416.86	338.68	-45.71	120
AE-04-03	2012-06-06	2012-06-07	564296.40	5515122.20	417.15	340.51	-45.38	129
AE-05-01	2012-05-28	2012-05-30	564424.40	5515059.70	440.51	338.18	-45.23	267
AE-05-02	2012-05-30	2012-06-01	564443.30	5514942.00	458.39	341.01	-44.76	366
AE-05-03	2012-05-27	2012-05-27	564394.10	5515142.40	436.48	342.01	-44.29	150
AE-06-03	2012-05-26	2012-05-27	564486.10	5515173.10	439.93	339.51	-46.17	143
AE-07-01	2012-05-22	2012-05-24	564619.60	5515115.70	439.80	339.01	-45.13	255
AE-07-02	2012-05-24	2012-05-26	564655.30	5515018.10	437.38	339.85	-44.89	375
AE-07-03	2012-05-21	2012-05-22	564585.90	5515202.30	436.54	342.85	-45.11	150
AE-08-03	2012-05-21	2012-05-21	564682.30	5515231.40	433.88	339.18	-43.93	171
AE-09-01	2012-05-16	2012-05-18	564816.20	5515190.00	436.28	339.01	-45.70	258
AE-09-02	2012-05-18	2012-05-20	564847.40	5515080.00	434.20	340	-44.95	357
AE-09-03	2012-05-16	2012-05-16	564782.60	5515277.20	430.30	340	-45.20	135
AE-10-03	2012-05-15	2012-05-15	564873.00	5515323.10	430.98	340.85	-44.47	117
AE-11-01	2012-05-10	2012-05-12	565000.40	5515254.20	431.14	339.01	-45.72	264
AE-11-02	2012-05-12	2012-05-14	565038.80	5515131.70	429.87	338.35	-45.14	357
AE-11-03	2012-05-08	2012-05-10	564965.00	5515352.50	430.13	338.51	-44.94	150
AE-12-03	2012-05-07	2012-05-08	565062.90	5515364.50	429.02	338.68	-45.48	138
AE-13-01	2012-05-03	2012-05-05	565193.40	5515302.40	427.22	339.35	-45.65	249
AE-13-02	2012-05-05	2012-05-07	565224.20	5515204.90	422.80	339.51	-43.73	354
AE-13-03	2012-05-02	2012-05-03	565167.30	5515384.20	426.09	338.85	-45.63	105
AE-14-03	2012-05-02	2012-05-02	565253.50	5515409.50	424.04	336.35	-45.00	111
AE-15-01	2012-04-27	2012-04-29	565380.40	5515369.70	419.04	339.01	-43.79	288
AE-15-02	2012-04-29	2012-05-02	565417.80	5515258.30	416.78	337.18	-45.90	357
AE-15-03	2012-04-26	2012-04-27	565354.60	5515440.30	421.81	340.01	-45.10	153
AE-16-03	2012-04-25	2012-04-26	565437.10	5515491.90	418.89	346.18	-45.98	141
AE-17-01	2012-04-24	2012-04-25	565573.40	5515431.20	416.20	340.51	-43.34	252
AE-17-02	2012-04-18	2012-04-21	565630.40	5515323.40	415.79	341.51	-44.44	384
AE-17-03	2012-04-23	2012-04-24	565539.20	5515526.00	418.20	340.35	-45.44	117

DDH	Start Date	End Date	EASTING UTM NAD83 Zone 18	NORTHING UTM NAD83 Zone 18	Casing Elevation (m)	Azimuth	Plunge	Length (m)
AE-18-03	2012-04-22	2012-04-23	565624.70	5515563.90	419.23	337.68	-44.48	123
AE-19-01	2012-03-16	2012-03-18	565767.20	5515496.80	417.74	338.85	-44.91	285
AE-19-02	2012-04-16	2012-04-18	565800.00	5515399.50	416.95	337.68	-44.50	354
AE-19-03	2012-04-21	2012-04-22	565749.00	5515604.50	414.81	342.51	-44.47	117
AE-20-03	2012-03-15	2012-03-16	565810.60	5515628.80	416.43	339.85	-47.13	114
AE-21-01	2012-03-11	2012-03-12	565945.10	5515566.30	422.86	340.35	-44.55	258
AE-21-02	2012-03-13	2012-03-15	565980.70	5515463.00	427.55	339.85	-44.52	381
AE-21-03	2012-03-10	2012-03-11	565905.50	5515660.10	420.08	341.18	-45.86	141
AE-22-03	2012-03-09	2012-03-10	565995.40	5515690.70	419.17	341.51	-44.54	123
AE-23-01	2012-03-07	2012-03-09	566124.20	5515663.60	426.45	341.01	-44.25	255
AE-23-03	2012-03-06	2012-03-07	566100.30	5515741.70	421.30	341.01	-44.95	114
AE-24-03	2012-03-06	2012-03-06	566191.40	5515765.50	423.60	341.51	-48.51	120
AE-25-01	2012-03-04	2012-03-05	566325.50	5515716.00	433.00	340.01	-44.51	270
AE-25-03	2012-03-04	2012-03-04	566278.80	5515787.30	429.00	340.35	-45.43	117
AE-26-03	2012-03-02	2012-03-03	566368.40	5515854.70	430.78	331.51	-44.46	114
AE-27-01	2012-02-28	2012-02-29	566501.10	5515807.30	436.40	339.51	-46.18	228
AE-27-02	2012-02-29	2012-03-02	566536.90	5515709.00	438.81	340.85	-44.60	345
AE-27-03	2012-02-27	2012-02-27	566453.70	5515893.70	432.04	339.85	-44.13	120
AE-28-03	2012-02-26	2012-02-27	566547.00	5515954.10	435.57	335.51	-45.73	114
AE-BK-04	2012-06-13	2012-06-14	564290.00	5515144.60	416.93	0	-90.00	153
AE-BK-06A	2012-06-14	2012-06-15	564486.60	5515171.00	439.97	0	-90.00	150
AE-BK-06B	2012-06-16	2012-06-17	564486.50	5515171.20	440.00	0	-90.00	147
AE-BK-06C	2012-06-17	2012-06-17	564486.20	5515171.90	439.91	0	-90.00	24
AE-BK-06D	2012-06-17	2012-06-18	564486.40	5515171.40	440.04	0	-90.00	150
AE-BK-08A	2012-06-18	2012-06-19	564690.10	5515193.60	433.98	0	-90.00	150
AE-BK-08B	2012-06-19	2012-06-20	564689.80	5515193.90	433.92	0	-90.00	150
AE-BK-10A	2012-06-21	2012-06-22	564860.00	5515358.00	430.00	0	-90.00	147
AE-BK-10B	2012-06-22	2012-06-23	564861.00	5515358.00	430.00	0	-90.00	150
AE-BK-12	2012-06-24	2012-06-25	565070.40	5515343.10	429.07	0	-90.00	150
AE-BK-14	2012-06-25	2012-06-26	565255.20	5515406.00	423.57	0	-90.00	150
AE-BK-16	2012-06-27	2012-06-28	565433.60	5515482.30	419.20	0	-90.00	150
AE-BK-18A	2012-06-28	2012-06-29	565625.00	5515561.00	419.00	0	-90.00	147
AE-BK-18B	2012-06-30	2012-07-02	565624.00	5515561.00	419.00	0	-90.00	150
AE-BK-20	2012-07-02	2012-07-03	565810.00	5515610.90	416.18	0	-90.00	147
AE-BK-22A	2012-07-03	2012-07-04	565995.30	5515690.00	418.84	0	-90.00	150
AE-BK-22B	2012-07-04	2012-07-05	565995.10	5515690.60	418.85	0	-90.00	150

DDH	Start Date	End Date	EASTING UTM NAD83 Zone 18	NORTHING UTM NAD83 Zone 18	Casing Elevation (m)	Azimuth	Plunge	Length (m)
AE-BK-24	2012-07-05	2012-07-06	566195.30	5515753.20	424.22	0	-90.00	150
AE-BK-26A	2012-07-06	2012-07-07	566364.10	5515832.90	430.64	0	-90.00	150
AE-BK-26B	2012-07-07	2012-07-08	566364.20	5515832.50	430.61	0	-90.00	150
AE-BK-28	2012-07-08	2012-07-09	566551.20	5515946.00	435.96	0	-90.00	150

10.3. Core Handling and Recovery

Core was retrieved from the core barrel and placed in sequential order into clean wooden core boxes. Efforts were made by the drillers to ensure that the core was clean and free of contaminants such as grease.

The core boxes were labeled with the drillhole ID (e.g. SW-10-02), core box number (e.g. Box 4) and core metre run (e.g. 20-23 m). Labeling was started on the left hand end of the box so that the down-hole depth is read from left to right. Each run, on average 3 m in length, was identified by a marker (wooden block) labeled with the depth. Any sections of unrecovered core were properly identified in the core box indicating the length of the missing section. On average, core recovery was 93%.

Towards the end of a drillhole, the geologist checked on-site the final core for mineralization to ensure drilling did not ended in magnetite-bearing mineralization. Completed core was taken to the core shack by BlackRock technicians for logging by the geologist.

10.4. Down-Hole Surveys

Upon completion of the drillhole, the deviation of the drillhole azimuth and depth was measured with Deviflex™, a non-magnetic gyroscopic electronic survey tool attached to the drill wireline. For the down-hole survey, the technician used the planned azimuth. Data from the Deviflex was downloaded in Word format and transferred to a computer from a USB key. The azimuth from Deviflex was then corrected to the true surveyed azimuth (10.5.1) to determine the final entry into the database. The data was reviewed and checked before copying and pasting to a spreadsheet.

Twelve holes were not surveyed with the Deviflex due to equipment malfunction, nine in the Southwest Pit, and three in the Armitage Pit.

The vertical holes drilled for metallurgical pilot work were not surveyed by the Deviflex instrument. These holes were not assayed on individual 3 m composite spacing and are not part of the database used for resource modelling.

10.5. Collar Surveys

Initially, hand-held GPS devices were sufficient to locate drilling sites but all were subsequently accurately positioned by a professional surveyor Mr. Paul Roy, arp.-geo., a.t.c. After drilling, all boreholes were subsequently surveyed, including azimuth and plunge.

10.5.1. Survey Methodology

Permanent control points #1, #203 and BR-9 were established by Mr. Paul Roy as locations for the survey base stations.

Once a drillhole was finished, the geologist placed a fluorescent orange picket or painted post next to the collar. The casing was left in the ground with a screw capping, on which the drillhole ID was engraved.

Borehole positions were measured by Global Positioning System (GPS) real time or Real Time Kinematic (RTK). Surveying stations were established in front of each casing to take measures at the base station; other stations were established at a sufficient distance from the first ones in order to serve as reference points to the total station.

The direction of each drillhole was determined by measuring two points on a 6 m long aluminum pipe of which 2 m was inserted in the casing. The aim was to precisely measure two points, as far apart as possible, in the vertical plane of the drillhole. Surveying was done by laser distancing with a total station from a local control point established during the RTK survey of the collar locations.

Notes to the Survey Data:

- All elevations are reported in reference to the mean sea level;
- The coordinates are reported for the drillhole collars at ground elevation;
- The collar coordinates refer to the UTM NAD83 Zone 18 grid that is based on the meridian 75o00' west for Zone 18; therefore 0o41' is to be added to obtain local astronomical azimuths or directions in regards to the local meridian, the project being on the approximate longitude of 74o02' west;
- The drillholes identified with the suffix "BK" are vertical drillholes and no azimuth was determined in these cases;

- The casing of borehole SW-10-03 was not found (stake only) and the one of bore hole SW-06-03 was too damaged to allow for measurement of its azimuth;
- The coordinates of the base stations 1 and 203 differ by a few centimetres from the ones used previously due to the introduction in the summer of 2011 of control points BR-1 to BR-9, which were attached directly to the geodesic grid.

10.5.2. Survey Instrumentation

All final survey coordinates and angles were surveyed using state of the art equipment. Collar and trench locations were surveyed using RTK GPS equipment. Where better control was required to calculate angles from the short baseline readings on the casing angles, a total station theodolite was used. Both instruments had sub-centimetre accuracy.

- Leica Viva GNSS GS15 receiver- accuracy within 1 cm;
- Leica Viva TS15 (total station theodolite).

10.6. Geological Logging

When the core boxes containing the core were received at the BlackRock facility in Chibougamau and prior to logging, the technician proceeded in the following order:

- The core boxes were laid out in sequential order, and the lids removed under the supervision of the geologist;
- The core in the boxes was confirmed to be in the correct order and properly labeled;
- The core boxes were relabeled with "Dymo" printed aluminum strips, stapled onto the start end of the core box;
- The core was measured for Rock Quality Designation (RQD).

10.6.1. Rock Quality Designation (RQD)

Before logging and sampling, the Rock Quality Designation or “RQD”, was measured the technician following these steps:

- The core was checked to ensure that all pieces were reasonably fitted together;
- Core breaks caused by drilling (fresh fractures) were fitted together and counted as one piece. If there was doubt about the fracture, it was counted as a natural break;
- The core length was measured along the center line to avoid unduly penalizing the quality of the rock mass where fractures ran parallel to the drillhole;

- Except for the first sample out of the casing, RQD = Length of core pieces>10 cm divided by 300 cm times 100;
- The core pieces that were soft weathered/altered intact lengths were not included. If there was any doubt about the competency of the rock, it was omitted from the core length calculation;
- The RQD data was entered on the log spreadsheet RQD Section.

10.6.2. Geological Logging

All the original logs were first hand-written and they were subsequently transferred onto an in-house Excel Spreadsheet format. Each drillhole had a designated spreadsheet. Details of the headings under which information is entered are as follows:

- Project Name;
- Company Name, Canton/Municipality, and Claim Number;
- Hole ID, Length, Azimuth, Dip, Core Size, Contract Driller;
- Collar Coordinates;
- Start Date, Finish Date, Geologist, Casing;
- From, To, Rock Code, Description (including a rock description, physical properties such as magnetic/non-magnetic, structural description.);
- Percentage Recovery;
- Sampled interval has "From", "To" with corresponding Sample ID (derived from sample ID booklets) with Blanks and Duplicates.

During the 2010 drilling campaign, samples were collected from both the Southwest and Armitage Pits. The entire core was transported to the Chibougamau rented facilities (seen in Figure 10-3); no logging was done directly on the drilling site. To accomplish the entire core logging and photographing, BlackRock had hired two consultant geologists for the Southwest Pit and one for the Armitage Pit.

The geologists were helped by two technicians in charge of core splitting and sample preparation. The selection of sampled intervals was done by geologists during logging and by identifying the magnetite horizons with a hand magnet. The entire magnetic zone was then sampled. Each sample had a length of 3 m. Moreover, a 3 m long sample was taken before and after the magnetic zone in order to validate the limits of the magnetite-bearing mineralization. The internal waste horizons longer than 3 m within the mineralized interval, were not sampled and were therefore considered as barren.

For both the Southwest and Armitage Pits, limits of all samples were marked on the core by the geologist and each interval was numbered and labelled. The technicians split each mineralized interval in two . One side of the core was put back in place into the core as witness for future use or validation whereas the other half was put into a numbered plastic bag. Plastic bags with half core samples were sent for sample preparation at the "*Table Jamésienne de Concertation Minière*" (TJCM) facility in Chibougamau.

Duplicate tags with unique sample sequence numbers were used for sample tracking control. Booklets of tags were used by the core loggers and splitters to assure unique sample IDs. One copy was left attached to the interval sampled in the core box and a copy was provided inside the bag of split core that was submitted to TJCM for preparation.

Figure 10-3: Core logging facility in Chibougamau

10.6.3. Hyperspectral Survey

As described in Section 9.4.2, BlackRock used a full hyperspectral scan of the core by Photonic Knowledge (PK). BlackRock scanned its entire core, both historical and newly drilled, covering over 95% of the core. Core that was omitted from the scan was historical core that had been used up in sampling. This program of work produced a "virtual core shack", a library log of high resolution pictures of the core (Figure 10-4).

Figure 10-4: Example of PK scanned core AE-02-01, 94.5 m – 113.3 m.
This is Part of the Virtual Core Shack that Encompasses Core for Essentially the Entire Project

10.7. Recovery

Generally, due to the competent nature of the rock, recovery of core and RQD were very good, averaging 93.0% and 92.8%, respectively. In Northern Québec, due to the temperate climate and quaternary overburden coverage, weathering does not have any negative effect on the RQD and the rock, neither at surface nor at depth.

10.8. Sample Length/True thickness

From interpretation of the geomagnetic surveys and surface outcrops, it is indicated that the igneous layering trends N070° and dips steeply at 60o-80o to the southeast on the Southwest Deposit. On the Armitage Pit, igneous layering is at N070° overall and dips at 60°-80° to the south-southeast. Azimuth and plunge of drillholes were therefore orientated perpendicular to the overall trend of the Southwest and Armitage Pits and at -45o (the limit of the drilling angle) to intercept mineralization as near as perpendicular to its strike and dip.

It was acknowledged that intercepting the layered structures at 45o would not give a true thickness of the stratification on the core, the apparent thickness being 10-15% greater than the true thickness depending on the dip of the stratification. Similarly, sampling that was done on 3 m lengths are apparent lengths.

Consequences for the resource model were minimal, as the rock units were modelled to join the intercepts of the drillholes with the rock units and create an envelope that defined the true unit thickness.

It should be noted that the widths of cross-cutting structures, such as faults, shear zones and dykes that run parallel/sub-parallel to the drilling orientation, may be greatly exaggerated if intercepted. For the BlackRock Project, this was an important consideration during interpretation since, as seen from the geological history of the region, there are major structures running obliquely across the strike of the deposits that are sub-parallel to the drillhole direction.

10.9. Geotechnical and Hydrological Drilling

10.9.1. Project Site Investigations

Seven short vertical geotechnical holes were drilled to define rock integrity in the proposed plant and infrastructure area. More precisely, four holes totalling 115.25 m, were completed in the mill area, and three holes, totalling 92.48 m, for the crusher area.

10.9.2. Open Pit Investigations

There have been eight geotechnical holes drilled in the proposed pit layback portions of the Southwest Deposit. They were logged and partially assayed, and were fully described for RQD and structural information.

10.10. Metallurgical Drilling

Samples for metallurgical testing were taken from trenches and core samples. However, trench samples were found to be poorly representative, as the sampling was open to bias and samples may have suffered some weathering. Since 2012, the BBA consultants have no longer accepted surface samples for inclusion in resource calculations.

Sampling from NQ drill core was unbiased but limited in terms of the volume/weight available, and led to the loss of the core record.

To achieve large 20 tonnes bulk samples per deposit, BlackRock used 20 HQ drillholes on each the Southwest and Armitage Pits. The drilling was spaced along the trend of the deposits and drilled to a vertical depth of 150 m. The holes were referenced with a "BK" suffix in the drillhole ID.

All BK cores were scanned by PK and logged by the geologist. To provide representative samples for metallurgical testing, samples were taken from the oxide layers, where internal waste was less than 4m in length.

Sampled core was not split, but put into drums to be sent to the testing laboratory.

10.11. Condemnation Drilling

The six condemnation drillholes, CD-1 to CD-6, were carried out across the area, from the southwest end of the Southwest Deposit, across to the northeast end of the Armitage Pit (Figure 10-5).

Condemnation holes were orientated to cut across the magnetic anomaly, e.g., CD-2 was orientated towards the north, whereas CD-6 was orientated to the northwest. This is the proposed site for the workshop and crusher facilities situated around Lac Denis. It is represented by a dislocation of the magnetic anomaly, and appears, based on the drilling, to be an area of shear zones with only minor oxide mineralization. The red outline in Figure 10-5 shows the approximate location of the magnetic anomaly around Lac Denis.

The condemnation drilling was fully scanned by PK and logged on the core bench. A hand magnet was used to determine if there was insufficient magnetite present to warrant splitting and assaying. Figure 10-6 shows an example of the sheared (foliated) nature of the host rock, with cross-cutting calcite veining. This type of deformation is not common in the main mineralized sequence.

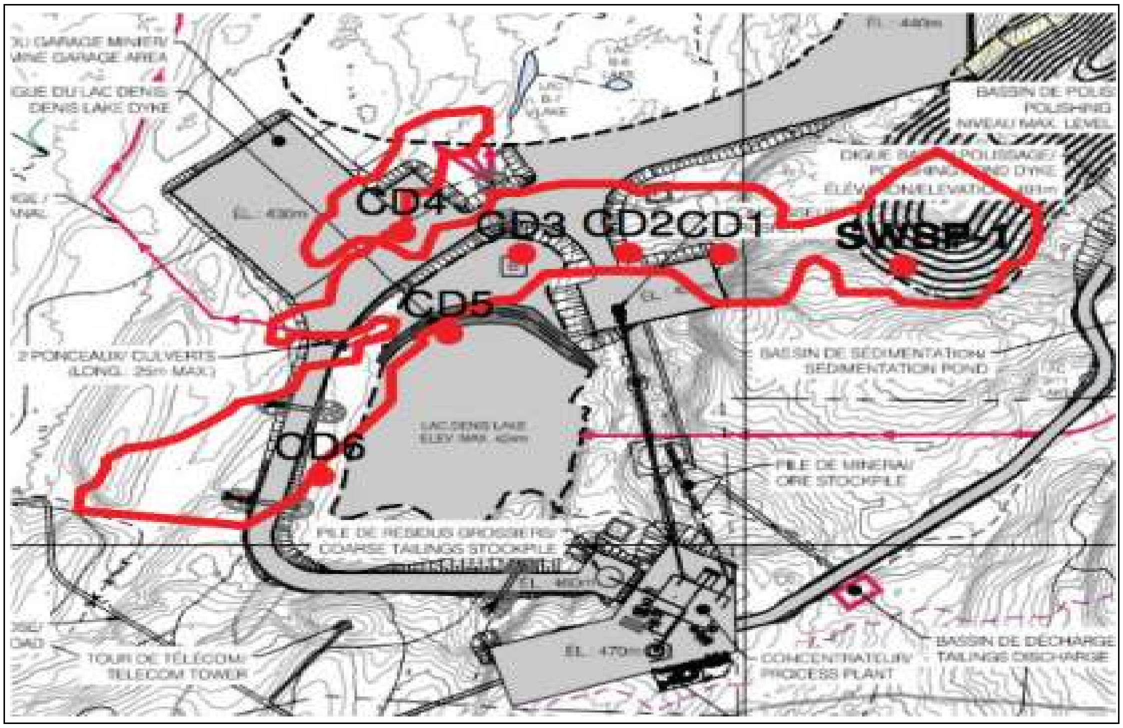


Figure 10-5: Condemnation drilling

Figure 10-6: CD-3 from 119 m to 140 m. Typical core from the condemnation holes around Lac Denis

11. Sample Preparation and Analyses

11.1. Sample Selection and Preparation

11.1.1. Selection Protocol

BlackRock's core has been sampled on nominal three metre standardized lengths based on what was practical and mineable in order to add greater consistency to the database. Interval selection was not influenced by the local geology. Areas with sufficient magnetite to be detected with a hand magnet were selectively assayed, but within the sampled intervals the core was split on the standardized 3 m composites that start and end on downhole multiples of 3 m. Additional material was generally sampled on the shoulders either side of oxide rich samples to ensure sufficient sampling coverage across the targeted mineralization. In 2012, the project was broadened to include ilmenite and samples were taken in both the new core and by resampling of the old core to include the majority of the material in and adjacent to the cumulate horizons.

The procedure included identifying sample intervals by the geologist using a wax crayon to mark the sample interval directly on the core. At the start of the sample, two sample tickets were put into the core box adjacent to the sample; one with the sample "From - To" interval (in m), which the technician staples in the core box, and the other one with a uniquely coded sample ID that accompanies the core to be split.

The technician then splits the sample leaving half in the core box as witness, and placing the other half into a sample bag with its accompanying ID ticket. Prior to 2012, all core was mechanically split. During 2012, although the preferred splitting methodology shifted to the use of a saw, core splitters were also employed to meet the study deadline.

Sampled core was put in a large sample bag which was numbered with a permanent marker with the Sample ID, and the Sample ID ticket was also added into the bag as a cross-check. In cases the core has been totally consumed by sampling, that section of the core box was labelled accordingly. Finally, the core boxes were transferred for storage onto the core racks in a secure, fenced area with a padlocked gate.

Sample bags were then sent to the "Table Jamésienne de Concertation Minière" (TJCM) laboratory as the preparatory lab. Pulps or crushed materials (Section 11.1.2) were then shipped to the appropriate laboratory for assaying. BlackRock has used COREM, ALS Minerals and SGS Canada for assaying.

COREM has implemented two quality management systems to ensure its laboratory services meet the highest standards of the mining industry. The pyrometallurgical characterization laboratory is certified ISO 9001:2008 and the analytical services laboratory is accredited ISO 17025:2005.

ALS Minerals is an internationally recognized minerals testing laboratory operating in 16 countries. It has a Quality Management System with an ISO 9001:2008 and ISO/IEC 17025:2005 certifications.

SGS Canada – Minerals Services – Lakefield has been accredited to ISO/IEC 17025:2005 standards by the Standards Council of Canada (SCC).

All the above laboratories are private and are totally independent and have no relationship with BlackRock Metals and/or Strategic Resources.

11.1.2. Preparation Protocol

Sample preparation for all BlackRock programs starting in 2009 has followed an identical protocol utilizing the "*Table Jamésienne de Concertation Minière*" (TJCM) laboratory as the preparatory lab:

Once a batch of bagged samples was ready, they were taken to the TJCM laboratory accompanied by a submittal note from the geologist that included the following detailed information and instructions:

- Weight of pulps to be prepared;
- The mesh size of the sample;
- The laboratory for each assay and its location;
- The type of assay requested;
- The sample split protocol for other tests (e.g. 20% samples to be sent for DTA and Pycnometer);
- Sample ID numbers assigned a second split for QA/QC duplicates and/or blanks;
- A copy list of recipients of the results;
- The responsible BlackRock geologist and the date of submittal.

After reception at TJCM, sample preparation consisted of:

1. Entire samples were crushed to greater than 70% passing 10 mesh;
2. A split of up to 250 g was then pulverized to greater than 85% passing 200 mesh and homogenized;
3. A 100 g portion of the resulting pulp of each sample was sent to the respective labs for WRA, DTA, SG and Satmagan analysis as ordered by the BlackRock geologist;
4. The remaining portion of the pulp and any coarse rejects were stored at the TJCM facility.

11.2. Chain of Custody and Security

A chain of custody protocol was set up to govern the transit of the samples as they passed on from BlackRock to the TJCM and from TJCM to the assay laboratories.

1. The geologist produced a dated submittal note detailing the Sample ID's and instructions (as above) for the sample processing and assay requirements;
2. The preparation laboratory (TJCM) received the samples and sent a dated acknowledgement of samples received;
3. TJCM sent prepared samples to the assay laboratories with instructions for assay requested, and gave notification of the dispatch of the sample IDs to BlackRock;
4. Upon receipt of the samples the receiving laboratories sent acknowledgement of samples received;
5. Once the assay certificates were issued to BlackRock, the remainder samples and pulps were returned to BlackRock.

11.3. Density Determinations

Specific gravity (SG) and density sampling have both been employed to allow for the conversion of our model volumes into tonnages.

Density variations within the Lac Doré Complex are locked to variations in modal mineralogy. Density within the constituent minerals ranges over a factor of 2X, from ~2.7 g/cm^3 for pure anorthite plagioclase to 4.7 and 5.2 g/cm^3 for pure ilmenite and magnetite, respectively. The wide range of values makes the bulk density of the rocks very sensitive to small changes in relative modal mineral percentages, and the high densities for the economic metal bearing minerals emphasize the importance of accurate density models, as well as assays for Satmagan, V_2O_5 and TiO2 in order to properly quantify Mineral Resources and reserves.

BlackRock has undertaken extensive sampling programs in order to quantify this density variation with as much precision as possible. There are three programs that have been executed to date. These form the basis of the conversion of block volumes to block tonnages.

11.3.1. Immersion Testing for Specific Gravity

Specific Gravity (SG) immersion testing (Archimedes' principle) has been carried out by BlackRock in two separate programs. In a standard SG test, a section of core is weighed in air, and again submerged in water. The ratio of these two values defines the SG for the material sampled.

11.3.1.1. Specific Gravity in Mineralized Units (2010)

The first program was undertaken by BlackRock in 2010 and produced 85 samples from the mineralized units and the material immediately adjacent to them.

- The Southwest Deposit is represented by 40 samples, all taken from drillhole SW-26-01 across all stratigraphic units;
- The Armitage Deposit is represented by 45 samples taken from all units but spread systematically on the even-numbered Drill Panels across the entire deposit.

For the 2010 and 2011 models, SGS used the SG information to derive regression relations to the corresponding Satmagan readings. These regression equations for the two deposits were then applied to the interpolated block Satmagan values to assign a density conversion for the block volumes to estimate tonnage in each case.

11.3.1.2. Specific Gravity in Waste Rock (2013)

The second program was completed in 2013 to characterize the density of the waste rock in the pit laybacks. There were a total of 45 additional samples taken in this program spread evenly between the hanging wall and footwall material away from the main mineralized layers. These data are used to assign an average density by sector to the waste material in the pit models for this study. A summary of the results is provided in Table 11-1.

Table 11-1: Summary findings from survey of 45 samples covering the hanging and foot wall portions in both pit laybacks

	Average SG Piece	Pycnometer	Piece/Pyc
Southwest Hanging wall	3.021	3.070	-1.6%
Southwest Footwall	3.100	3.173	-2.3%
Armitage Hanging wall	2.975	3.060	-2.8%
Armitage Footwall	3.099	3.207	-3.4%

On average, 11 samples were taken from waste rock on each side of the mineralized units in each pit. Both SG and pycnometer measurements were made of the same material that represented 1 m composited material that was previously unsampled from the drilling.

As is the case in the comparison study detailed in Section 11.3.3 below, there is a systematic bias to slightly heavier pycnometer values, and the footwall in each pit has a higher discrepancy than the hanging wall. There is also a slightly larger discrepancy between the methods in the Armitage over the Southwest. All of this variance is considered minor and within the precision tolerance of the methods. As is the case within the mineralized units, the pycnometer information is accepted as the more representative sample.

11.3.2. Pycnometer Testing for Density

Pycnometer measurements of bulk density were employed on a systematic basis to produce a random distribution of data across all rock types in the drilling. The use of pycnometer readings on pulps allows for a direct measurement to be made for each 3 m composite that is then tested for Satmagan, WRA and DTA.

This method is deemed less prone to sampling error than the SG testing, which uses a small sample (whole rock of 50-500 grams) relative to the whole interval it represents (~20,000 grams). The potential for error in taking a non-representative sample is due to the alternating layers of light and heavy cumulates that occurs on all scales in this deposit. This method is prone to potential bias due to the loss of natural pore space and voids in the rock during the pulverization process.

A total of 1,043 pycnometer tests comprise the database used for the density modelling in this study. There are 518 density measurements from the Armitage and 525 for the Southwest deposits. These were taken systematically and cover both deposits randomly (nominally every 5th sample assayed was submitted for density determination). All rock types are represented with statistically significant sample populations.

11.3.3. Comparison of Immersion vs. Pycnometer Methods

In practice, there is some variance in the comparison of the same sample intervals by these two methods, but they generally agree very well with each other. Figure 11-1 shows the comparison of 73 samples that report values in both datasets.

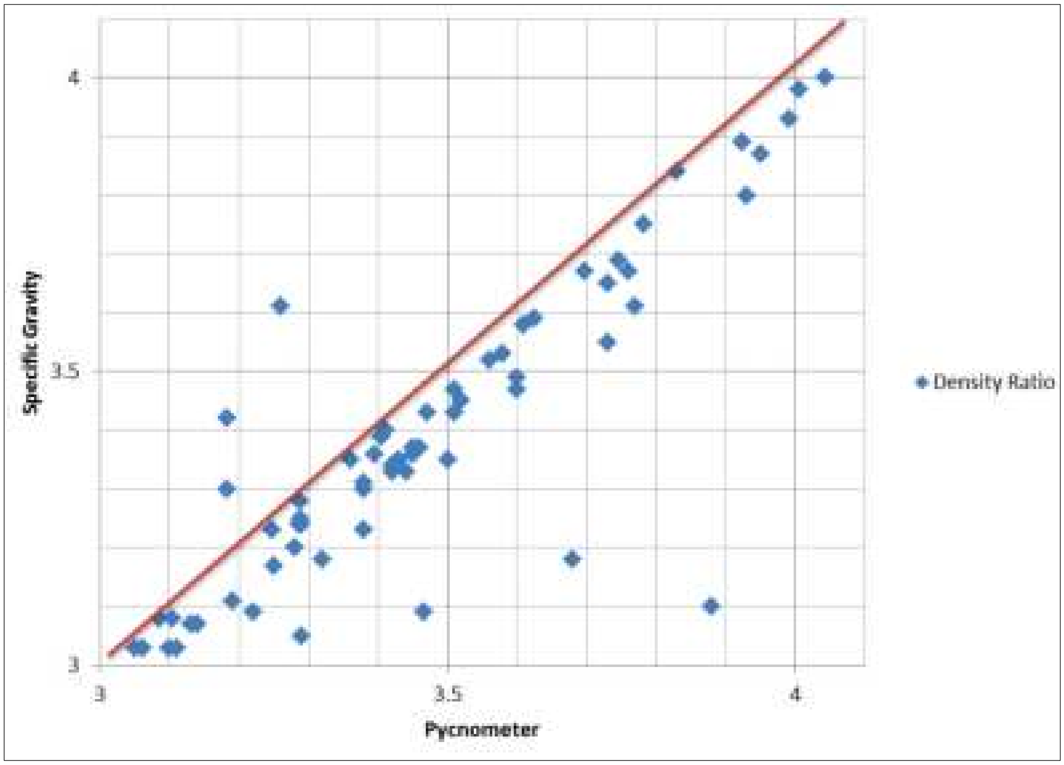


Figure 11-1: Comparison plot of pycnometer vs. specific gravity for 73 samples in both the Armitage and Southwest Pits

The diagonal line represents the 1:1 ideal correlation. These represent comparisons between labs and between pycnometer readings from 15 cm sections of core and 3 m composited intervals of core for about 42 samples done at COREM, and dual tests on the same 15 cm interval for about 31 samples done in the SGS verification work. All immersion tests for specific gravity were completed at TJCM.

Note that there are a number of outliers as well as a steady state a degree of scatter in the data. This is probably due to the difference in sampling support with the SG representing a more select subset of the 3 m composite interval.

There is also a systematic bias toward a ~2.25% higher average density values in the pycnometer data. This slight bias is probably reflecting a small amount of pore space available in these greenschist facies crystalline rocks. The magnitude of the variance is well within acceptable levels, and the raw data is accepted for the modelling work.

Figure 11-2 shows the same comparison between 45 samples taken from layback areas in both proposed pits. The same slight bias toward heavier values in the pycnometer data is apparent with an average of ~2.5%.

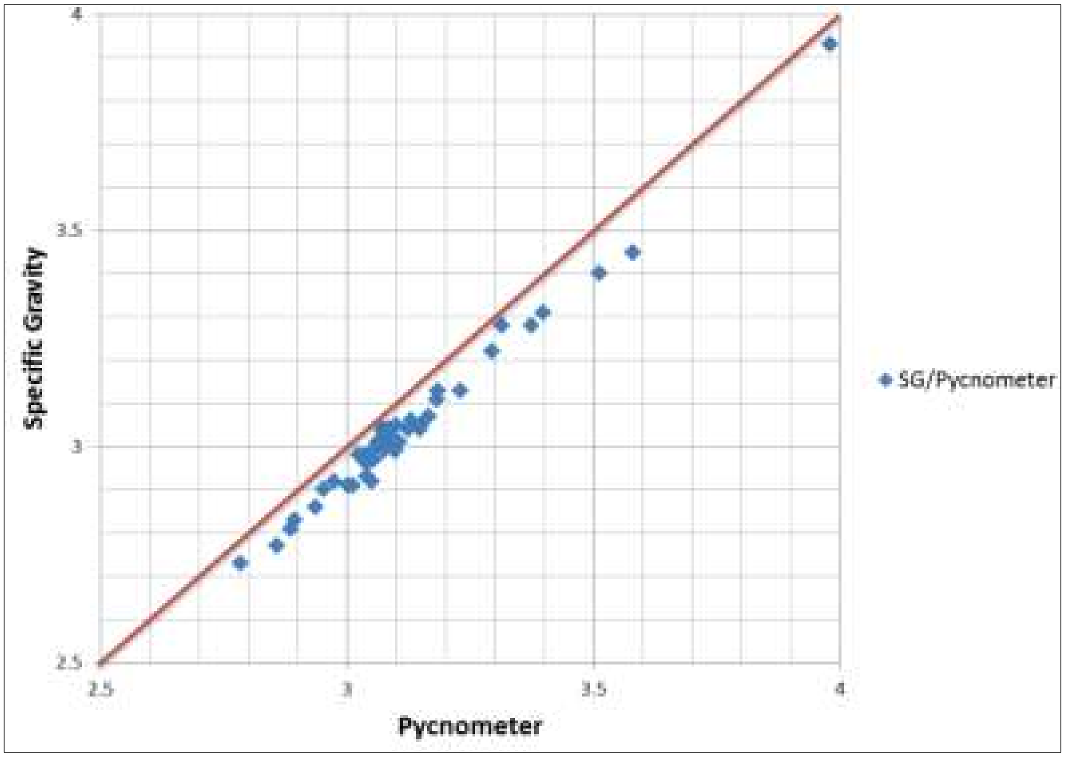


Figure 11-2: Comparison of SG and pycnometer data for waste rock samples taken in 2013

Both deposit areas are represented in these data. The diagonal line represents the 1:1 ideal correlation. Both datasets were generated at SGS from the same samples and pulps.

11.3.4. Modal Density

The SG tests support the general accuracy of the pycnometer measurements, and the superior 3 m support size of those data conforms to identical sample intervals submitted for assay. These 1,043 samples represent about 10% of all assays sampled, and are not sufficient to allow for direct integration into the block model. However, this dataset can be used to develop and refine a density model based upon the proportions of the constituent minerals present in any given sample or block.

The Lac Doré Complex has a very simple mineralogy that displays a wide spread in density values for the four main mineral components that comprise these rocks; magnetite, ilmenite (both free and entrained as exsolution laminae), iron bearing silicates (predominantly chlorite, amphiboles and pyroxene) and anorthite (and associated alteration minerals). This simplicity and wide density spread has allowed for the development of a robust regression algorithm that yields an estimate of modal mineralogy from assays of WRA and Satmagan. Coverage for both of these assay types is excellent throughout the mineralized units, and much of the nearby interstitial material.

The following outlines the methodology used to derive the modal regression:

- Satmagan is taken as a direct proxy for %Magnetite (Fe_3O_4);
- %TiO_2 from WRA is converted to %Ilmenite (FeO-TiO_2) by dividing the assayed TiO_2 value by its stoichiometric proportion (0.5265);
- Total elemental Fe (Fe_T) is then calculated from the WRA Fe_2O_3 by dividing by the conversion constant 1.4297 to reduce the oxide representation back to elemental proportion;
- The elemental Fe content in the modal magnetite (=72.36% Fe) and modal ilmenite (=36.81% Fe) is then calculated and subtracted from the Fe_T;
- The residual Fe_T is then converted back into Fe_2O_3 by multiplying by 1.4297 and that value is assumed to be equal to the amount of Fe-silicate present;
- Finally, the total modal percentage of Light (Ca-Al) Silicates = 100% - %magnetite + %ilmenite + %Fe-silicate.

The resulting Modal Mineralogy becomes a powerful tool in validating the metallurgical test results and in allowing a metallurgical/mineralogical block model to be generated. As noted above, the wide density spread between the main ore-bearing oxide phases and the silicate gangue allows for the development of a modal bulk density model. There is precedent to validate this approach in the literature on LMIs, as well as in the mining industry in Québec where QIT uses bulk density as a primary method for grade control in some Fe-Ti oxide deposits.

To adapt the Modal Mineralogy into a Modal Density model the 1,043 pycnometer samples and their accompanying WRA values were used. The starting assumptions for densities of the four mineral groups are summarized in Table 11-2.

Table 11-2: Density input assignments for the modal density model

Mineral Class	Initial Density Assumption	Final Density Result	Basis for Assumptions for Initial Proportions
Magnetite	5.20	5.10	Satmagan = Magnetite
Ilmenite	4.72	4.70	TiO_2/0.5265 = Ilmenite
Fe-Silicates	3.40	3.42	Fe_T - MagFe - IlmFe (expressed as residual Fe_2O_3) = Fe-Silicates
Light Silicates	2.80	2.90	100-%Magnetite-%Ilmenite-%Fe-Silicates = Light Silicates

These values were then adjusted empirically to minimize the difference in the mean, median and standard deviation for the global dataset. The density for ilmenite was held closest to the ideal value reported in the literature. The magnetite was lightened slightly to account for the small amount of Ti that is substituted within the lattice, and the silicate categories were allowed to adjust to best fit within broader ranges. The final results of this process are shown in bold next to the starting values in Table 11-2 above.

Table 11-3 below provides the final statistics for the model. Columns labeled +/-5% and +/-10%, provide statistics for datasets that have had "outlying" data trimmed to allow evaluation of the weighting of these data in the statistics for full dataset. The full dataset was used in the optimization process to derive the Modal Density; the Modal percentages for the four components are simply multiplied by the corresponding density value and the total of the four resulting numbers:

Modal Density = (%Magnetite)*5.10 + (%Ilmenite)*4.70 + (%Fe-Silicate)*3.40 + (%Light Silicate)*2.90.

Table 11-3: Statistics for the final regression model densities as a best fit to all measured density values for both pits

	+/-5%	+/- 10%	All Data
Number of Samples	963	1026	1043
Average Difference	0.020%	-0.015%	-0.048%
Median Difference	0.082%	0.075%	0.059%
Standard Dev	2.4%	2.9%	3.8%

11.4. Analytical and Test Laboratories

All assaying for the project has been done using just three labs; ALS Chemex, SGS and COREM. Most of the metallurgical work has been done at COREM and SGS-Lakefield.

11.5. Instrumentation, Procedures and Precision

The primary analytical methods employed on the BlackRock project have been whole rock analysis (WRA) for assaying of TiO_2 and V_2O_5, LECO for S, Satmagan for the mineralogical assay of magnetite content and DTA for estimates of weight recovery of magnetite. The density values used as the basis for density modelling and resource calculation were systematically measured using gas pycnometer.

11.5.1. Whole Rock Analysis

Accurate assaying is critical to the success of any iron ore exploration, resource definition or metallurgical program. Whole rock analysis (WRA) for major and minor element content by X-ray fluorescence spectrometry (XRF) is an industry standard for the analysis of oxide iron ores. A pulverized sample is first fused using lithium metaborate as the flux to dissolve resistive minerals and chemically homogenize the sample, followed by X-Ray fluorescence spectrometry. X-ray fluorescence (XRF) is based on wavelength-dispersive spectroscopic principles that are similar to an electron microprobe (EPMA), but applied to bulk analyses of large samples as opposed to very narrow (2-5 µm) focus of the microbeam techniques. The precision of the analysis in all elements reported are excellent with the limit of detection reported at 0.01%.

11.5.2. Satmagan

Accurate analysis of ferromagnetic compounds such as magnetite is difficult and time consuming by conventional wet chemical methods. The Satmagan (SATuration MAGnetic ANalysis) method was designed specifically to measure the magnetite content of iron ores and has been in general use for over 40 years. The instrument measures the force (total magnetic moment) acting on a small ~1.2 cm^3 pulverized sample by applying a strong enough magnetic field to saturate the magnetic component of the sample while measured within a known vertical magnetic spatial gradient.

The instrument is calibrated to standards of known magnetite content and is more accurate and reliable than measurements based on magnetic susceptibility. An average grain size greater than 150 µm (100 mesh) provides accurate measurements. For finer materials, the Satmagan gives slightly lower readings, so a different calibration curve is required. The range of measurement is 0 to 100% by magnetite weight with reproducibility of 0.2% and precision of less than 0.4%. Operating temperature range of the instrument is +10°C to +40°C (+50°F to +100°F).

11.5.3. David Tube Analysis (DTA)

DTA is a simple technique to quantitatively analyze the magnetic component of a sample. It employs an electromagnet which can generate a magnetic field intensity of up to 4,000 gauss, a glass separation tube and a motor driven agitation mechanism. The tube is positioned between the poles of the magnet at an angle of approximately 45 degrees (the angle is adjustable). Between 10 and 30 grams of sample are used for the analysis. During operation the agitating mechanism that supports the water-filled glass tube moves forward and backward while simultaneously rotating. Agitation continues at a constant speed until the slimes are washed from the tube. The magnetic material is attracted and held tightly between the two poles of the electromagnet. Of the assay techniques used, this is the most prone to operator bias and its precision and accuracy are not as good as the chemical and geophysical techniques as a result.

11.5.4. LECO

A portion of the drilling was analyzed for S using a leco sulphur analyzer. A small sample (0.01 to 0.1 g) is heated to approximately 1,350°C in an induction furnace while passing a stream of oxygen through the sample. The sulphur dioxide released from the sample is measured by an infrared detection system and a value for total elemental S is reported with a precision of 0.05%.

11.5.5. Pycnometer

A gas pycnometer is an effective way of accurately determining density for large representative sample intervals of non-porous rocks such as those on the BlackRock Project. A pycnometer is a simple instrument consisting of two chambers; one to hold the sample and a second chamber of a known reference volume. Pressurized gas is admitted to the sealed sample chamber which has a pressure measuring transducer and is connected to the second chamber via a valved connection. The sample volume can be precisely determined by exploiting the volume-pressure relationship known as Boyle's Law:

$$V_s = V_c + \frac{V_r}{1 - \frac{P_1}{P_2}}$$

where V_s is the sample volume, V_c is the volume of the empty sample chamber (known from a prior calibration step), V_r is the volume of the reference chamber (again known from a prior calibration step), P_1 is the first pressure (i.e., in the sample chamber only) and P_2 is the second (lower) pressure after expansion of the gas into the combined volumes of sample chamber and reference chamber.

The resulting volume is converted into density by calculating the ratio of sample's mass to its volume and is reported with a precision of 0.01%.

11.6. Quality Assurance and Quality Control (QA/QC)

11.6.1. 2010 through 2012

As mentioned in the earlier sections, BlackRock's 2010 and 2011-2012 drilling programs (Table 11-4) included an in-field QA/QC during initial core sampling and 2012 resampling that involved insertion of blanks and duplicates into the sample stream going to ALS Chemex Inc. (ALS), SGS Canada Inc. (SGS) and COREM Inc. (COREM)

Table 11-4: Summary of BlackRock DDH Core Sampling

Parameter	Southwest Pit		Armitage Pit		Total
Year	2010	2011-2012	2011	2011-2012	
No of Drillholes	49	103	28	102	**205**
Metres Drilled	12,429	23,519	8,093	22,588	**45,800**
No of sampled intervals (3 m length)	2,090	4,370	2,633	4,001	**8,371**
SAT assays	2,090	4,134	1,573	3,978	**8,112**
WRA assays	1,060	3,724	288	3,969	**7,693**
Davis tube assays	147	802	288	776	**1,578**
Density assays	69	676	272	774	**1,450**
Hyperspectral (PK Scan) metres	0	22 987	0	19, 662	**42,649**

On average for the 2010 and the 2011-2012 drilling programs, BlackRock geologists inserted a duplicate sample into the field every 10th routine sample. Duplicate samples were collected by quarter sawing the predetermined sample intervals and using ¼ core for the duplicate sample, ¼ for the regular samples, and the remaining half core was returned to the core tray for reference.

For the blank sample, BlackRock did not use certified material from an accredited laboratory. Instead, for the 2010 campaign BlackRock used halite, calcite, quartzite and barren anorthosite for blank material.

Table 11-5: Distribution of BlackRock's routine and QA/QC samples

Sample Type	BlackRock		ALS		SGS		COREM	
	XRF WRA	SAT	XRF WRA	SAT	XRF WRA	SAT	XRF WRA	SAT
BlackRock Routine	7693	8112	7212	-	481	4202	-	4356
BlackRock in-field Duplicate	-	-	325	-	146	83	-	318
BlackRock in-field Blank	-	-	166	-	18	169	-	154
ALS QA/QC Duplicate	-	-	488	-	-	-	-	-
ALS QA/QC Blank	-	-	1666	-	-	-	-	-
SGS QA/QA Duplicate	-	-	-	-	27	79	-	-
COREM QA/QC	-	-	-	-	-	-	-	-

Each laboratory also conducted in-lab QA/QC programs. The number of samples and type of analysis for both of these programs are summarized in the Table 11-5. ALS and SGS in-lab QA/QC assay results were available to BlackRock. COREM did not provide any of the in-lab assay results but their protocol is summarized in Table 11-6.

Table 11-6: Summary of COREM Laboratory QA/QC Program

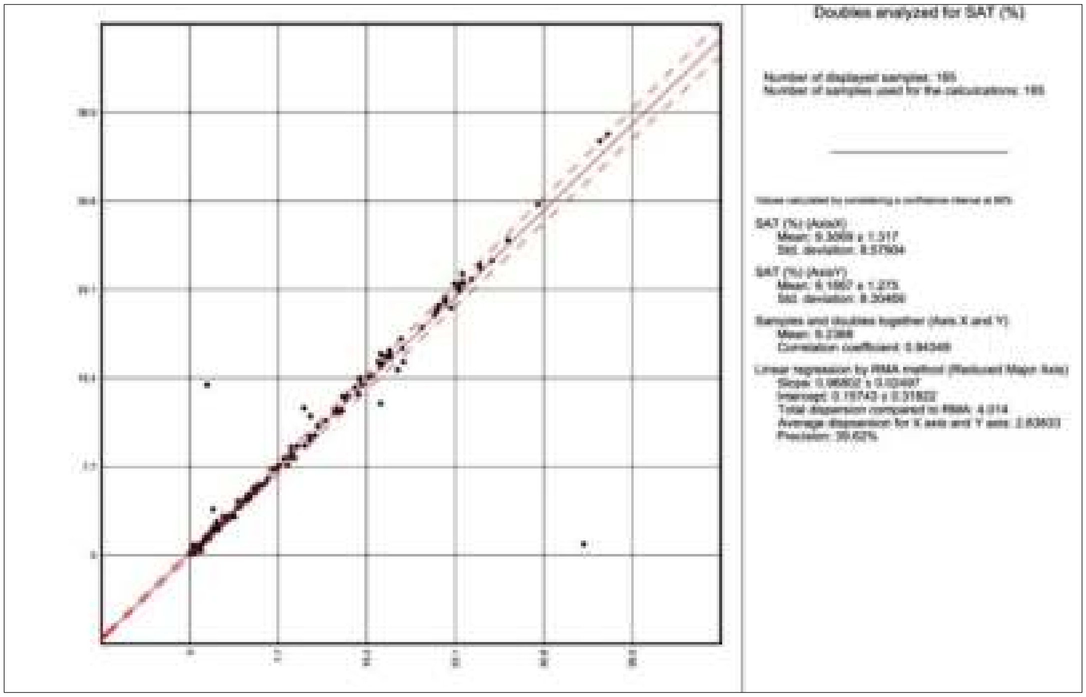
ASSAY	Sample per Sequence (max)	Blank	Duplicate	MRI / MRC
SAT131: Satmagan	20	1	1	3
A25: WRA by XRF	20	*	1	2
B41: S by LECO	20	1	1	2
DEN120: Density by Pycnometer	10	**	1	2

*Instrument is calibrated daily using a certified pellet.
**For XRF, 1 blank is assayed every 20 samples or at the beginning of a new batch.

BlackRock QA/QC program additionally included an X-check comparison of laboratory by sending pulp from the same sample to a different lab.

11.6.2. BlackRock In-Field Duplicate Results

Using GeoticLog QA/QC drafting tools, BlackRock generated graphs for duplicate WRA and Satmagan assays. Figure 11-3 and Figure 11-4 are graphs for Satmagan, Fe_2O_3, TiO_2, and V_2O_5.



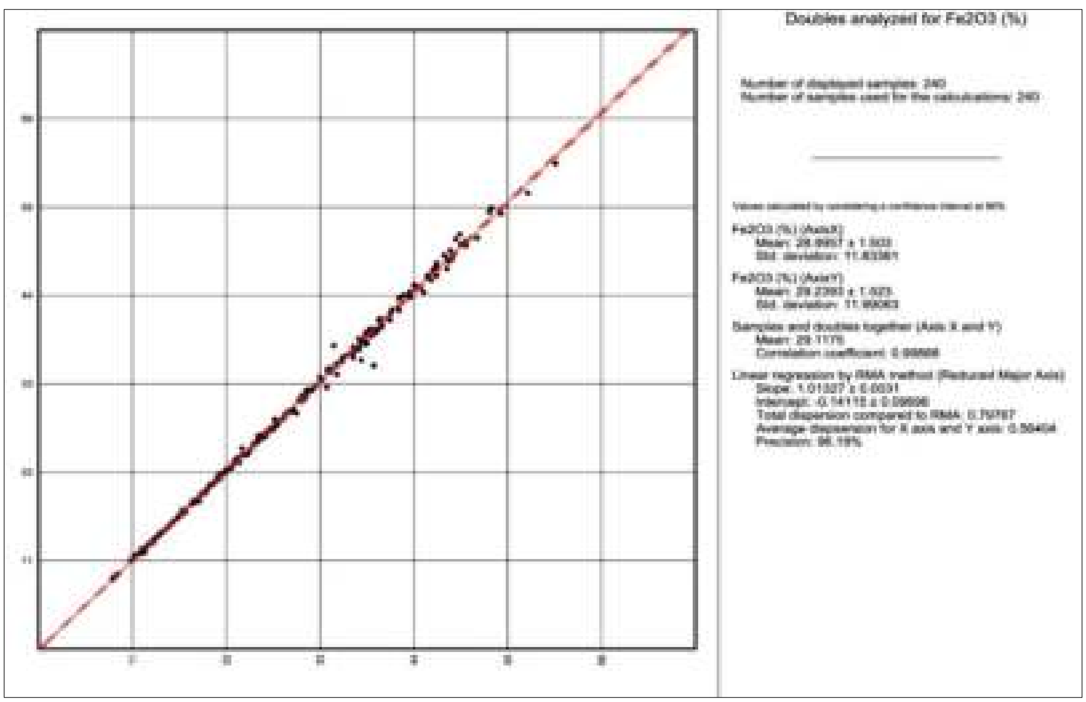
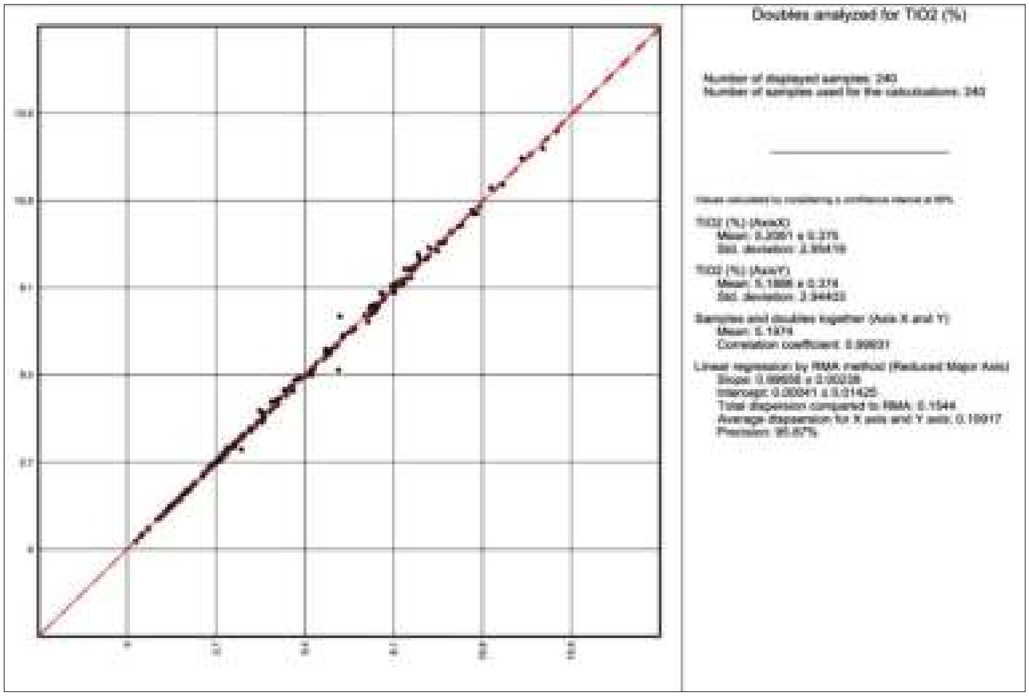


Figure 11-3: Satmagan and Fe_2O_3 assay results for in-field Blackrock's duplicate



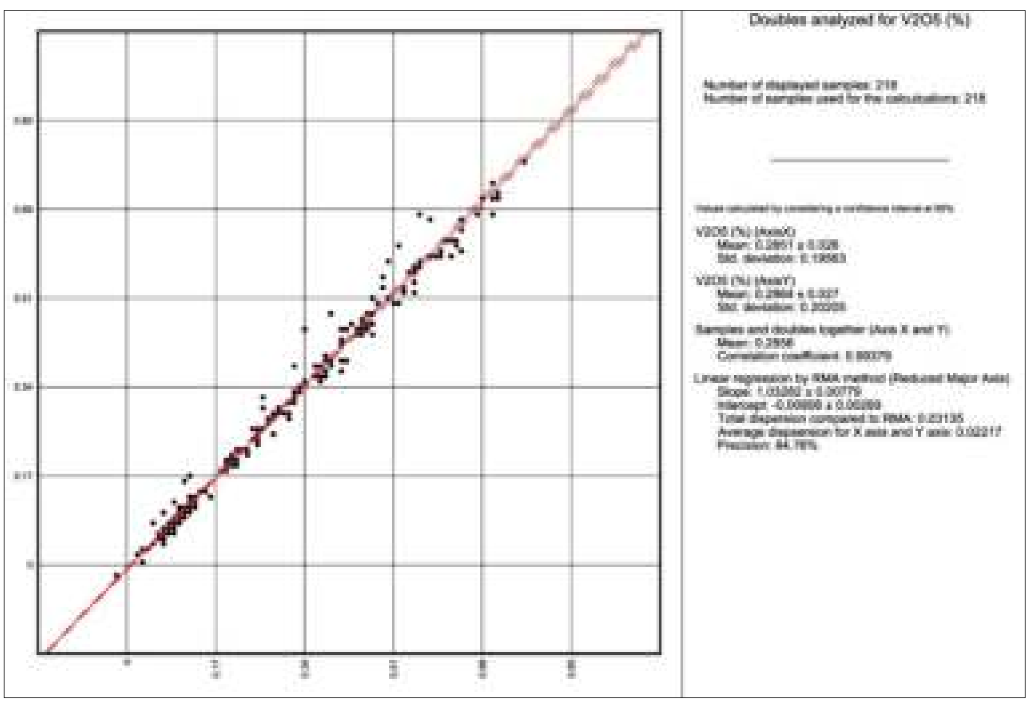


Figure 11-4: TiO_2 and V_2O_5 assay results for in-field BlackRock's duplicate

All graphs in Figure 11-3 and Figure 11-4 show a perfect linear correlation between the original Satmagan and WRA assays and their duplicate counterparts. Correlation coefficients range from 0.945 to 0.999. Correlation coefficient is above 0.99 for all major elements, except for those where assay values approach the detection limit.

11.6.3. BlackRock Blank Results

During the 2010 drilling program on the Southwest Deposit , halite and quartzite were first used as the blank. Quartzite was found to be too abrasive, and was picking up iron from the grinding mill.

Calcite replaced quartzite and halite as the blank for the 2010 drilling program on the Armitage Deposit and the winter 2011 drilling program on the Southwest Deposit . Calcite proved to be a problem as, on calcination of the sample, it produced CaO that was too refractory and would not fuse for the XRF sample.

Finally, for the 2012 drilling program on Armitage, BlackRock used a homogenized sample of anorthosite as the blank.

11.6.3.1. Halite Blank

Halite was used as blank only for Satmagan testing and returned an average value of 1.7%. As observed on Figure 11-5, halite blank gave excellent duplicate results with 46 of the 47 samples within the 95% range of correlation coefficient. One sample returned a suspect Satmagan value higher than 14% and this is clearly a mislabelled sample. This sample had not been re-run to confirm this at the time of this report.

11.6.3.2. Calcite Blank

Satmagan testing showed an average of 0.6% for the calcite blank. Due to the calcination problem mentioned above, calcite did not return any WRA results. On the other hand, it was successfully assayed by LECO furnace for S. On Figure 11-5, blank calcite returned highly consistent Satmagan assay results with 149 of 154 samples within the 95% level of confidence. Five samples returned values outside the 95% limit. For three of them, it was again possible to suspect a misidentified sample or labelling. Investigation is in progress.

As expected, only five of the 45 calcite blanks assayed for S returned values of any kind, and these were all basically at the limit of detection. Therefore, assay values for S are considered excellent.

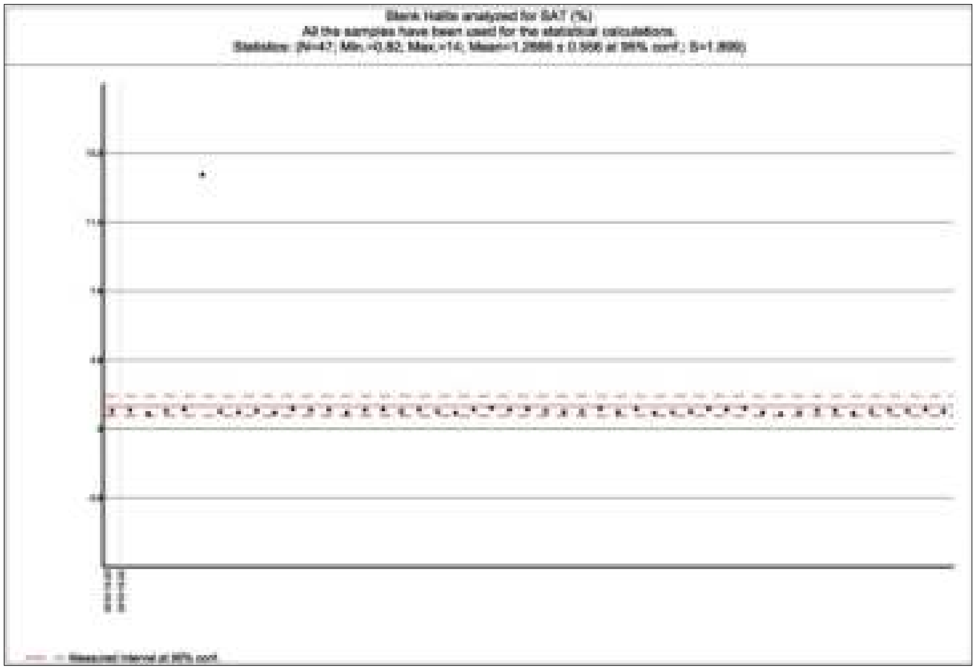
11.6.3.3. Quartzite Blank

Satmagan assay results for quartzite blanks average 2.1% (Figure 11-7). Fe_2O_3 assays generally do not indicate any systematic iron enrichment related to grinding. The isolated higher Fe_2O_3 values may be related to a sampling mistake, sample labeling errors or isolated iron enrichment. BlackRock does not consider enrichment as being a significant factor and does not think it has affected normal routine samples assayed for Satmagan and Fe_2O_3.

11.6.3.4. Anorthosite Blank

Assays for anorthosite blanks show somewhat irregular results that are probably related to poor homogenization. On Figure 11-8 and Figure 11-9, Fe_2O_3, TiO_2 and V_2O_5 show a weak dispersion around the average at the 95% level of confidence in opposition to Satmagan. One point plots far off the line and could reflect a misidentified sample.

SiO_2 and Al_2O_3 are well clustered in or near the 95% level of confidence and MgO, CaO, Na_2O, K_2O are more dispersed. Although the variation is limited, it is noteworthy. The mixed assay results for different major oxide is hardly explainable and the use of anorthosite as bank is questionable.



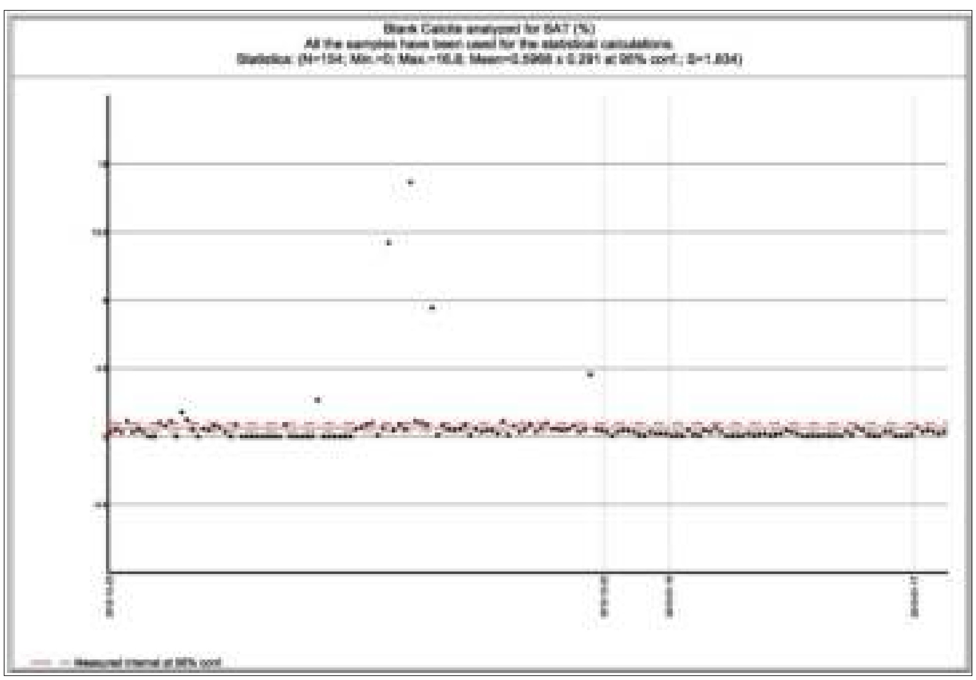


Figure 11-5: Satmagan assay results for blank halite

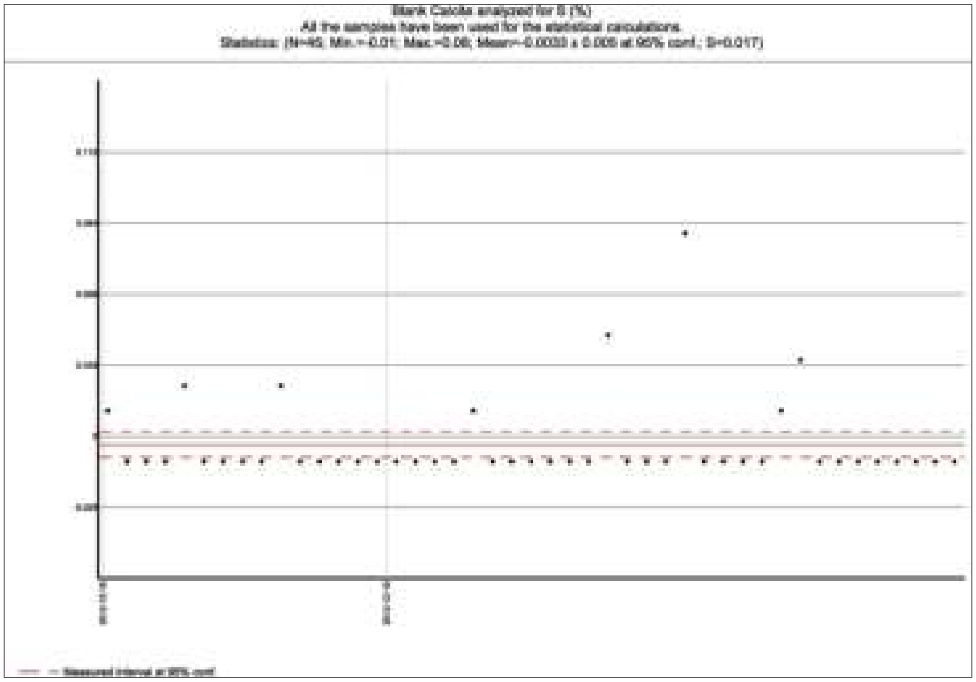
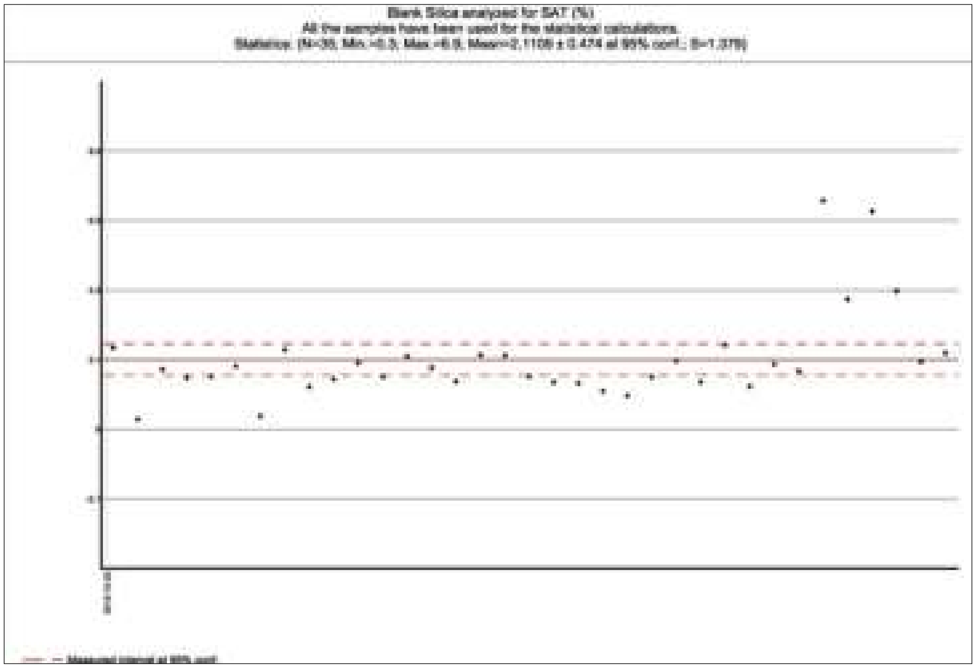


Figure 11-6: Satmagan and S assay results for blank calcite



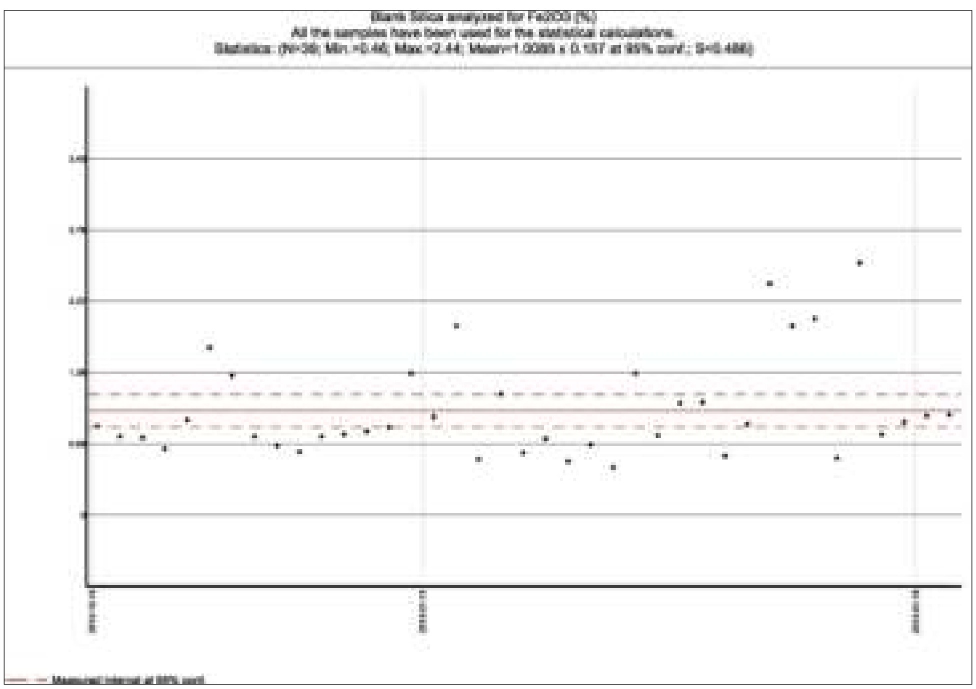
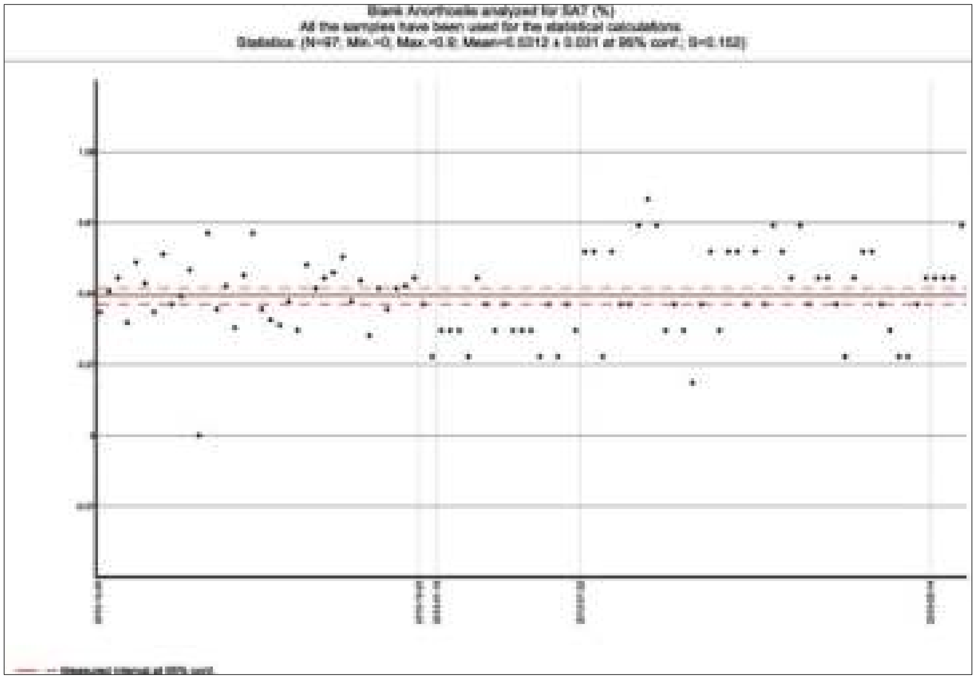


Figure 11-7: Satmagan and Fe_2O_3 assay results for blank quartzite



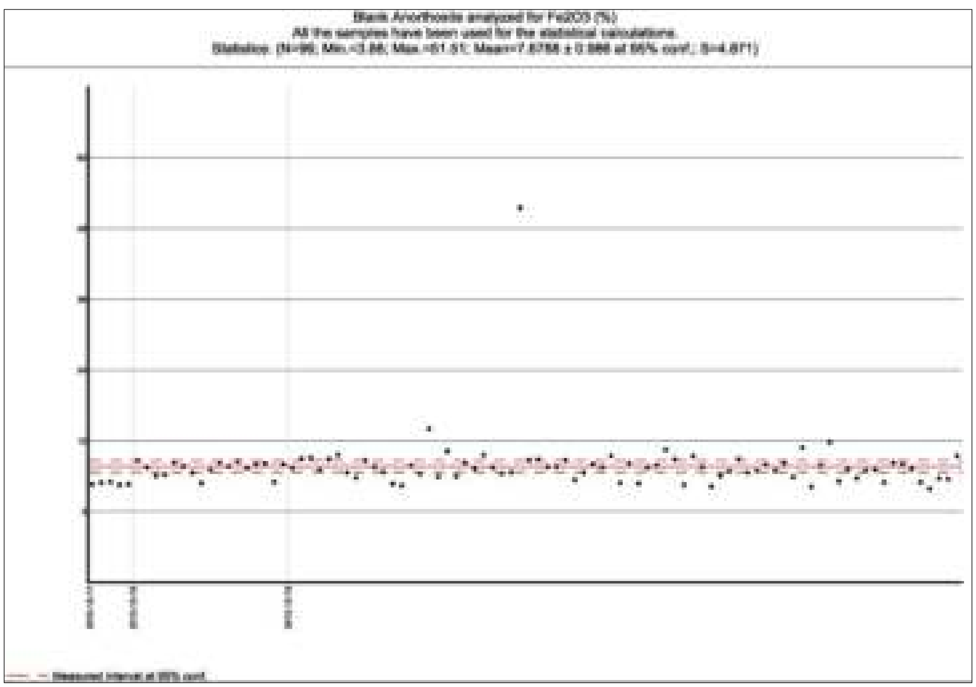
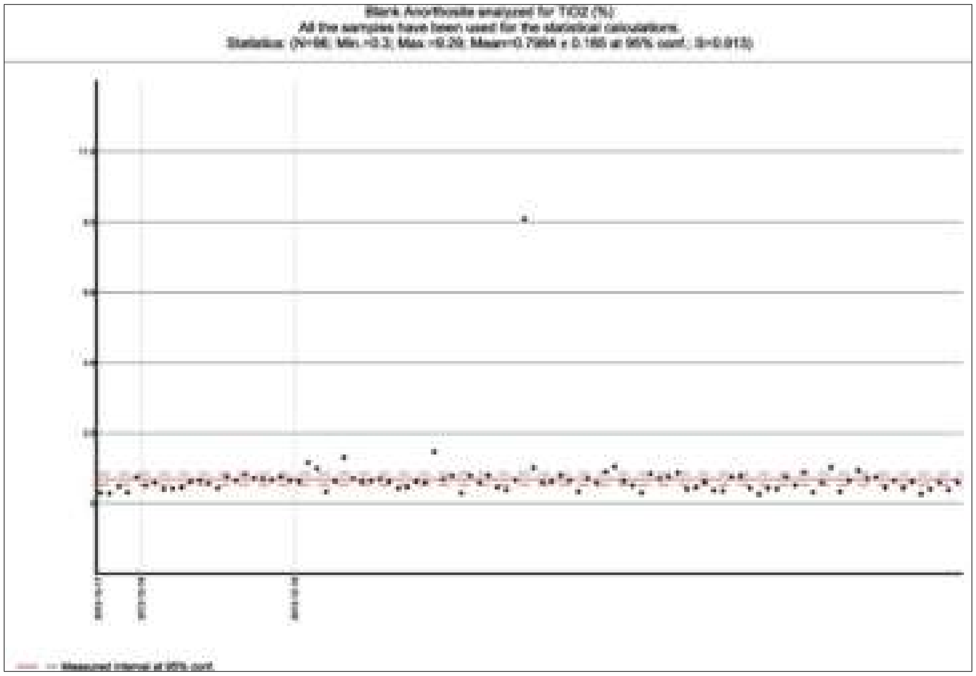


Figure 11-8: Satmagan and Fe_2O_3 assay results for blank anorthosite



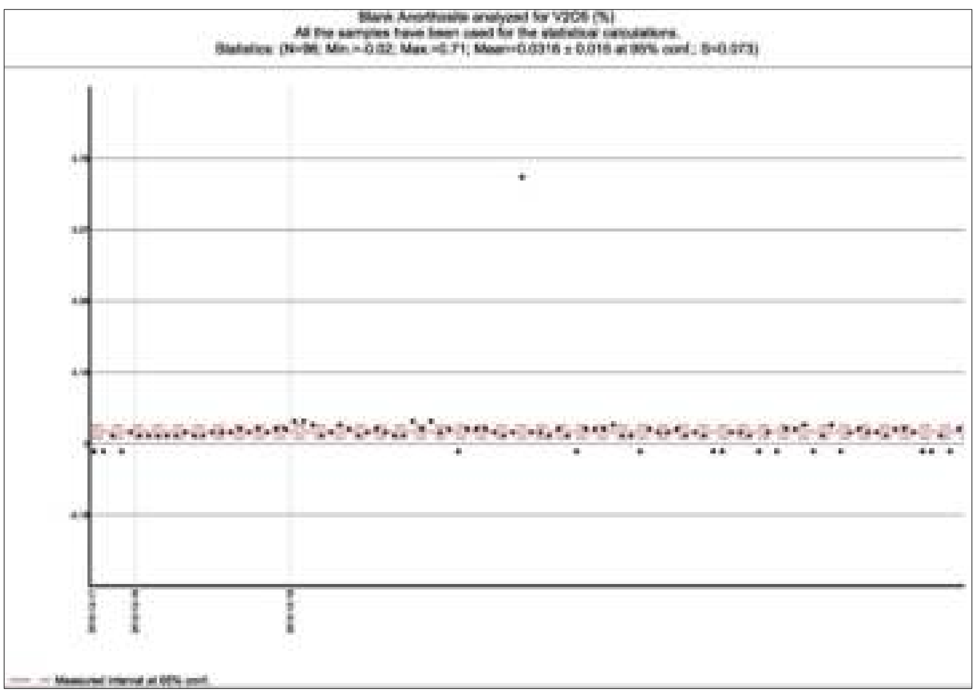


Figure 11-9: TiO_2 and V_2O_5 assay results for blank anorthosite

11.6.4. ALS in-Lab QA/QC Result

As mentioned at the beginning of this chapter, ALS Chemex employed an in-lab QA/QC control that included 488 duplicates and 1666 standards and blanks. The large number of control samples inserted into the sample stream is related to the use of specific standards for independently assaying the major oxides including Fe_2O_3, FeO and elemental S.

11.6.4.1. ALS In-Lab Duplicate

Figure 11-10 and Figure 11-11 are graphs of original vs. duplicate assay results for Fe_2O_3, TiO_2, V_2O_5 and S.

Graphs in Figure 11-10 and Figure 11-11 show a perfect linear correlation between original and duplicate assays for Fe_2O_3, TiO_2, V_2O_5 and S with R^2 above 0.996.

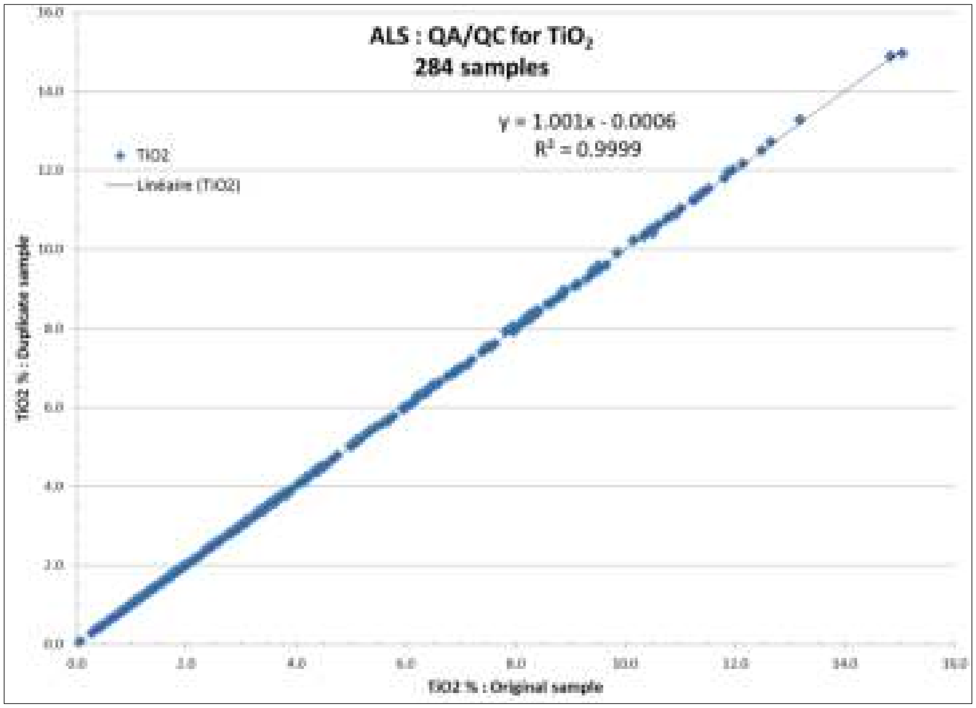
11.6.4.2. ALS In-Lab Standard and Blank

As aforementioned, ALS Chemex is an accredited laboratory and operates its own internal QA/QC program. It's internal QA/QC for the 2010 through 2012 programs were similar.

During the 2010, 2011 and 2012 drilling programs, ALS utilized 1,314 standards and 352 blank assays. Nine different standards were reported for QA/QC, with six comprising the bulk of the control. The standards used are listed in Table 11-7 with their certified assay values. ALS does not report the nature of the material that was used in their internal blanks.

Figure 11-12 and Figure 11-13 present graphic summaries of ALS standard and blank assay results for Fe_2O_3, TiO_2, VeO5 and S.

As shown on Figure 11-12 and Figure 11-13, the standards and blanks generally performed well as indicated by the clustering of results around averages which are close to the Certified Reference values summarized in Table 11-7.



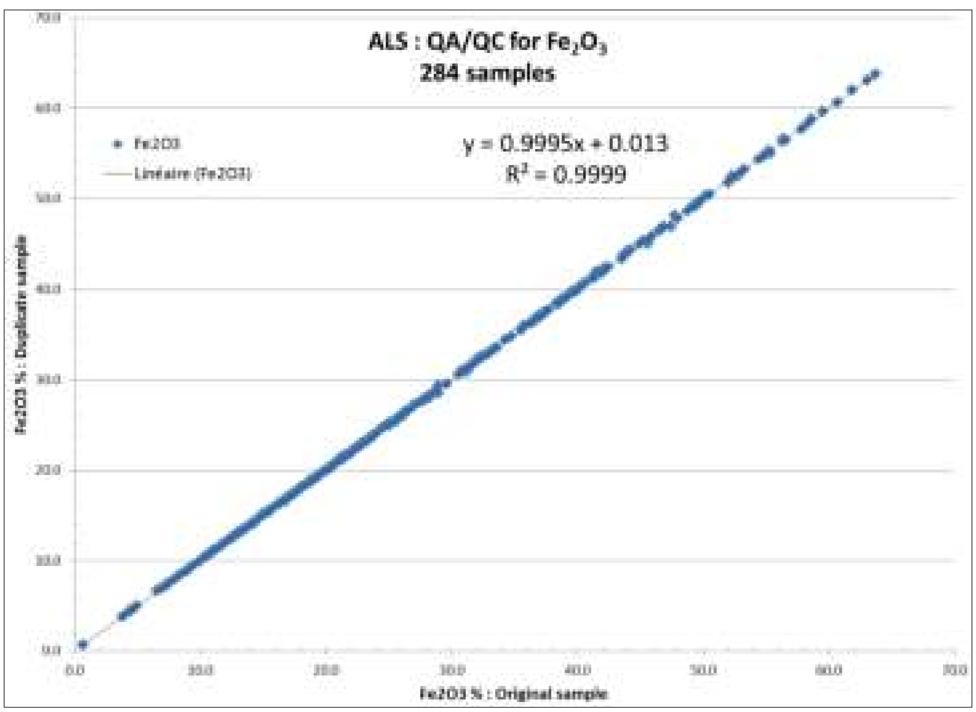


Figure 11-10: ALS Assay results for Fe_2O_3 and TiO_2 for in-lab duplicate

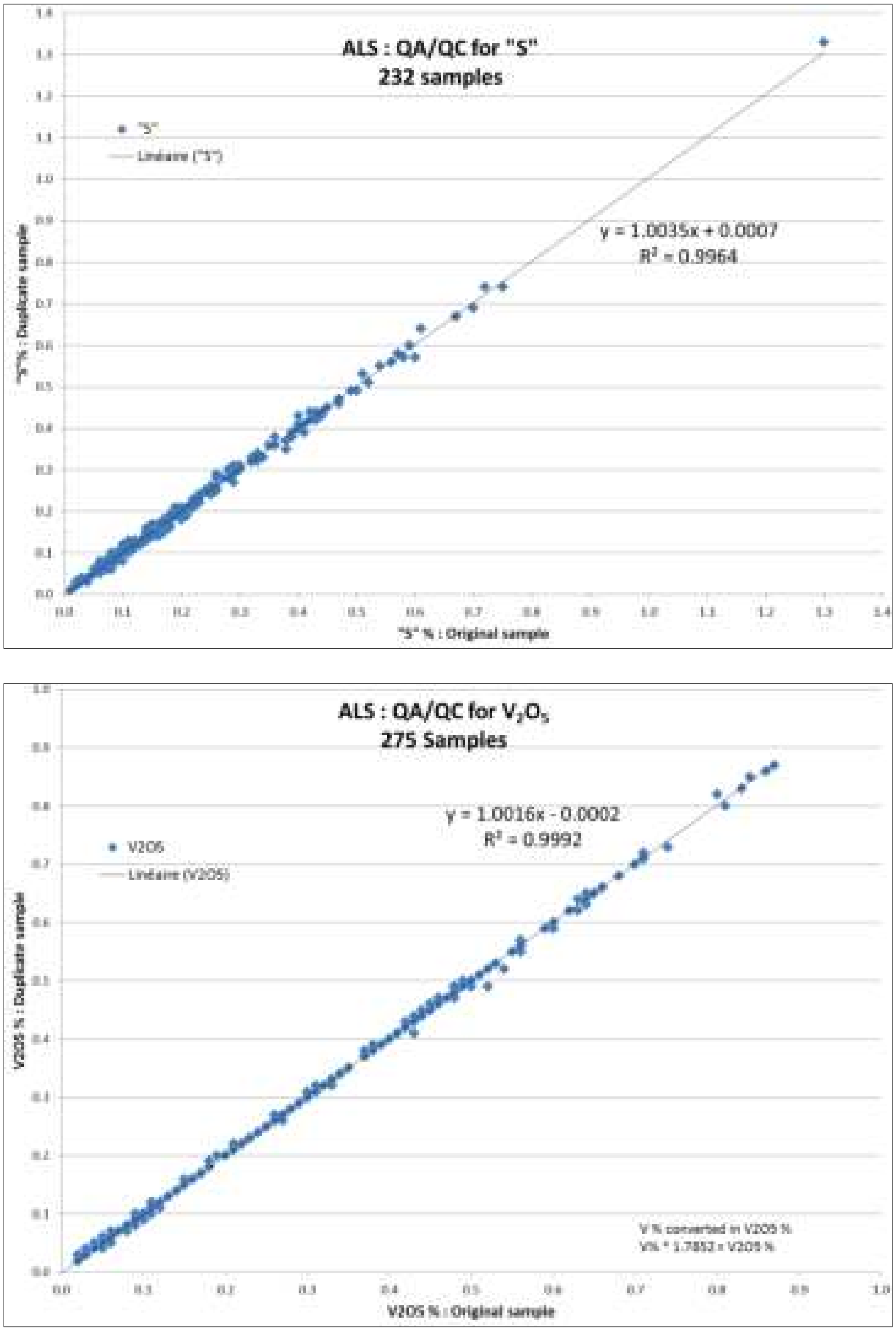
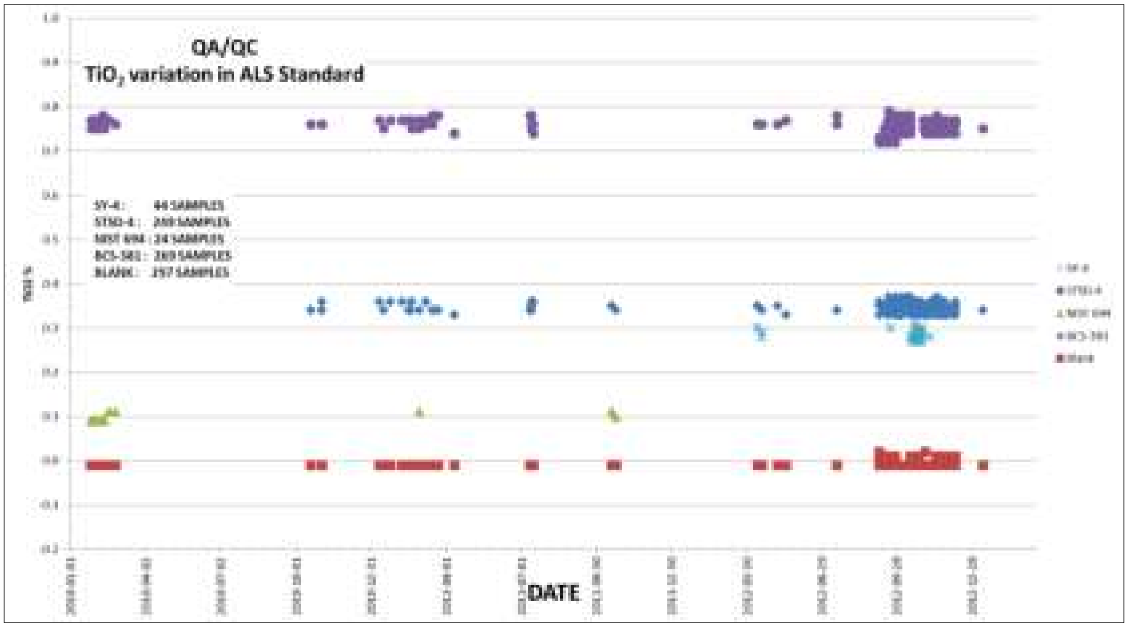


Figure 11-11: ALS Assay results for V_2O_5 and S for in-lab duplicate

Table 11-7: List of ALS Main in-lab standard with their certified values and the average and standard deviation calculated from BlackRock certificates

Standard	Description	SiO_2 %	Al_2O_3 %	Fe_2O_3 %	CaO %	MgO %	Na_2O %	K_2O %	Cr_2O_3 %	TiO_2 %	MnO %	P_2O_5 %	SrO %	BaO %	LOI %	Total %	V_2O_5 %	FeO %	S %
BCS-381	Certified Values	8.78	0.67		49.00	1.03			0.33	0.35	3.16	15.70					0.94	3.69	0.19
(Techlab)	Avg.	8.71	0.68	19.06	49.38	1.02			0.33	0.34	3.17	15.69					0.94		
	Std. Dev.	0.081	0.013	0.143	0.315	0.022			0.010	0.011	0.026	0.145					0.017		
STD-4	Certified Values	58.90	12.10	5.70	4.00	2.10	2.70	1.60		0.80	0.20	0.20			11.60	99.90			
(CANMET)	Avg.	58.74	12.05	5.68	4.02	2.13	2.73	1.59		0.760	0.19	0.22			11.290	99.65			
	Std. Dev.	0.199	0.080	0.056	0.029	0.046	0.031	0.030		0.013	0.005	0.004			0.103	0.274			
SY-4	Certified Values	49.90	20.69	6.21	8.05	0.54	7.10	1.66		0.287	0.108	0.131			4.56				
(CANMET)	Avg.	50.04	20.81	6.18	8.02	0.54	7.14	1.64		0.29	0.10	0.13			4.55				
	Std. Dev.	0.586	0.111	0.045	0.060	0.019	0.066	0.029		0.010	0.003	0.004			0.100				
NIST-694	Certified Values	11.20	1.80	0.79	43.60	0.33	0.86	0.51			0.0116	30.20					0.31		
((NIST*)	Avg.	11.61	1.86	1.88	42.56	0.60	0.85	0.540			0.11	29.04					0.31		
	Std. Dev.	0.045	0.007	0.011	0.130	0.006	0.009	0.011			0.000	0.064					0.009		
UTS-1	Certified Values																		1.00
(CANMET)	Avg.																		0.98
	Std. Dev.																		0.020
UTS-2	Certified Values																		3.23
(CANMET)	Avg.																		3.260
	Std. Dev.																		0.063

* National Institute of Standards and Technology



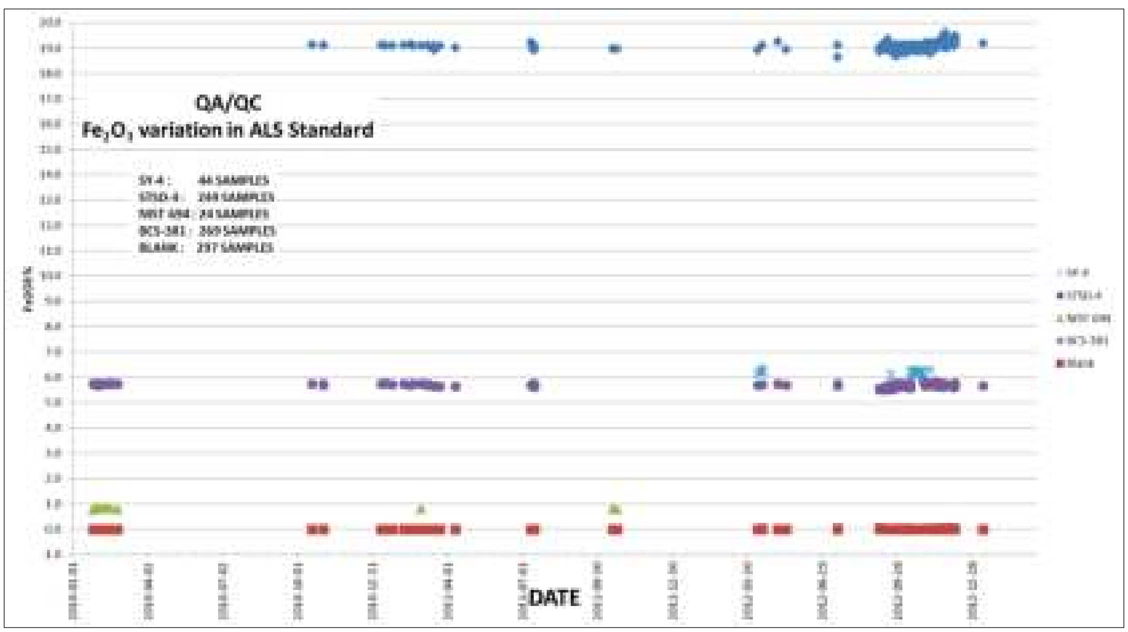
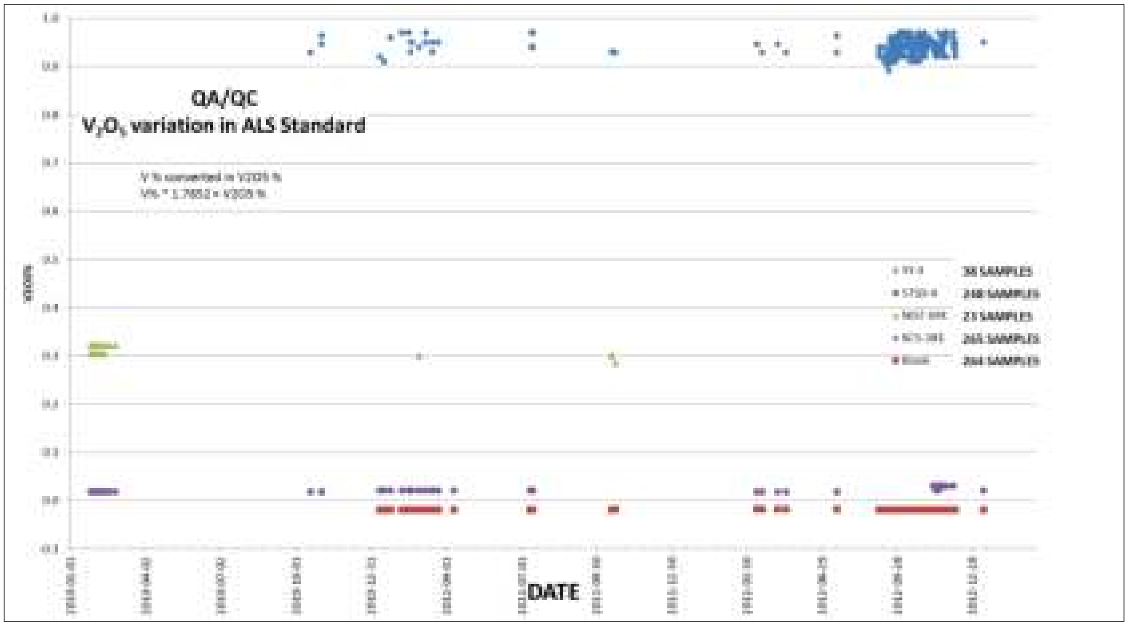


Figure 11-12: ALS Fe_2O_3 and TiO_2 Assay results for in-lab standard and blank assays



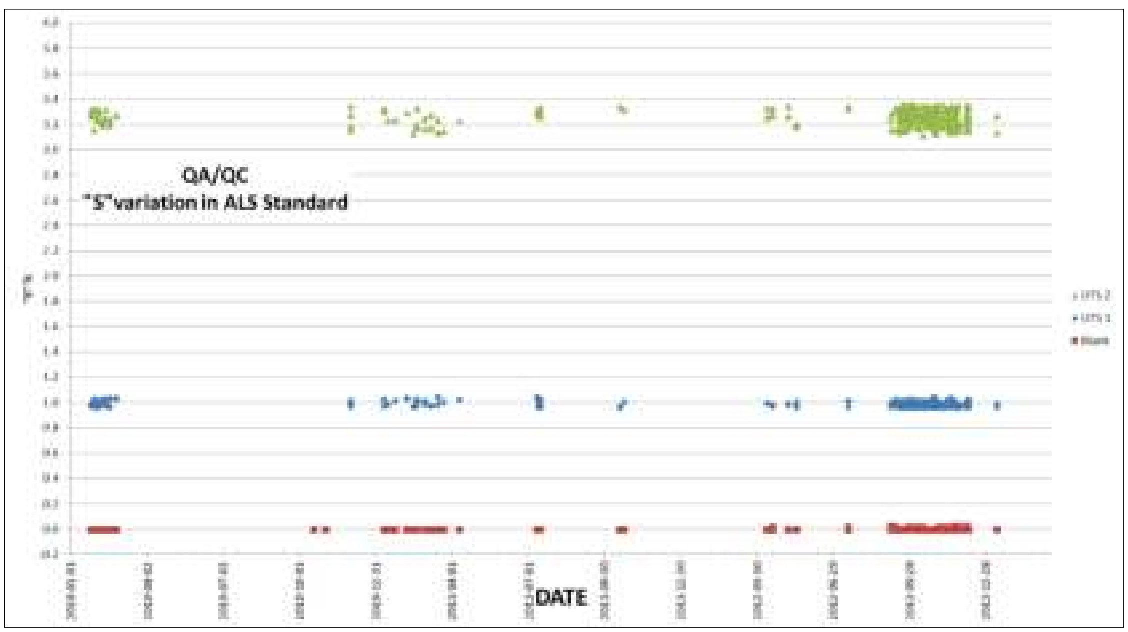


Figure 11-13: ALS V_2O_5 and S Assay results for in-lab standard and blank assays

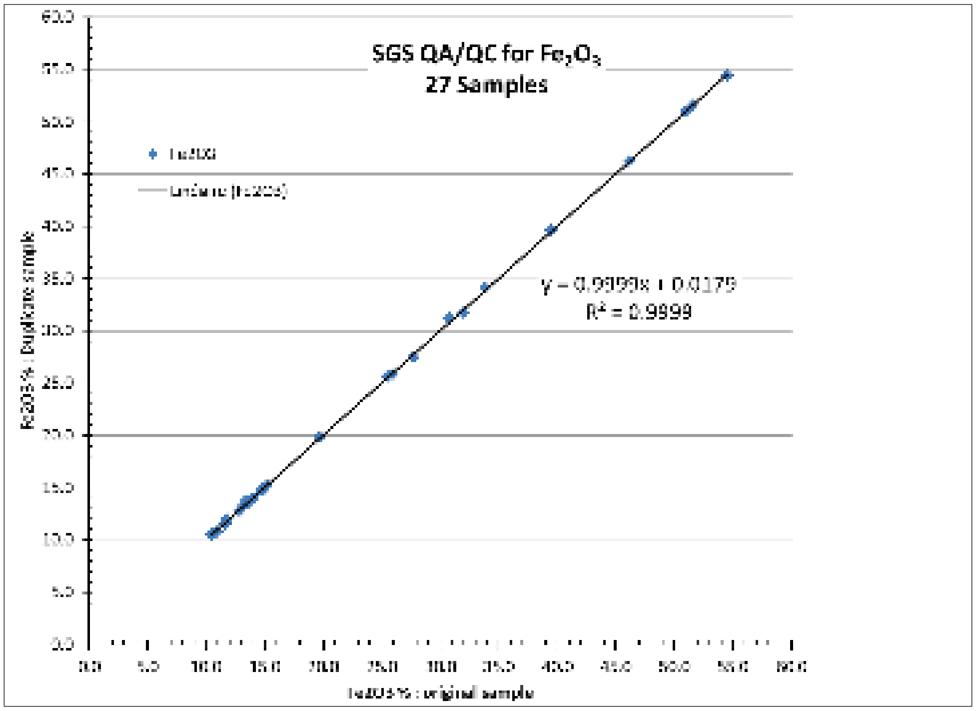
11.6.5. SGS In-Lab QA/QC Duplicate Results

As aforementioned, SGS Canada Inc. is an accredited laboratory and operates its own internal QA/QC program. No blank or standard assay results are available from SGS for the 2010 to 2012 BlackRock diamond drilling programs. No duplicate assay information is available for the 2010 and 2011 Satmagan testing at SGS. No samples were sent to SGS in 2010 for routine WRA assay.

In 2010, BlackRock sent 147 samples to SGS for DTA and 10 DTA duplicate samples were processed. The latter are not part of this SGS QA/QC section. No duplicate samples are available for the 2011 DTA testing. No samples were sent to SGS in 2011 for routine WRA assay.

For the 2012 diamond drilling program, SGS processed 27 WRA and 79 Satmagan duplicate samples. Comparison of original and duplicate assay results for Satmagan, Fe_2O_3, TiO_2 and V_2O_5 are illustrated in Figure 11-14 and Figure 11-15.

In all the graphs in Figure 11-14 and Figure 11-15, correlation between original and duplicate assay results is very strong with an R^2 above 0.999.



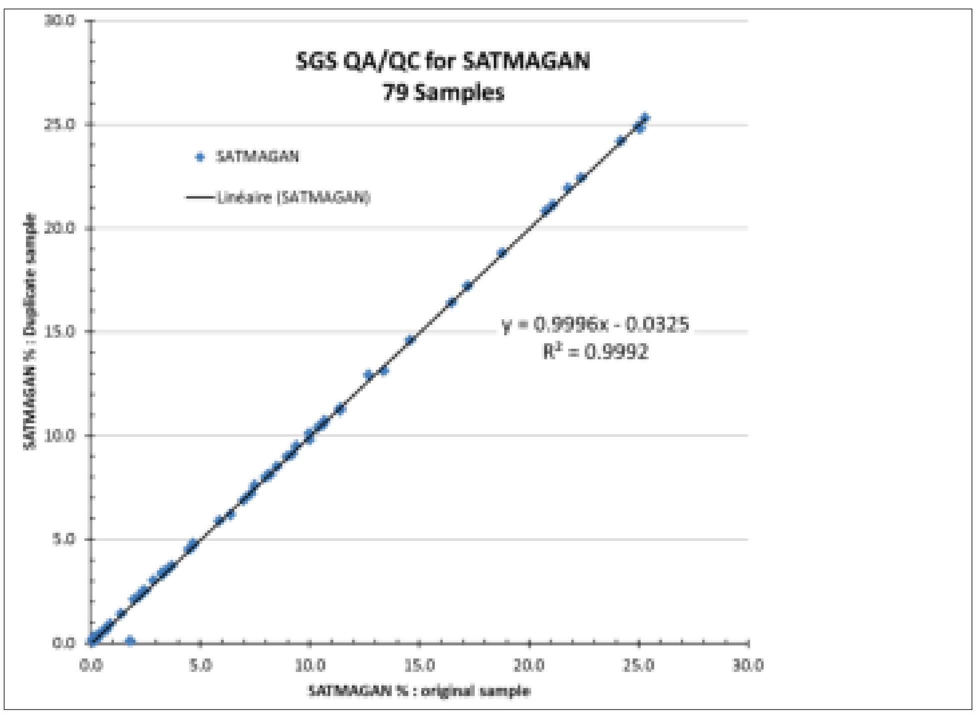
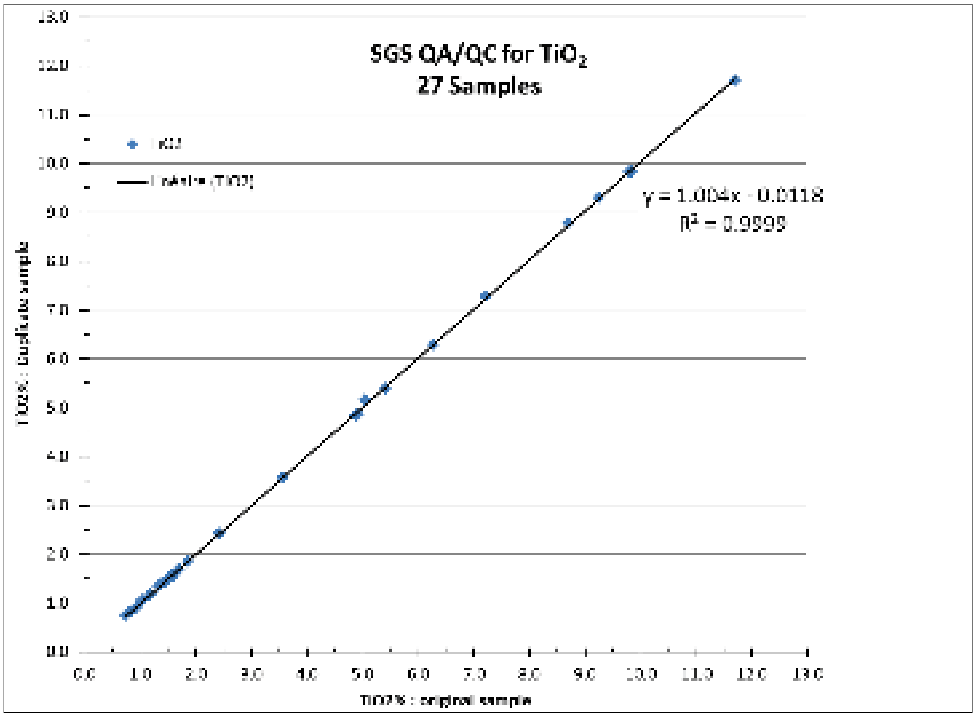


Figure 11-14: Satmagan and Fe_2O_3 Assay results for SGS in-lab duplicate



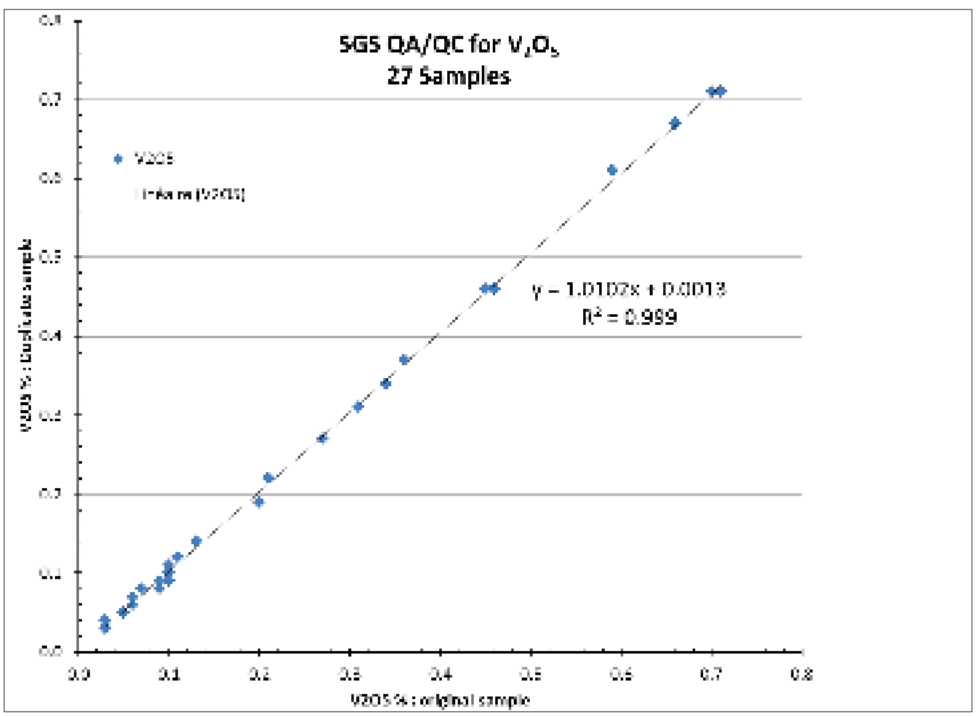


Figure 11-15: TiO_2 and V_2O_5 Assay results for SGS in-lab duplicate

11.6.6. Comparison between SGS and ALS

For the 2012 diamond drilling program, 151 samples were sent to both ALS Chemex and SGS Canada and assayed for WRA and S. Results are presented in Figure 11-16 and Figure 11-17 for Fe_2O_3, TiO_2, V_2O_5 and S.

Correlation between ALS and SGS is excellent. Mainly all the results for each oxide are near or on the 1:1 ratio line and with an R^2 higher than 0.99.

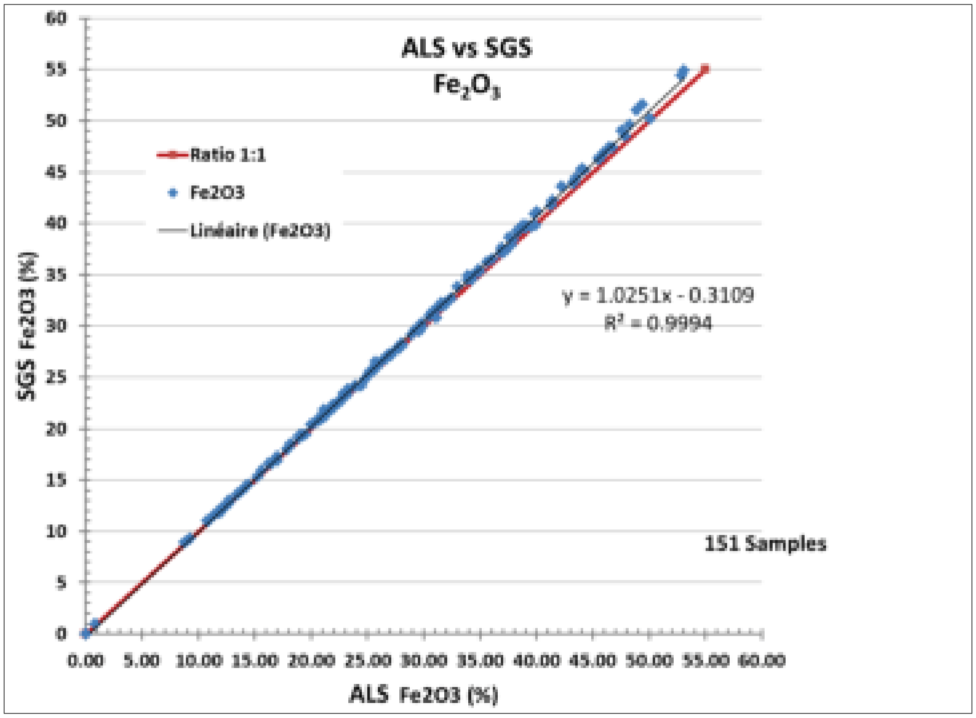
Only the correlation for Cr_2O_3 indicates a weak dispersion with an R^2 value of 0.874 and this may be related to the values lying near the detection limit of the instrument.

P_2O_5 and MnO both have a good correlation with an R^2 at 0.95 and 0.97 respectively, but the correlations deviate from the 1:1 ratio line indicating a slight difference in the calibration curves between the instruments used in the two labs.

11.6.7. Comparison between COREM and ALS

For the 2012 diamond drilling program, 117 samples were sent to both ALS Chemex and COREM Inc. Results are presented in Figure 11-18 and Figure 11-19 for Fe_2O_3, TiO_2, V_2O_5 and S.

- Correlation between ALS and COREM is excellent. Mainly all the results for each oxide are near or on the 1:1 ratio line and with an R^2 higher 0.99;
- Only correlation for Cr_2O_3 indicates a weak dispersion with an R^2 value of 0.857 and again this is probably related to the detection limit;
- P_2O_5 and MnO both have a good correlation with an R^2 at 0.95 and 0.97 respectively but correlations deviate from the 1:1 ratio line;
- P_2O_5 has a good R^2 correlation at 0.95 and the deviation from the 1:1 ratio line is quite similar to the one observed for ALS and SGS. This suggests a difference between ALS and the two other laboratories.



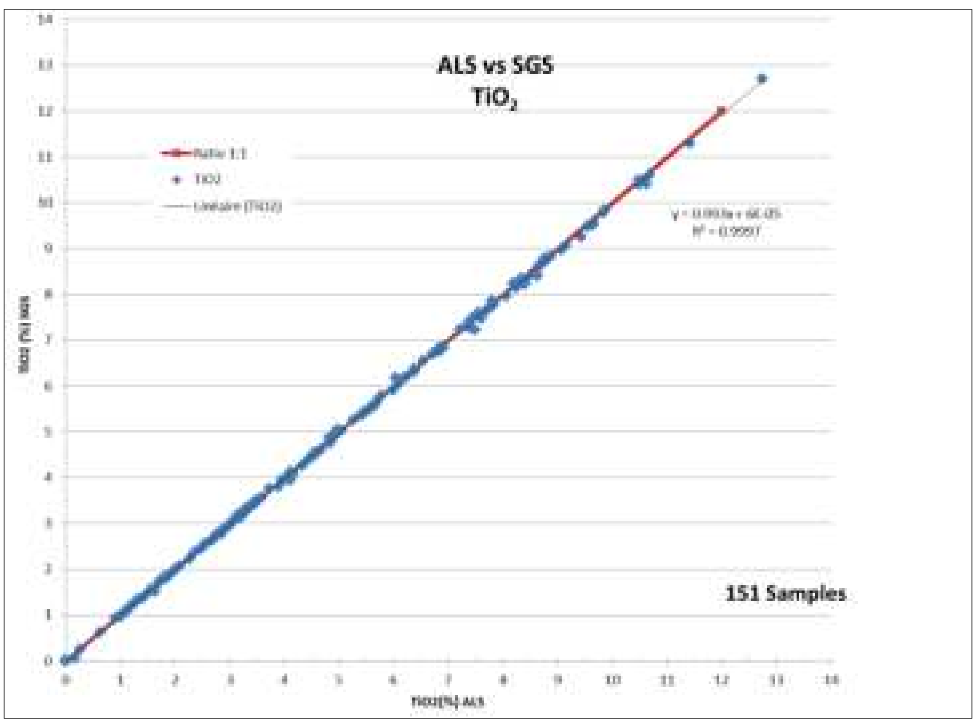
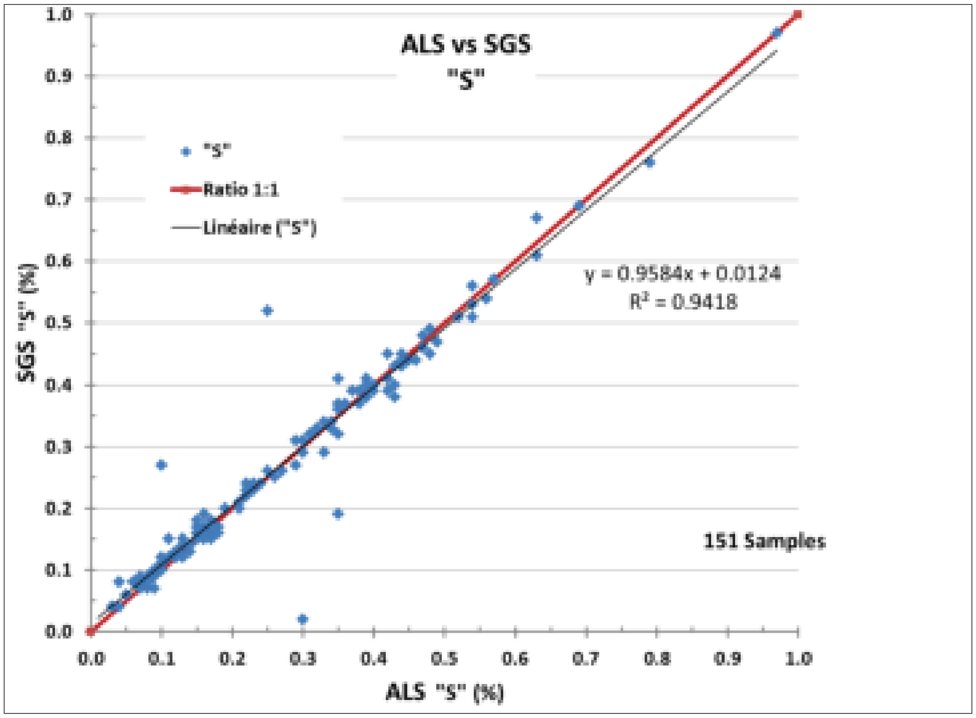


Figure 11-16: Comparison of ALS and SGS Assay Results for Fe_2O_3 and TiO_2



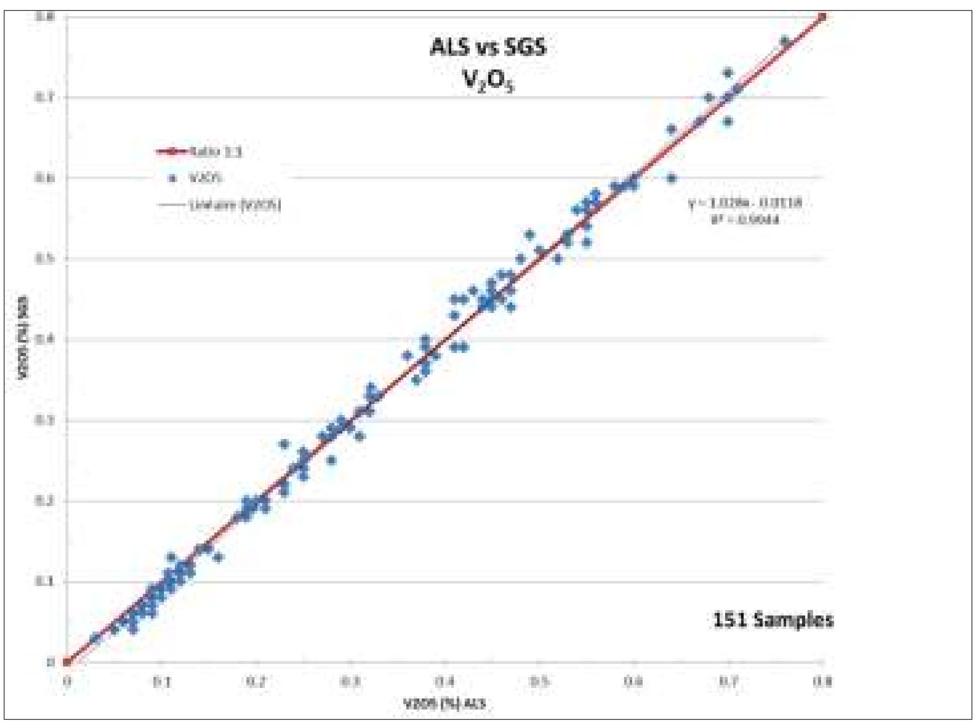


Figure 11-17: Comparison of ALS and SGS assay results for V_2O_5 and sulphur "S"

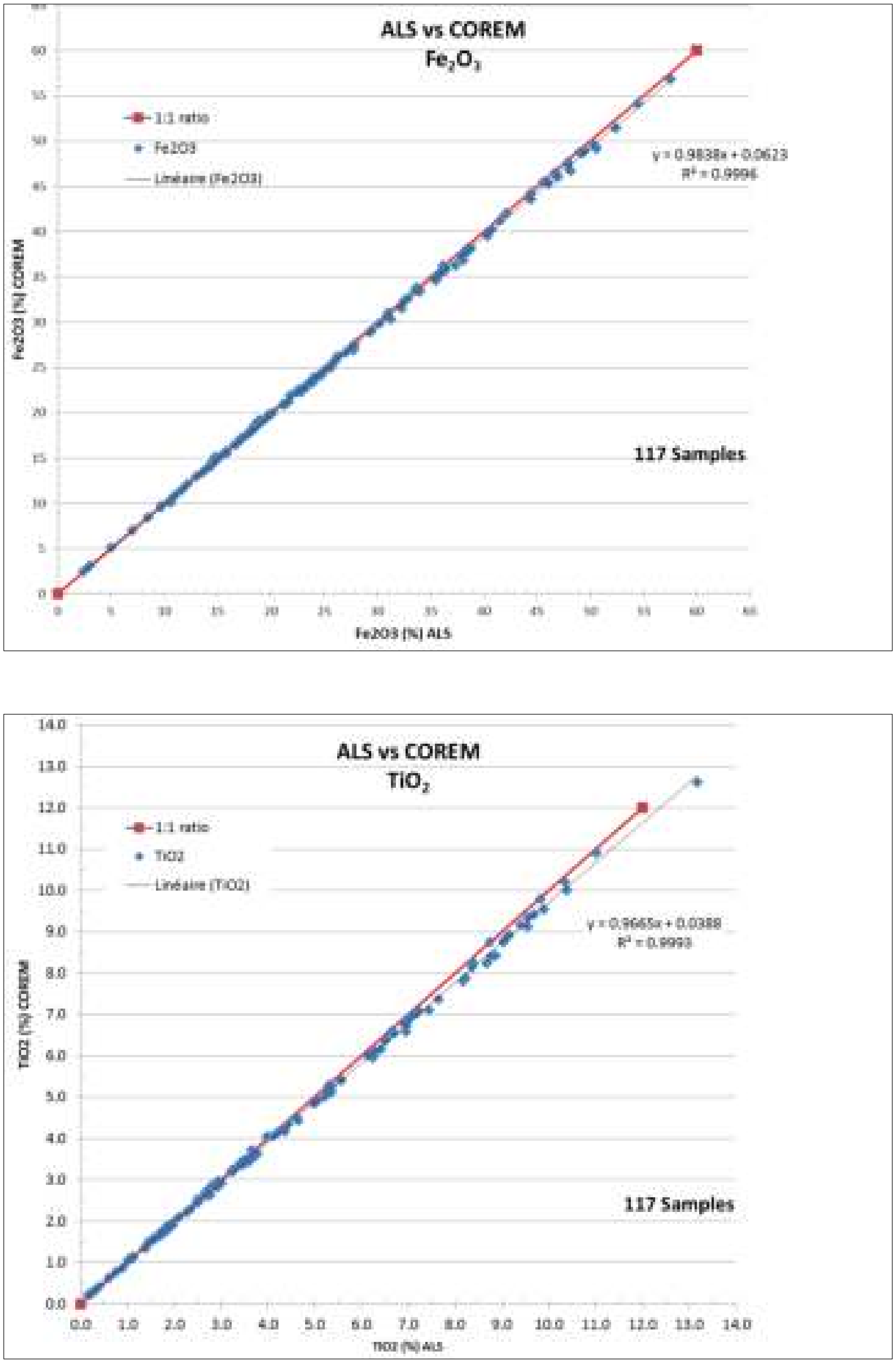
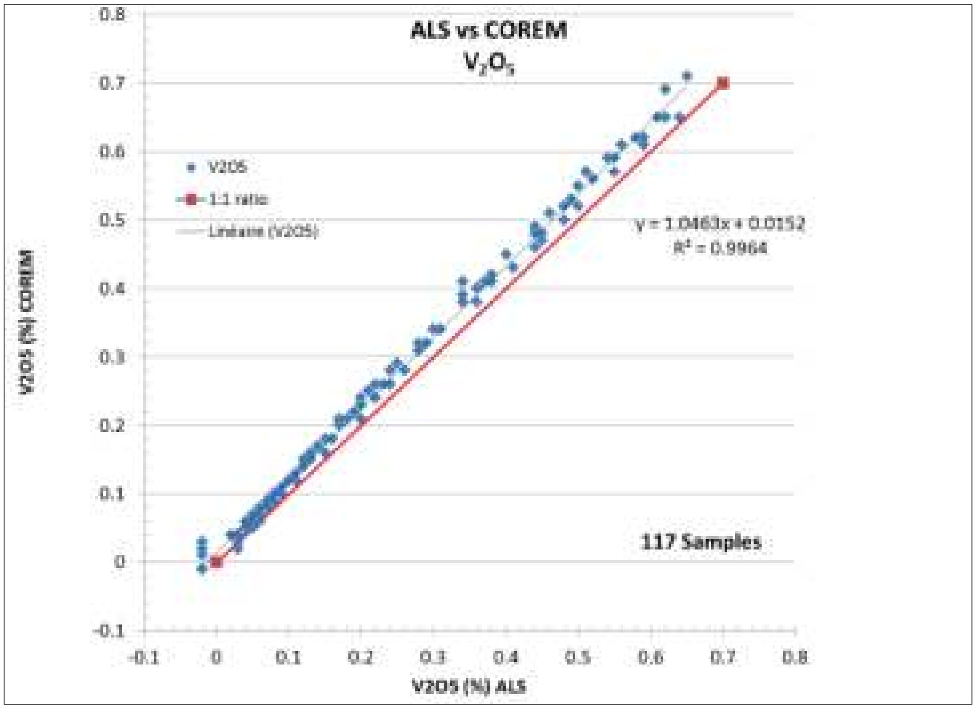


Figure 11-18: Comparison of ALS and COREM Assay results for Fe_2O_3 and TiO_2



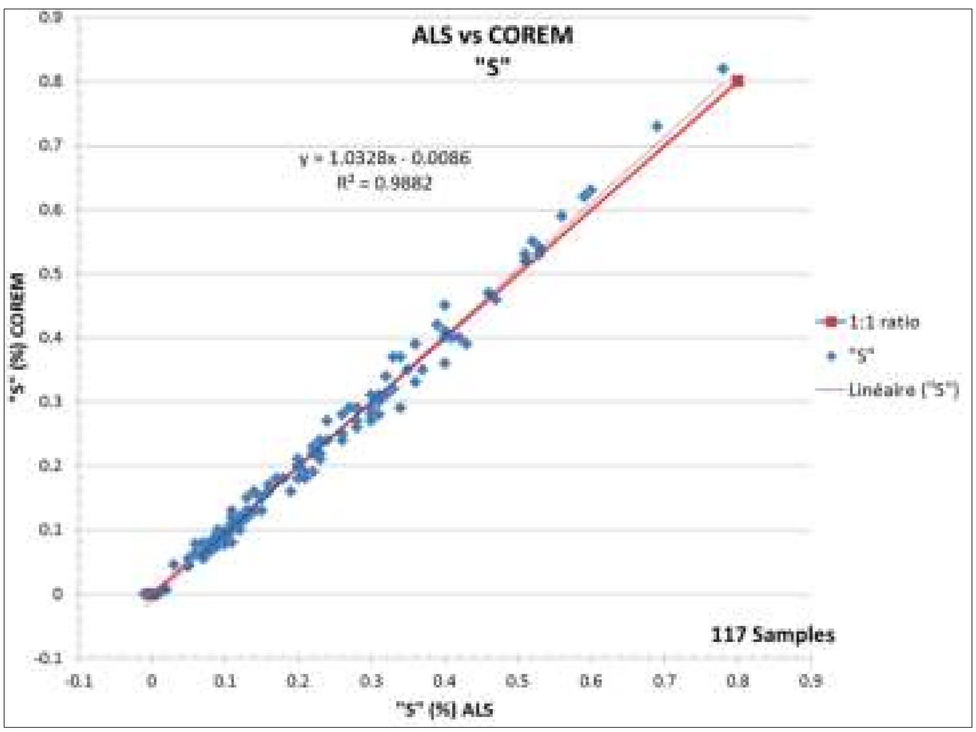


Figure 11-19: Comparison of ALS and COREM Assay results for V_2O_5 and sulphur "S"

11.6.8. Satmagan Re-assaying at COREM

As part of its QA/QC program, BlackRock has sent back to COREM in 2012, 319 of the 2010's samples to be re-assayed for Satmagan. The material consisted of pulp.

Figure 11-20 indicates a good correlation between the 2010 Satmagan assay results and the 2012 with a linear correlation almost on the 1:1 ratio line and an R^2 value of 0.968.

On the other hand, all 2010's samples with a Satmagan values above 10% returned a 5-10% higher Satmagan value in 2012. Validation is in progress with COREM.

Six samples (2%) returned completely different Satmagan values. This is probably related to misidentified samples in 2010 or in 2012 or possible contamination. Investigation is in progress.

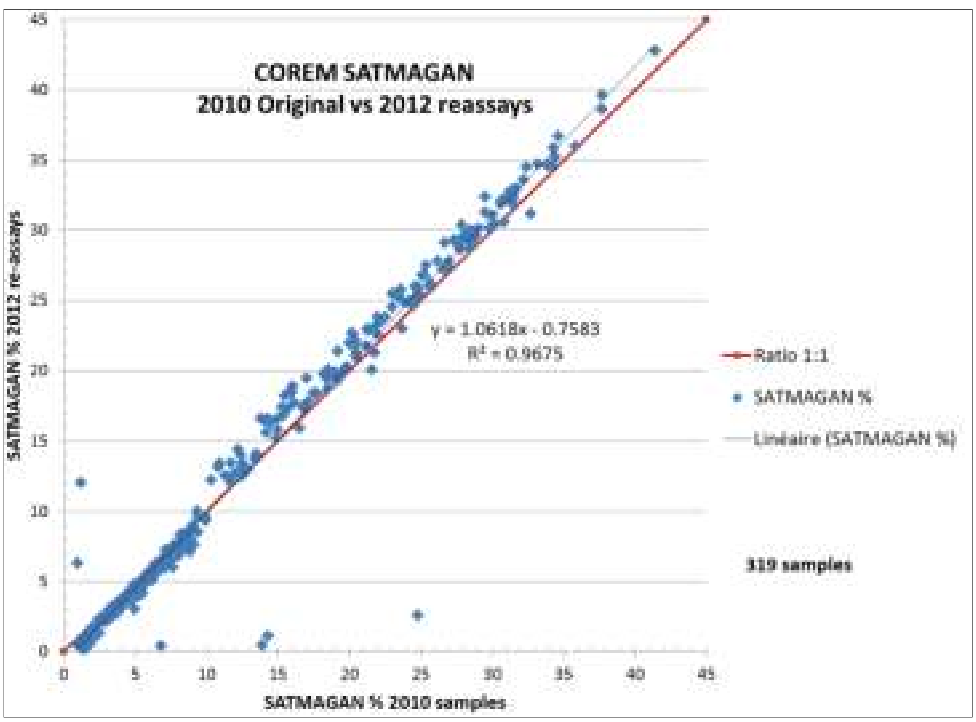


Figure 11-20: Comparison of 2010 Satmagan assays 2012 re-assays at COREM

11.6.9. Comparison of Satmagan assays at SGS and COREM

As part of its QA/QC program, for the 2012 diamond drilling program, 136 samples were sent to both SGS Canada and COREM Inc. and assayed for Satmagan.

Figure 11-21 generally indicates a good correlation between SGS and COREM for Satmagan assays. Correlation is linear and straddles the 1:1 ratio line with an R^2 value of 0.938. However seven samples show significant differences for Satmagan values between COREM and SGS. This represents about 5% of the cross-checked samples and should be investigated further by BlackRock.

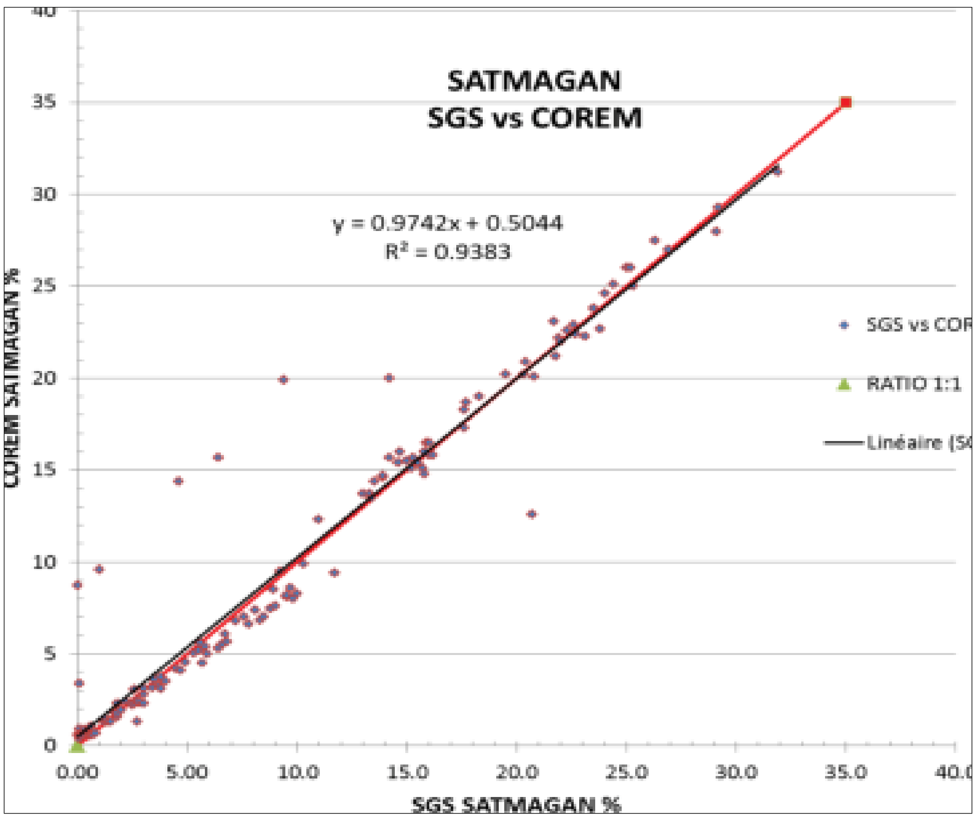


Figure 11-21: Comparison of Satmagan results between SGS and COREM

11.6.10. Summary of the BlackRock QA/QC Program

BlackRock QA/QC program succeeded in validating the laboratory results for all providers examined. The following conclusions that can be drawn from this examination QA/QC data indicates that:

- WRA assays at ALS CHEMEX were consistent and reliable from all drilling programs from 2010 to 2012;
- Satmagan and WRA assays at COREM Inc. were consistent and reliable from all drilling programs from 2010 to 2012;
- Satmagan and WRA assays at SGS Canada were consistent and reliable from all drilling programs from 2010 to 2012;
- An excellent correlation for WRA assays between ALS Chemex and COREM and ALS Chemex and SGS;
- An excellent correlation for Satmagan assays between SGS Canada COREM;
- All assay data from the accredited labs can be considered robust and reliable for use in resource estimation.

11.7. Database

In the fall of 2012, BlackRock geologists assembled a comprehensive, validated database using GeoticLog as the data entry software. GeoticLog is a module within a software package developed by Geotic Inc. (www.geotic.ca) and was used to organize and download all assay information and to manage the input of related drilling information including surveying and core description in a single database. BlackRock geologists then used GeoticGraph for laying out drilling information and creating drill sections and level plans, and GeoticCAD for geological interpretation of drill sections and level plans.

GeoticLog works under Windows XP to Windows 7 and data is saved in Access or SQL format. It includes modules for computer core logging, sampling, importing assay results and surveying data directly from Excel certificates. More importantly, it provides complete data validation and certificate management, QA/QC management and graphs. The application includes a user's rights management interface to control data access and make data secure. Finally, all data can be exported into the Gemcom database format.

Data collected and imported in GeoticLog were:

- Borehole collar surveys;
- Borehole deviation surveys;

- Modified original Excel drill logs including rock description, assaying meterage and RQD;
- Original PDF WRA and Satmagan certificates and Excel files from ALS, SGS and COREM. Excel files were directly imported into GEOTIC using the importation application;
- In-field QA/QC Satmagan and WRA duplicates and blanks;
- Photonic Knowledge data.

BlackRock geologists have rechecked and validated original data.

The master database is stored on the BlackRock main server in Montreal. Access and modification to the database is restricted. All certificates and original information and data are also stored on the main server. Upon final validation, all the drilling information and data were imported in GEMCOM, for calculating the new mineral inventory.

11.8. Opinion and Comments Regarding Sampling and Assaying Protocols

Over the years, BlackRock has developed a comprehensive set of sampling and analytical protocols. In fact, since acquiring the Southwest and Armitage properties and form their first exploring program on the properties, BlackRock has made every effort to set up proper sampling and analytical protocols and to immediately address any issues regarding it. As part of their QA/QC program, BlackRock has instigated numerous crosschecks to find discrepancies in analytical results between samples themselves, between the different laboratories used and even within the laboratories themselves. BlackRock has introduced numerous field duplicates, field blanks, lab duplicates and blanks to build their robust QA/QC program. Part of this QA/QC program involved the creation and maintenance of a proper relational database. The use of a relational database greatly facilitates the check for errors in the vast amount of data collected.

It is of the QP's opinion that this robust sampling protocols and QA/QC program set up by BlackRock is adequate for this advanced project.

12. Data Verification

As part of the preparation of independent technical reports for BlackRock, SGS Geostat required numerous site visits and independent sampling to certify that the work done by BlackRock met best practice guidelines. Since this project was done over a long period of time, numerous site visits were required. The site visits took place at various times in the exploration stages in 2010, 2011 and 2013. Site visits consisted of visiting the drill during a drilling campaign, visiting the logging and splitting facilities, the TJCM sampling and analytical facilities and meeting with various BlackRock personnel involved in the project.

Site visits were performed on March 17 and April 26-29, 2010 by Vincent Cloutier, P.Eng., for the Southwest Deposit. This visit involved the aforementioned site visit activities and the collection of 54 independent samples for the purposes of independent analytical testing at SGS Lakefield. Vincent Cloutier's observations and results from his visit were reviewed by Claude Bisaillon. A site visit for the Armitage Deposit drilling and sampling campaign was performed by Claude Bisaillon, Eng., on February 23-25, 2011. This visit involved the aforementioned site visit activities and the collection of 35 independent samples for analytical testing at SGS Lakefield. On May 28-30, 2013 a set of independent samples were collected during a site visit. These samples were used on a confirmatory basis for the core testing at SGS Lakefield and were not used as the basis of the database design. The 2013 data verification focused on two areas: 1) Validation of the database and relations between each table (collars, deviations, lithologies and assays); 2) Independent control sampling on core samples (1/4 core).

12.1. Database Validation

The database transferred to SGS Geostat for resource estimation purposes was created by BlackRock using Geotic©, but provided to SGS in Excel format. SGS Geostat proceeded to import the database to Geobase© for validation and corrections. The database contains 251 holes and trenches, 9,933 survey measurements, 8,856 assays and 6,489 lithologies.

No major issues were found in the database during the automated validation process conducted by SGS using Access and Genesis©. Deviations at 0 m depth were also removed from the deviation table and transferred to collar orientation columns in the collar table, which is the database structure specification for Genesis©.

As a result of its data validation efforts, SGS believes that the drillhole data representing the magnetic iron mineralization intersected by drilling at both the Southwest and Armitage Deposits are appropriate for use in the preparation of Mineral Resource estimates.

12.2. Independent Control Sampling

The 1/4 core for the chosen samples was sent to SGS Lakefield, Ontario. The samples were characterized at SGS Lakefield to determine WRA, Satmagan and density. Results for all elements of interest were reported to SGS Geostat. In 2010, 38 samples were also subjected to DTA. Details for the 2010, 2011 and 2013 programs are provided below.

12.2.1. Independent 2010 SGS Samples – Southwest Zone

SGS Geostat, through Mr. Vincent Cloutier, performed independent sampling of the Southwest Zone between April 26 and 29 of 2010. The independent sampling was performed on quarter core (half of the halved cores).

A total of 54 samples were collected. From these, 38 were grouped into mineralized intervals and 16 more were grouped into the waste interval. The independent samples were shipped to SGS Lakefield in Ontario, Canada for assaying. The shipment was received at the SGS Lakefield site on May 3, 2010.

The 38 mineralized samples were prepared as depicted in Figure 12-1 and were submitted for DTA. The remaining 16 waste samples were prepared as depicted in Figure 12-2.

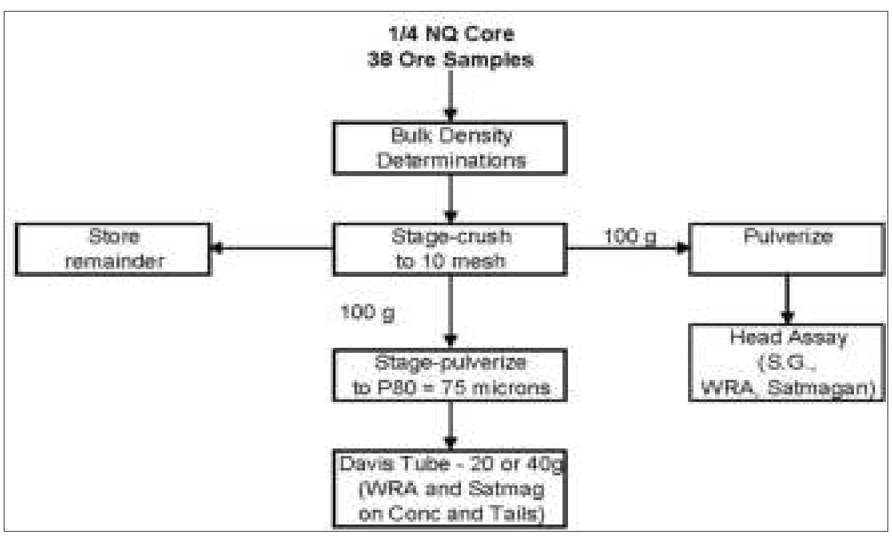


Figure 12-1: Sample preparation diagram – SGS independent ore samples

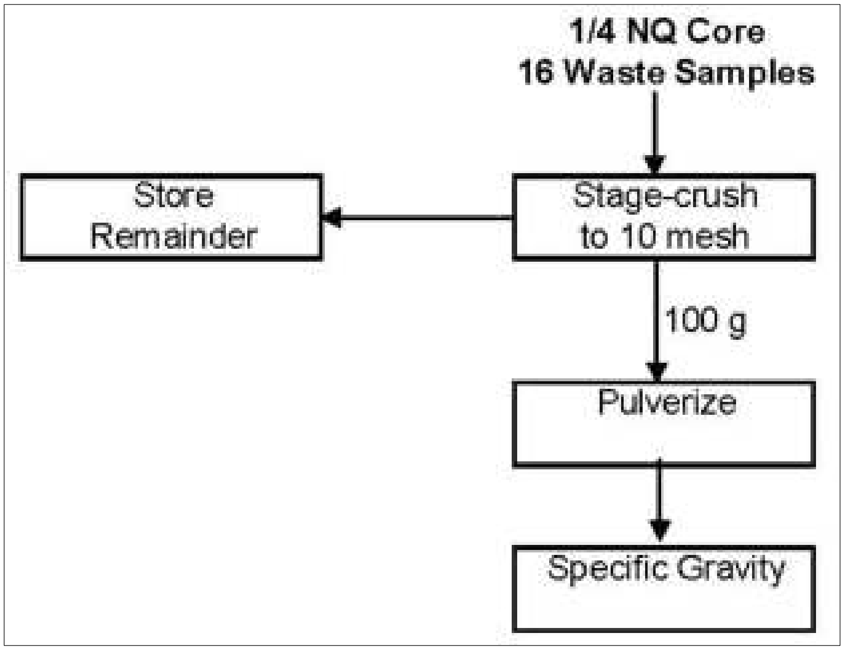


Figure 12-2: Sample preparation diagram – Waste Samples

12.2.2. Davis Tube Results

The graph in Figure 12-3 shows the cumulative frequency plot of reported WRA and Satmagan values for the DTA concentrate recovered from the 38 SGS independent ore samples. The DTA concentrates have total iron grades of 56.1% to 66.2%, with an average of 61.1%. Total Fe recovery in the concentrate averages 54.5%, but is wide, ranging between 6.5% and 75.8%. The average Satmagan grade is 73.8%, with a very good recovery of 96.7% (ranging between 88.1% and 99.7%).

The average TiO_2 and V_2O_5 grades in the DTA concentrates are 8.2% and 1.2%, respectively. The total weight recovery to the concentrate ranges between 0.55% and 53.8%, with an average of 28.9%.

The SiO_2 grade in the concentrate generally ranges from 1.1% up to 5.2%, although there is a single sample whose concentrate is as high as 10.8% SiO_2. The Al_2O_3 falls between 0.8 and 3.1% for all samples.

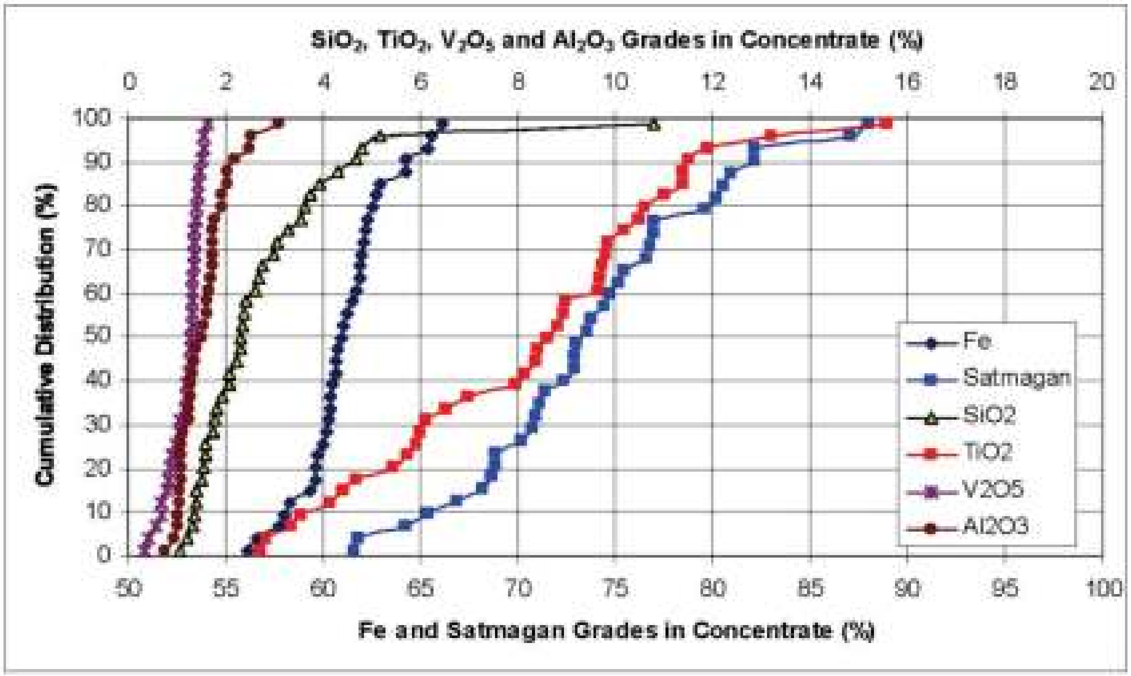


Figure 12-3: Characteristics of Davis Tube concentrates from SGS independent samples

These DTA results were used to develop relationships between the chemical assays and the magnetic recovery. The relationships between Satmagan of the head and weight, Total Fe, TiO_2 and V_2O_5 recoveries are presented in Figure 12-4. For typical magnetite ores, the relationships between Satmagan of the head and Fe and weight recoveries are linear with little scatter. For the BlackRock ore, the correlations are not as satisfactory as the ones observed for typical magnetite ores, due to the ilmenite and Ti-magnetite occurrences, but they are still acceptable. The correlations between Satmagan of the head and V_2O_5 and TiO_2 recoveries are considerably more scattered.

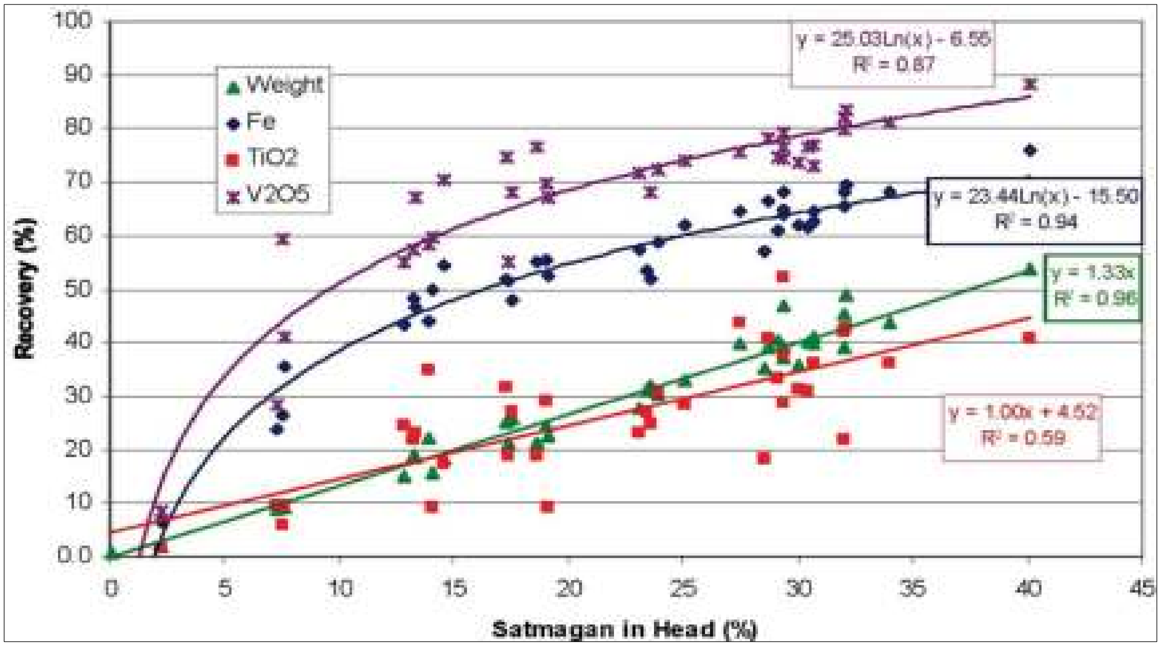


Figure 12-4: Relationships of Satmagan of head and recoveries from SGS independent samples

Thirty-seven pulp duplicates were sent by BlackRock to COREM as part of its QA/QC program for Satmagan results. Figure 12-5 shows the correlation of original and duplicated Satmagan values for these 37 duplicate samples. With an R2 of 0.999, the duplication is nearly perfect.

Values of the Satmagan, from the 38 independent samples sent to SGS Lakefield (from the quarter split) and the same samples sent to COREM (from the original half split), were also compared. Here again, the R2 of 0.962 indicates a good reproduction of the original Satmagan results (Figure 12-6).

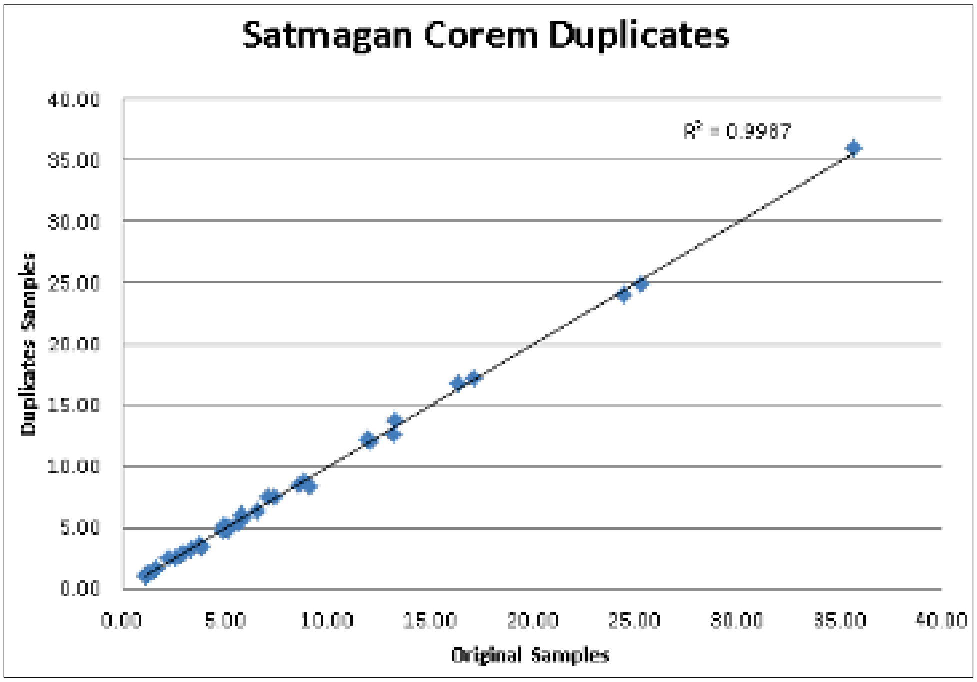


Figure 12-5: Satmagan duplicates at COREM

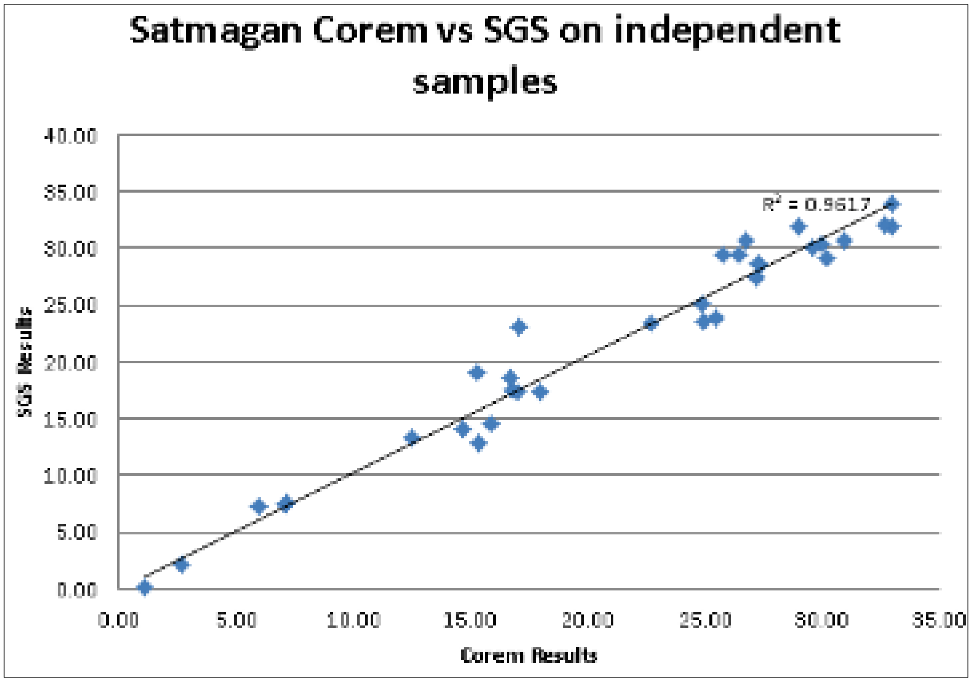


Figure 12-6: Satmagan duplicates at SGS Lakefield

Based on these data, it was concluded that the Satmagan results replicate extremely well in the same laboratory (COREM), and slightly less between two different laboratories (COREM and SGS Lakefield). Differences may originate in part from the heterogeneity of the rock sample (quarter vs. half core) or from the Satmagan itself.

12.2.3. Independent 2011 SGS Samples – Armitage Zone

The 2011 independent sampling program for the Armitage Zone was performed between February 23 and 25 by Mr. Claude Bisaillon, of SGS Geostat. As was done in the Southwest program, the independent sampling was performed on the quarter core (half of the halved cores). A total of 35 samples were collected. The independent samples were shipped to SGS Lakefield in Ontario, Canada for assaying. The shipment was received at the SGS Lakefield site on March 7, 2011, and was prepared as per the method illustrated in Figure 12-7.

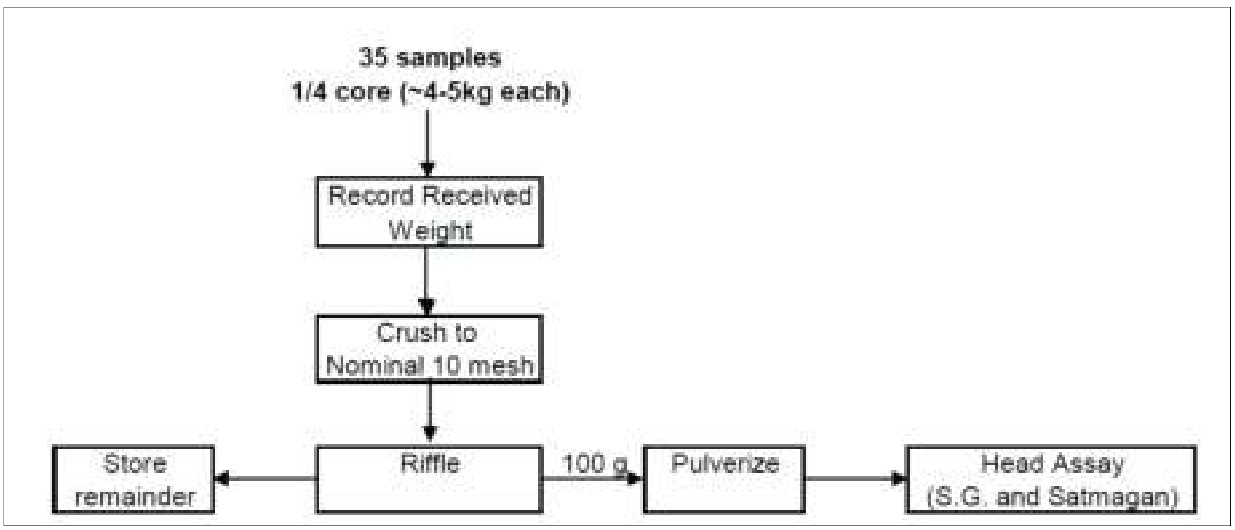


Figure 12-7: Sample preparation diagram – Satmagan – SGS Lakefield

Pulp duplicates were sent by BlackRock to SGS Lakefield as part of its QA/QC program for Satmagan results. Figure 12-8 shows the correlation of original and duplicated Satmagan values for 34 pulp duplicates. With an R^2 of 0.996 (and a regression line on the first diagonal), the duplication is nearly perfect.

The authors compared the values of the Satmagan on the 35 independent samples sent to SGS Lakefield (from the quarter split) and the same samples sent by BlackRock to SGS Lakefield (from the original half split) as part of their analyses program. The R^2 of 0.937 (and a regression line close to the first diagonal) indicates a good reproduction of the original Satmagan results (Figure 12-9).

Based on the data, it was concluded that the Satmagan results replicate extremely well in the same laboratory (SGS Lakefield). The lower R^2 obtained in Figure 12-9 can also be explained by the nature of duplicates involved (new pulp from quarter core instead of same pulp).

Mr. Cloutier and Mr. Bisaillon's conclusions, from these past site visits, were that the Satmagan results replicated extremely well in the same laboratory (COREM) and slightly less between two different laboratories (COREM and SGS). Differences are most likely due to the heterogeneity of the rock samples (1/4 vs. 1/2 core), or to the Satmagan instrument itself.

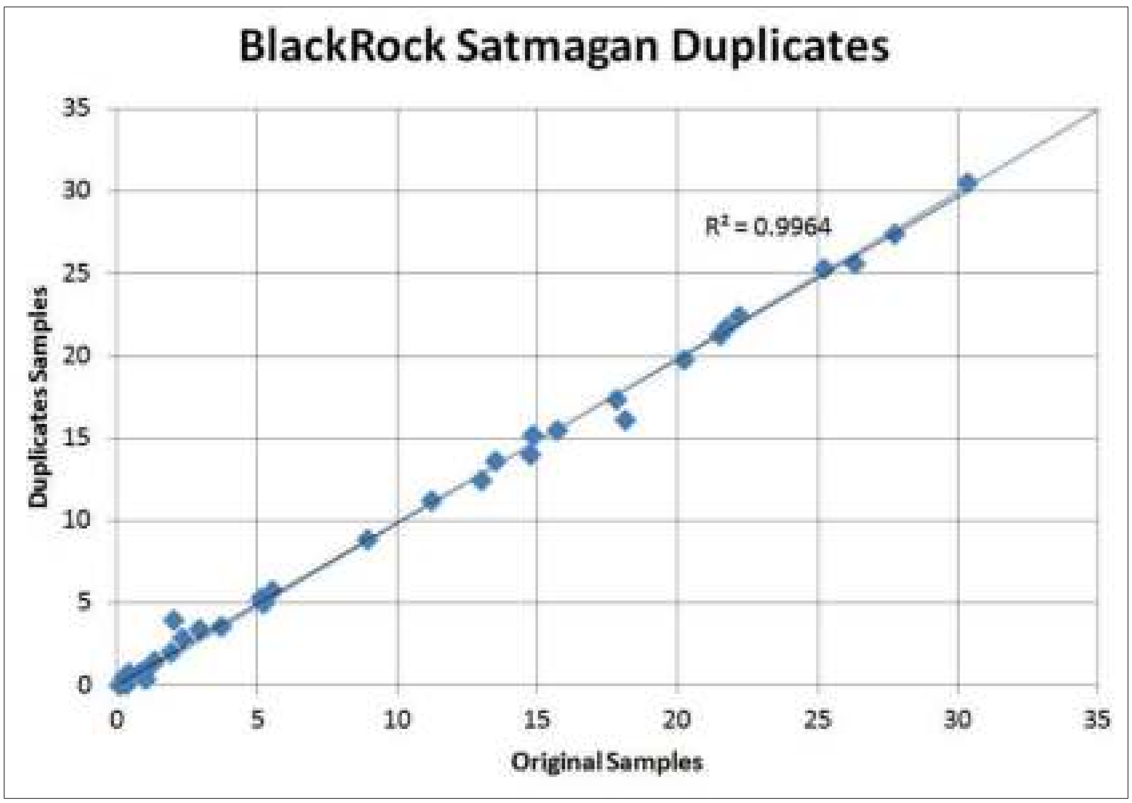


Figure 12-8: BlackRock Satmagan duplicates at SGS Lakefield

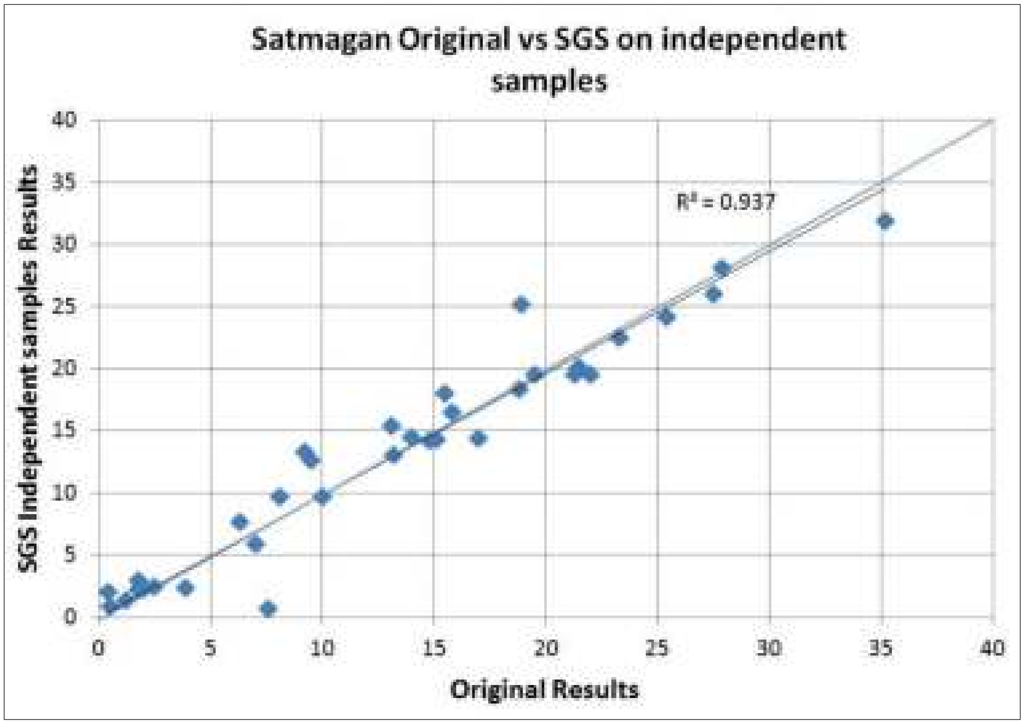
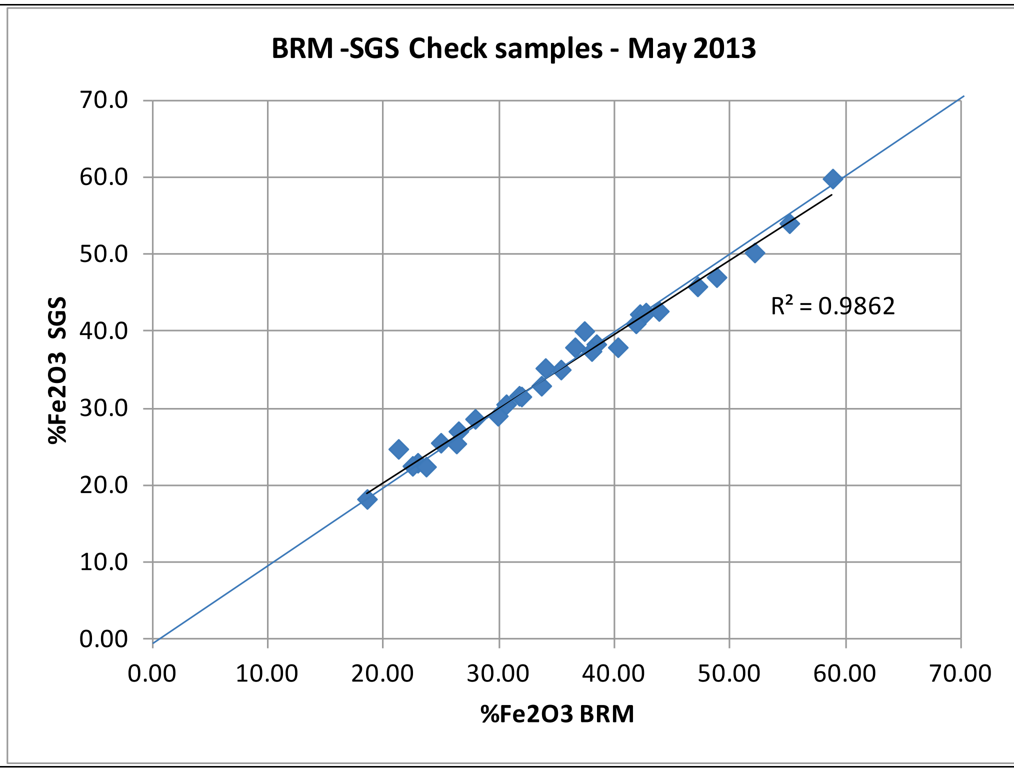


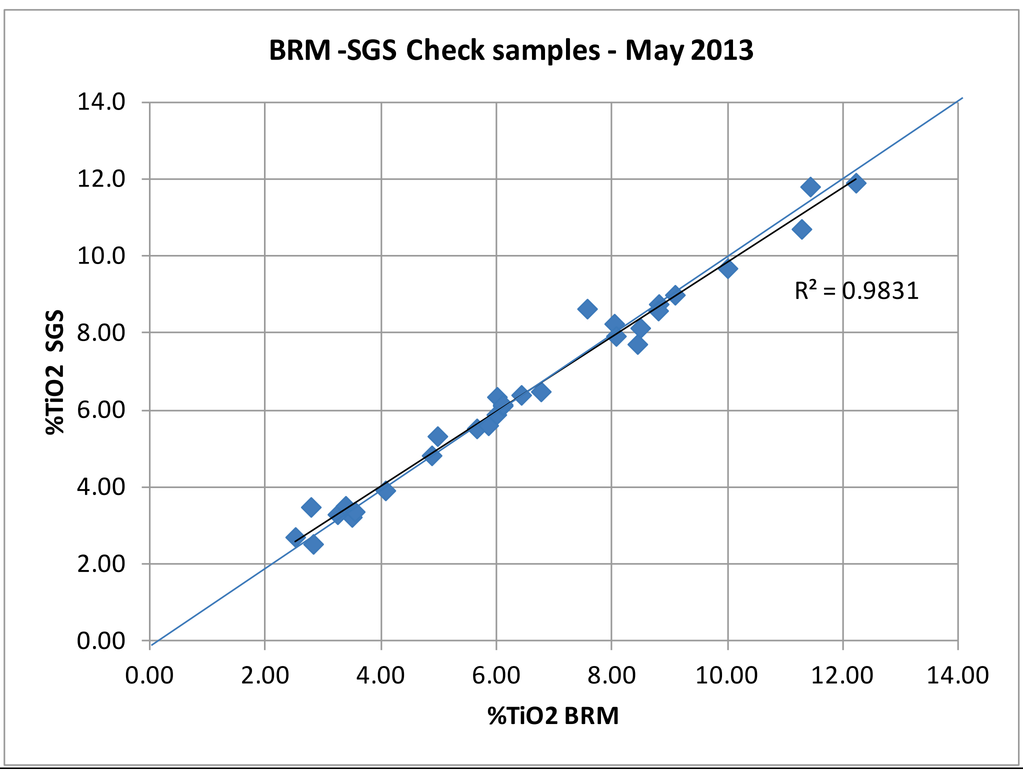
Figure 12-9: SGS Satmagan duplicates at SGS Lakefield

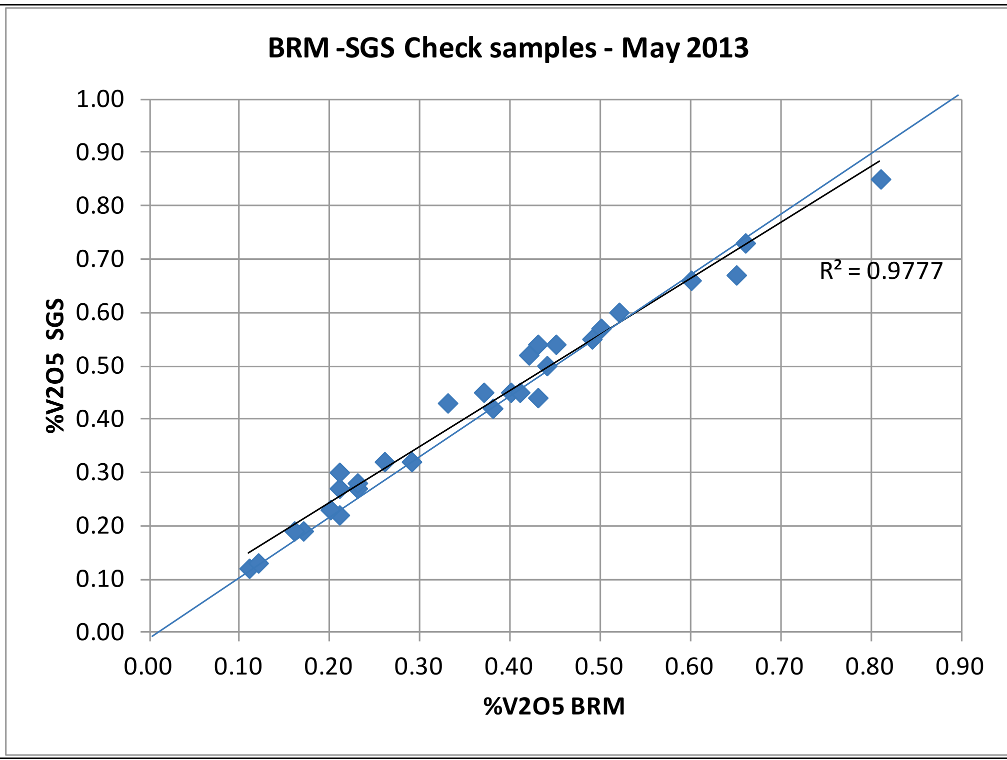
12.2.4. Independent 2013 SGS Samples

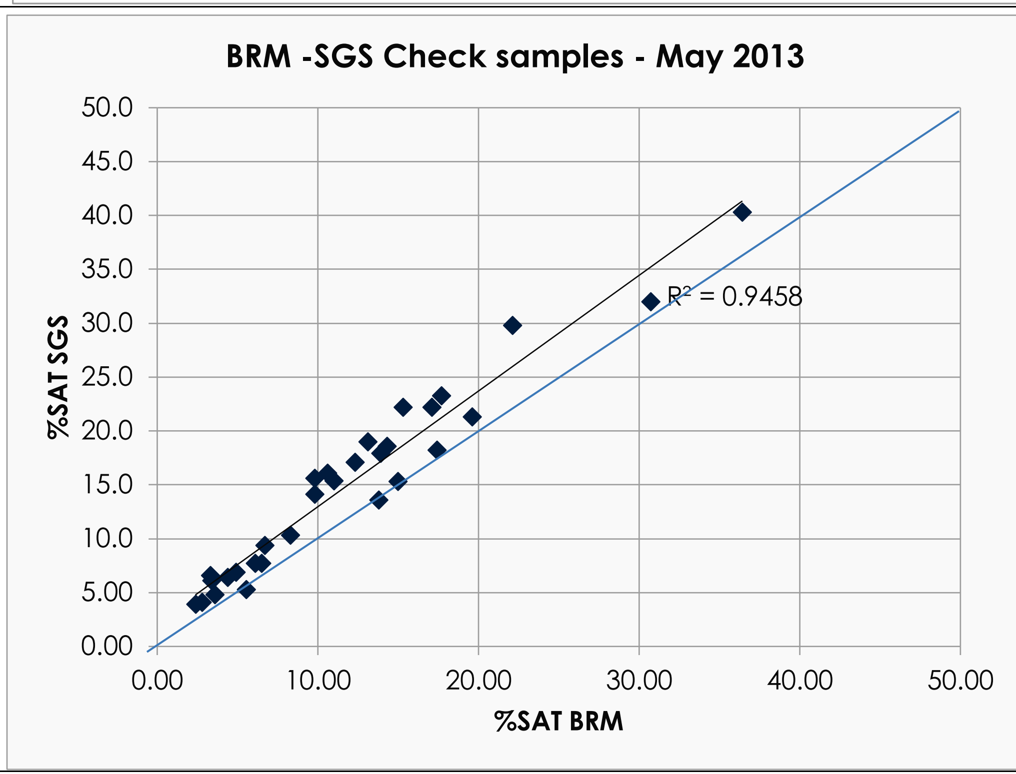
During Ms. Karina Sarabia's (Géo. Stag.) May 2013 site visit, a series of 30 independent control samples were selected on certain holes. These samples were analyzed and compared to original assay results. The sample pairs were plotted on scattergrams and then compared. The majority of the elements (including Fe_2O_3 and TiO_2) correlate well, as shown in Figure 12-10 below. However, the V_2O_5 and Satmagan values from SGS are 12% and 20% higher, respectively. SGS Geostat recommends that this difference be investigated.

The grouping of V_2O_5 values show a good relative correlation and the 12% difference may originate from a calibration problem. In the case of the Satmagan values, the samples were rerun at the SGS Québec City lab. The results between the two SGS labs are very consistent and show an extremely good correlation, with and R^2 of 0.999. Since the Satmagan values from BlackRock are on the conservative side, SGS Geostat found it acceptable to use the BlackRock values until the difference is explained.


BRM -SGS Check samples - May 2013
70.0
60.0
50.0
40.0
30.0
20.0
10.0
0.00
0.00
10.00
20.00
30.00
40.00
50.00
60.00
70.00
%Fe2O3 SGS
%Fe2O3 BRM
R² = 0.9862


BRM -SGS Check samples - May 2013
14.0
12.0
10.0
8.00
6.00
4.00
2.00
0.00
0.00
2.00
4.00
6.00
8.00
10.00
12.00
14.00
%TiO2 SGS
%TiO2 BRM
R² = 0.9831


BRM -SGS Check samples - May 2013
%V2O5 SGS
%V2O5 BRM
R² = 0.9777
1.00
0.90
0.80
0.70
0.60
0.50
0.40
0.30
0.20
0.10
0.00
0.00 0.10 0.20 0.30 0.40 0.50 0.60 0.70 0.80 0.90


BRM -SGS Check samples - May 2013
%SAT SGS
%SAT BRM
R² = 0.9458
50.0
45.0
40.0
35.0
30.0
25.0
20.0
15.0
10.0
5.00
0.00
0.00 10.00 20.00 30.00 40.00 50.00

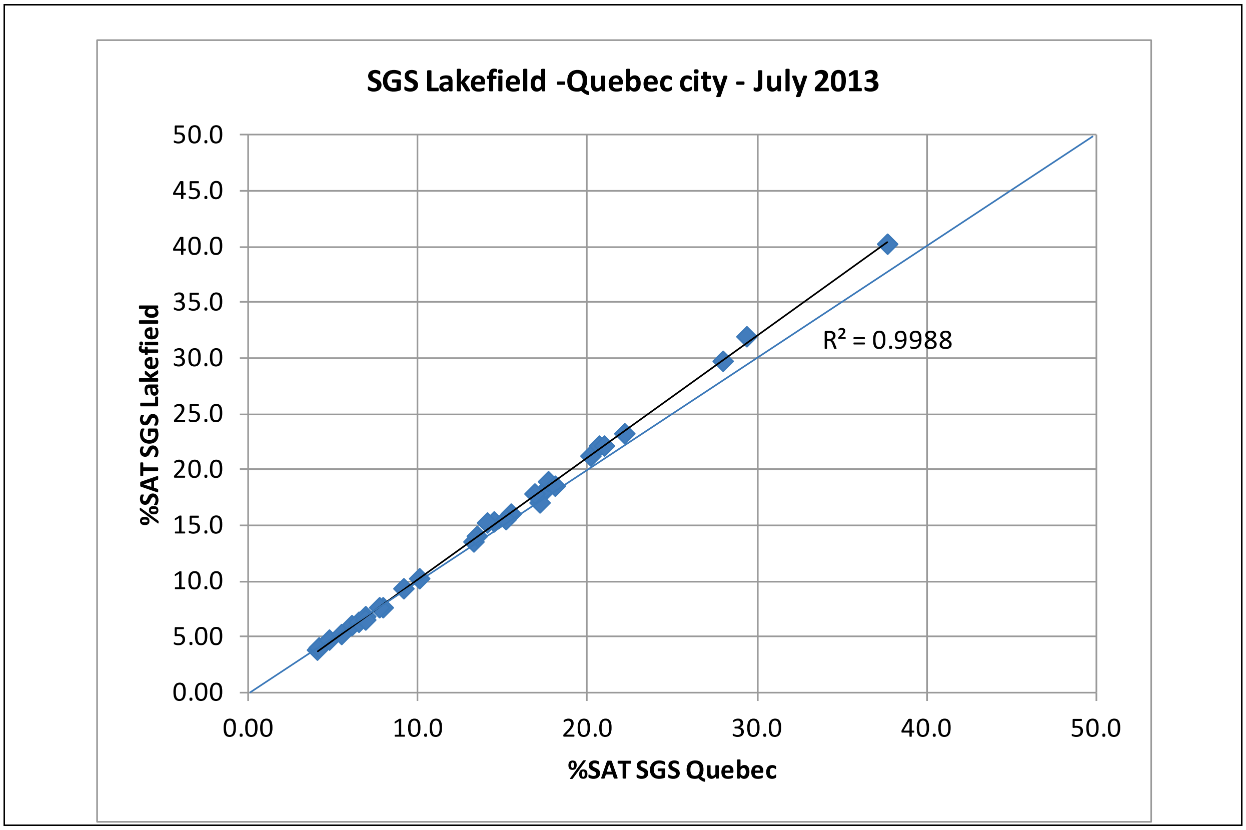


Figure 12-10: Check sample results

12.3. Comments on QA/QC

In terms of QA/QC, SGS Geostat is satisfied with the in-house QA/QC program set up by BlackRock. The details of this program have been clearly explained in the previous section of this report.

As explained earlier in this section, the difference in V_2O_5 and Satmagan values from SGS and BRM will need to be investigated in the next round of testing, resampling.

It is the opinion of the QP that the magnetic iron mineralization assay values are reliable enough to be used in the process of Mineral Resource estimation.

13. Mineral Processing and Metallurgical Testing

13.1 Mine and Beneficiation Plant

Detailed testwork campaigns were completed to define the beneficiation processes for obtaining a magnetite concentrate. The testwork program was developed to obtain the proper flowsheet and process design criteria. The main components of the test program were physical characterization, flowsheet development, process optimization and pilot scale validation of the processes. Bench-scale and pilot plant testwork have been conducted on both the Southwest and Armitage deposits. Although the testwork performed on the Armitage deposit is discussed in this chapter, the mine plan and process plant design were performed taking into consideration the Southwest Pit alone. The results and conclusions of the Armitage testwork are for reference purposes only.

13.1.1 Sample Selection

As there were several campaigns for both bench-scale testwork and pilot plant trials, it stands to reason that the samples used for each are not identical in terms of location in the pit. The material used for the preliminary bench-scale testwork came from surface samples in regions within the proposed pit design. Each pilot plant sample selected was chosen to be representative of the run of mine expected from the designed pit. These samples were from drill cores and were selected in collaboration with BlackRock geologists. Drill core sections containing over 7% Satmagan, with appropriate mine dilution, were chosen and used in testwork campaigns.

13.1.2 Physical Characterization

The physical characterization of the Southwest and Armitage Pits (such as specific gravity, ore hardness, abrasion, etc.) yields information required for the sizing of key equipment in the beneficiation plants, as well as information that is critical for determining wear life of certain equipment parts.

13.1.2.1 Southwest Pit Physical Characterization

Specific Gravity

The specific density measurements carried out using a pycnometer resulted in density values varying between 2.8 and 4.4 t/m^3. The average ore zone density was 3.5 t/m^3 and was used for metallurgical and equipment design purposes.

Ore Work Indices (Hardness)

The Bond Ball Mill Work Index (BWi) and Rod Mill Work Index (RWi) were determined using the standard Bond ball and rod mill procedures with a reference screen size of 150 µm. The results yielded a work index of 14.95 and 12.9 kWh/t for the rod mill and ball mill tests, respectively.

Table 13-1: Results for rod and ball mill work index tests

Parameter	# of Samples	Average (kWh/t)	80th Percentile (kWh/t)
Rod Mill Work Index (RWi)	9	14.95	15.8
Ball Mill Work Index (BWi)	11	12.9	13.6

From the results in Table 13-1, it can be ascertained that the material is moderately hard. The ratio of RWi/BWi yields 1.15. This result falls into a transition area but implies that there may be a likelihood for the creation of pebbles during primary grinding.

Similar testwork was performed at two external laboratories in order to determine the Crusher Work Index; information used in the sizing and selection of primary crusher. This information can be found in Table 13-2.

Table 13-2: Crusher work indices for the Southwest Pit

Parameter	Work Index (kWh/t)
Number of samples	9
Average	10.65
80th Percentile	11.4

The results show a design crusher work index of 11.4 kWh/t when using the 80th percentile of hardness. The work index indicates a medium to hard ore with maximum work indices up to 16.8 kWh/t.

Ore Competency (Abrasion Index and Resistance to Impact)

A preliminary Drop Weight Test (DWT) was performed on a trench sample from the Southwest Pit, which suggested an Axb factor of 47, indicating a moderate to soft hardness ore. However, a full-competency campaign revealed the ore to be substantially harder, with a 75th percentile hardness (design) of 31.81 Axb, as seen in Table 13-3. There was no significant indication of ore hardness variability within the pit.

Table 13-3: SMC and DWT results for Southwest Zone Pit

Sample Name	Axb	t_a
SMC Average	38.31	0.27
SMC Design - (75th Percentile)	31.81	0.23
DWT Average - (8 samples)	42.09	0.51
DWT Design - (75th Percentile) - (8 samples)	29.87	0.34
Overall Average	38.98	0.32
Overall Design - (75th Percentile)	31.81	0.24

13.1.2.2 Armitage Pit

Density Measurements

The densities of various BlackRock drill core samples were prepared at TJCM in Chibougamau Québec and measured at ALS, SGS Lakefield and COREM in Québec City. The average density measurement of 736 samples was 3.34 g/cm^3.

Ore Work Indices (hardness)

Armitage sampling was divided into sectors in order to analyze the differences along the strike and along the depth of the pit. Twelve sub-samples were obtained that had material above the cut-off grade of 7% Satmagan and were located within the proposed pit.

Table 13-4: Hardness information for the Armitage Zone Pit

Parameter	BWi (kWh/t)	RWi (kWh/t)	Ai (g)	CWi (kWh/t)
Number of samples	12	3	3	12
Average	12.17	14.43	0.25	10.65
Design (80th Percentile)	12.82	15.24	0.3	11.86

The hardness information in Table 13-4 indicates that the Armitage Deposit is similar to Southwest in terms of the BWi, RWi, CWi and Abrasion Index (Ai) (Table 13-1 and Table 13-2).

Ore Competency (Abrasion Index and Resistance to Impact)

Table 13-5 below shows the results from the measurements of ore competency (Axb value) performed at SGS Lakefield.

Table 13-5: Armitage Zone SMC and DWT results

Sample Name	Axb	t_a
SMC Average	34.3	0.25
SMC Design (75th Percentile)	30.0	0.22
DWT Average	30.1	0.35
DWT Design (75th Percentile)	28.1	0.325
Overall Average	33.9	0.26
Overall Design (75th Percentile)	29.9	0.22

The Axb design value (75th percentile) for the combination of DWT and SMC tests is 29.9, which is slightly lower than the Axb value obtained for the Southwest deposit (see Table 13-3). These results suggest that the SAG mill may process less material from Armitage than from Southwest.

13.1.3 Magnetite Beneficiation Process

13.1.3.1 Weight Recovery Satmagan Correlation Equations

Four hundred and four drill core samples from the Southwest Pit were submitted for Davis Tube analysis (DTA). The samples were grinded to 100%, passing 75 microns, where 20 g subsamples were processed by DTA separation.

Using the results of the Satmagan, DTA and WRA of the samples, correlation equations were developed between the Satmagan reading and DTA results. The results obtained from the DTA were normalized in order to calculate the weight recovery based on a 62% Fe concentrate.

Table 13-6: Weight recovery vs. Satmagan correlation equation

Magnetite Circuit Weight Recovery Equation	R2 Value
$\text{Weight Recovery\%} = 1.366 \times \text{Head\%Satmagan}$	0.87

The correlation equation shown in Table 13-6 was derived by plotting the Satmagan head grades versus the normalized weight recoveries from the DTA. The results from the bench-scale and pilot level testing fit well with the equation.

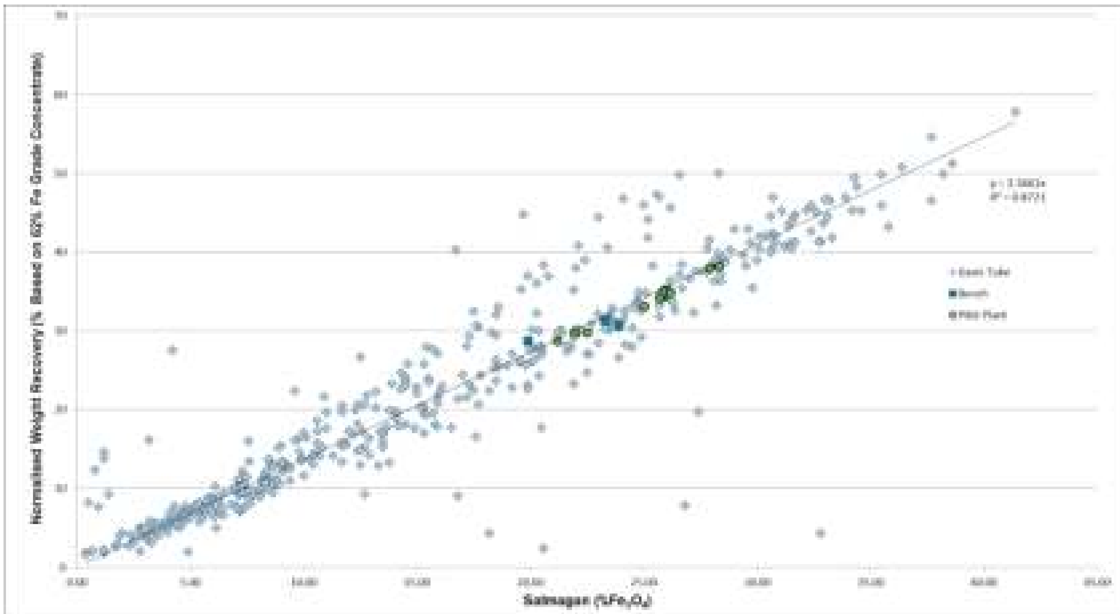


Figure 13-1: Satmagan versus normalized weight recovery for all Southwest Davis Tube results

Figure 13-1 depicts the correlation equation obtained for the weight recovery of a magnetite concentrate based upon the feed grade of Satmagan at a concentrate grade of 62% Fe (based upon Davis tube results). Also shown on the chart are the results from bench scale and pilot plant testwork performed on the Southwest Pit. The results from the lab and pilot tests agree with the correlation of Satmagan and concentrate weight recovery.

Although there is a strong correlation for the average recovery of concentrate from the Southwest Pit; the Southwest Pit is comprised of four different lithologies and could be further correlated by type of lithology.

Two hundred 294 samples classified as ore from the Southwest drill cores were used to plot the recovery equations based upon lithology. From this analysis, four equations were determined to be used for mining models and financial analyses.

Table 13-7: Equations for mining models and financial analyses

Zone	Magnetite Circuit Weight Recovery Equation	R2 Value
BCS	$\text{Weight Recovery\%} = 1.3353 \times \text{Head\%Satmagan}$	0.87
MCS	$\text{Weight Recovery\%} = 1.4102 \times \text{Head\%Satmagan}$	0.79
TITAN	$\text{Weight Recovery\%} = 1.4692 \times \text{Head\%Satmagan}$	0.85
ULS	$\text{Weight Recovery\%} = 1.3527 \times \text{Head\%Satmagan}$	0.83

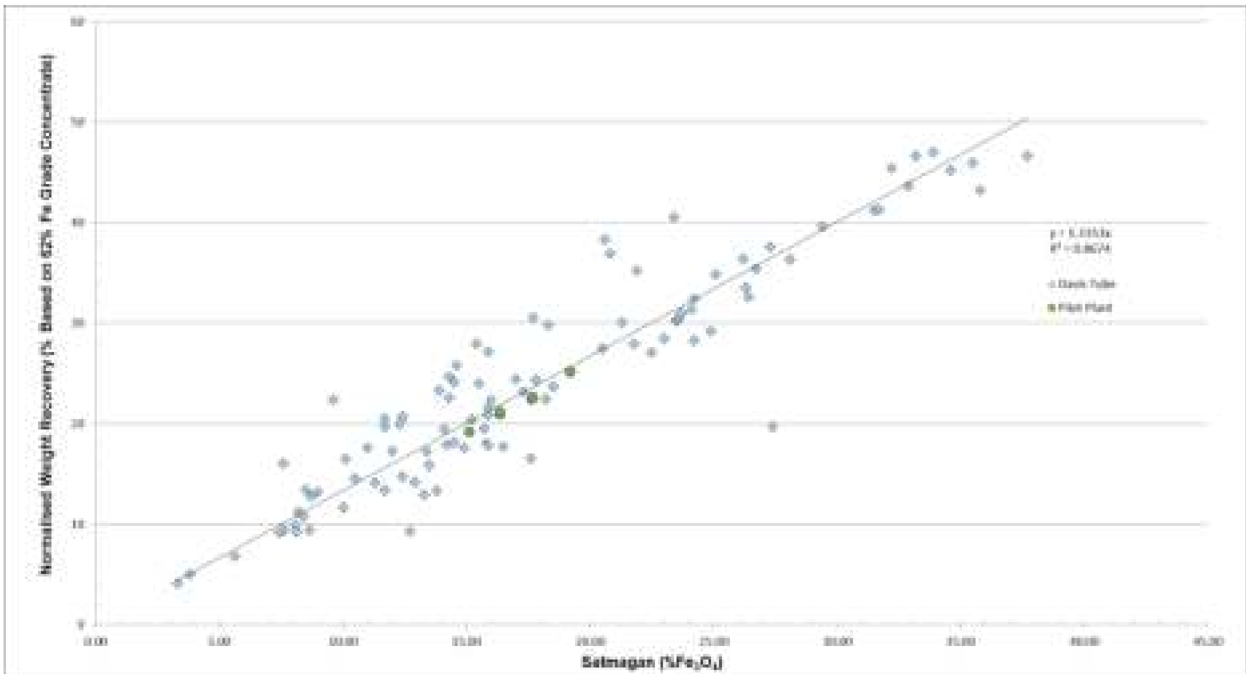


Figure 13-2: Satmagan versus normalized weight recovery for BCS Zone Davis Tube ore samples

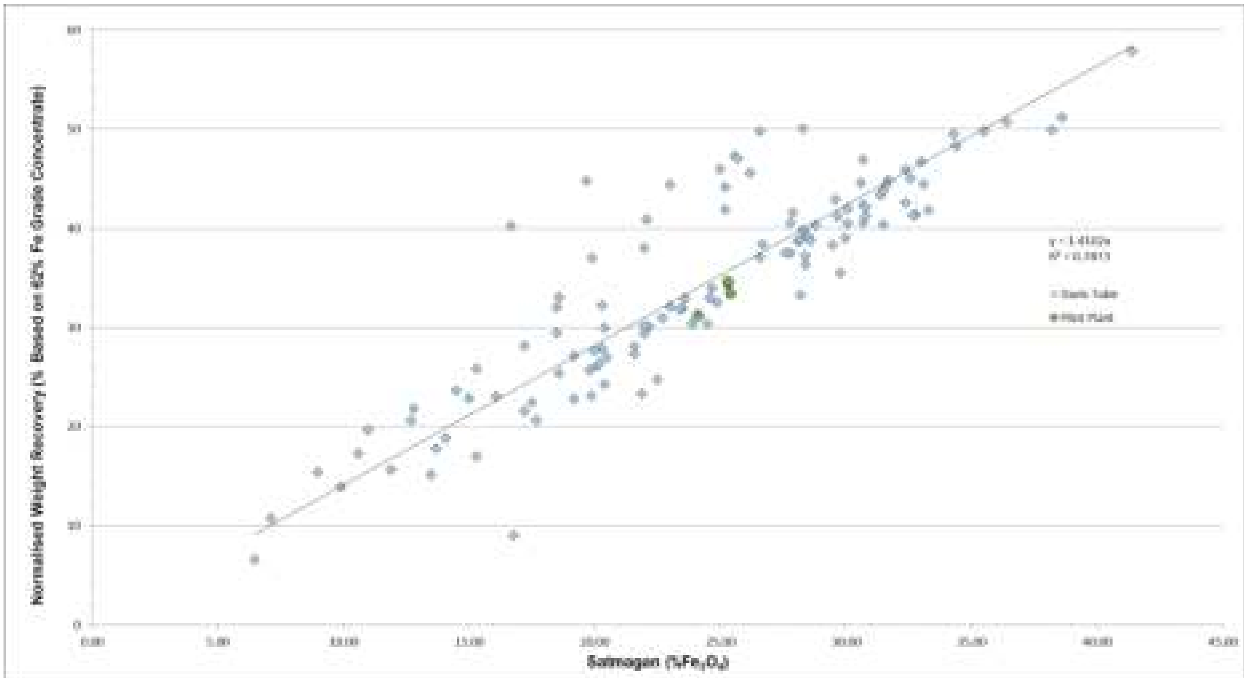


Figure 13-3: Satmagan versus normalized weight recovery for MCS Zone Davis Tube ore samples

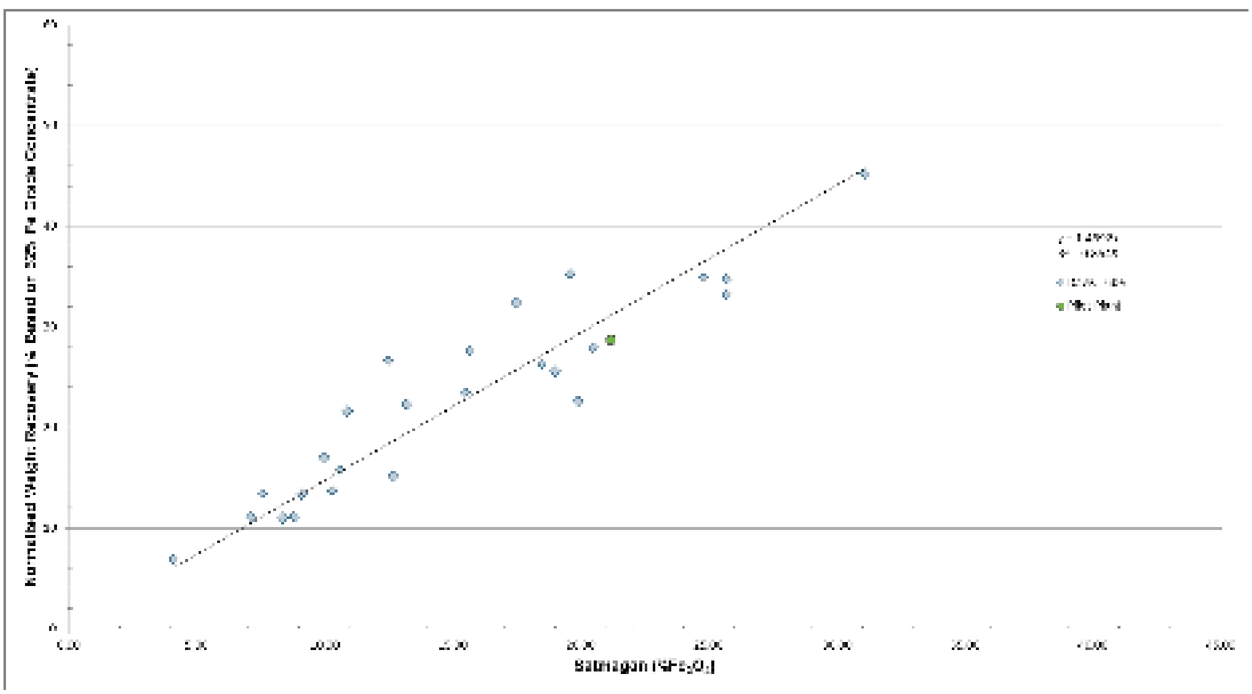


Figure 13-4: Satmagan versus normalized weight recovery for TITAN Zone Davis Tube ore samples

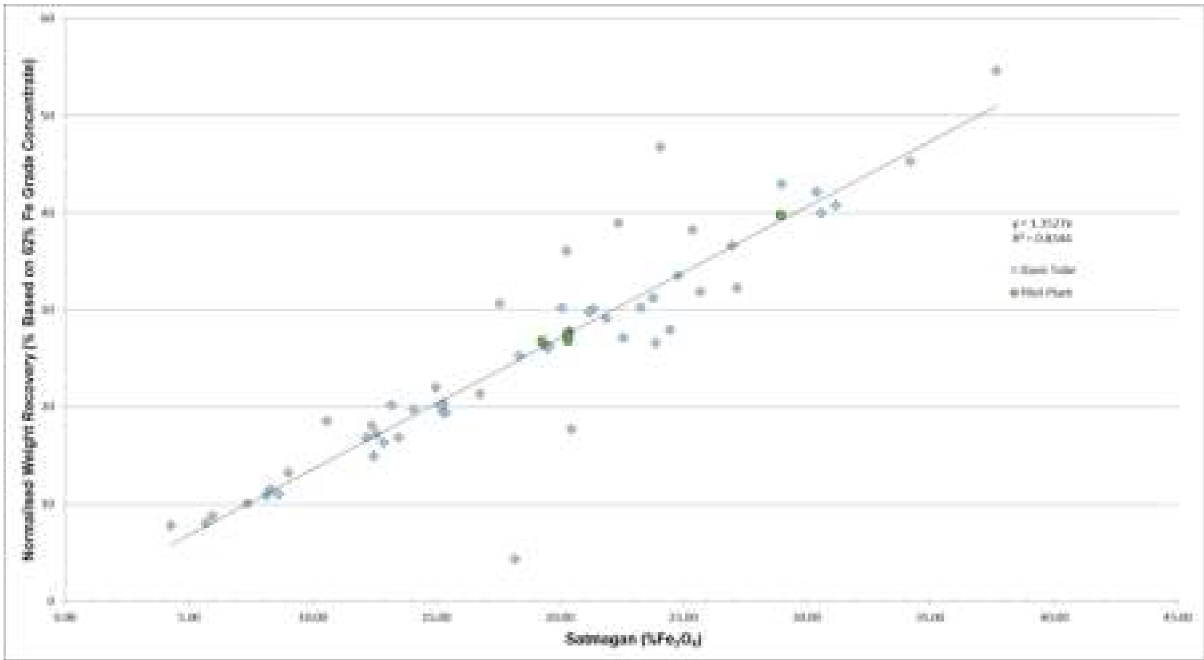


Figure 13-5: Satmagan versus normalized weight recovery for ULS Zone Davis Tube ore samples

The data graphed in Figure 13-2, Figure 13-3, Figure 13-4 and Figure 13-5 compare the correlation, by lithology, between Satmagan and normalized concentrate weight recovery for both Davis tube tests and pilot tests. As was the case for the overall weight recovery equation, there are strong correlations in the analysis of Satmagan versus weight recovery on a lithological basis which is confirmed by the pilot plant tests.

13.1.3.2 Vanadium Correlation Equations

The behavior of vanadium (V_2O_5) in the magnetite recovery process was investigated in order to predict the vanadium grade in the final magnetite concentrate. As vanadium is associated with iron species, the investigation focused on determining the upgrading factor of vanadium in relation to iron in the final concentrate. Using data from 294 Davis tube test results, the following equations were determined for each lithological zone:

$$BCS\ Zone: \ \text{Concentrate}\%V_2O_5 = \left(1.3765\left(\frac{62\%Fe}{Head\%Fe}\right) - 0.3149\right) \times \text{Head}\%V_2O_5$$

$$MCS\ Zone: \ \text{Concentrate}\%V_2O_5 = \left(1.7863\left(\frac{62\%Fe}{Head\%Fe}\right) - 0.9135\right) \times \text{Head}\%V_2O_5$$

$$TITAN\ Zone: \ \text{Concentrate}\%V_2O_5 = \left(2.3311\left(\frac{62\%Fe}{Head\%Fe}\right) - 1.7015\right) \times \text{Head}\%V_2O_5$$

$$ULS\ Zone: \ \text{Concentrate}\%V_2O_5 = \left(1.8004\left(\frac{62\%Fe}{Head\%Fe}\right) - 0.874\right) \times \text{Head}\%V_2O_5$$

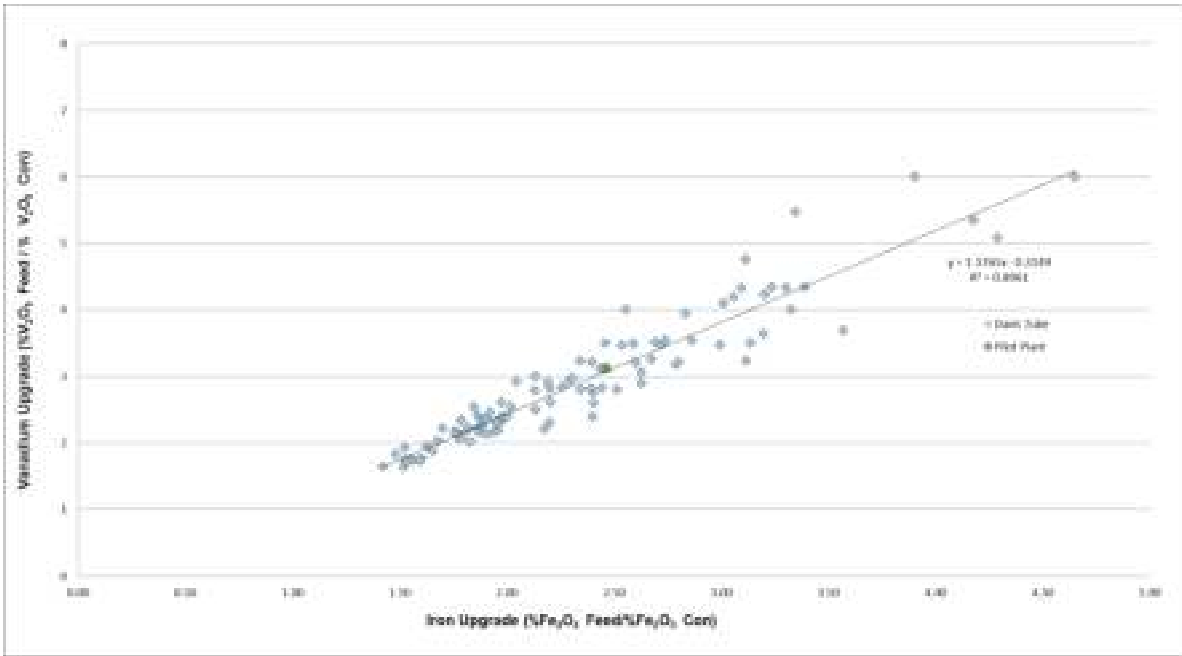


Figure 13-6: Vanadium upgrade versus iron upgrade correlation for BCS Zone

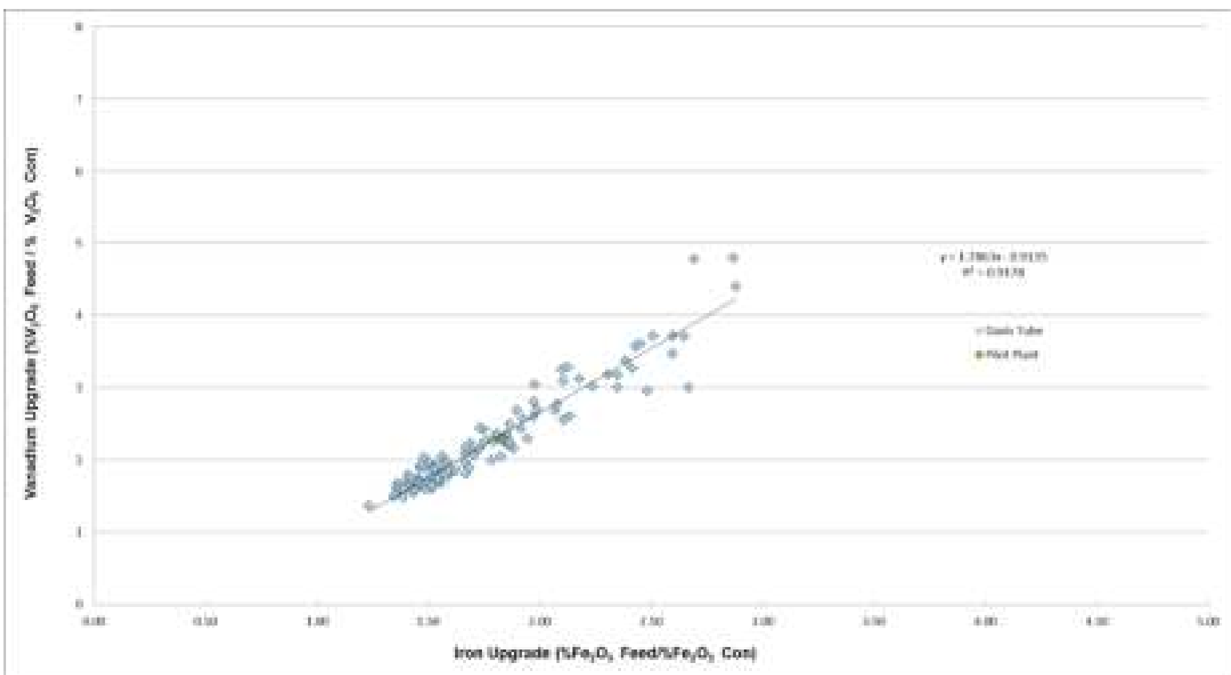


Figure 13-7: Vanadium upgrade versus iron upgrade correlation for MCS Zone

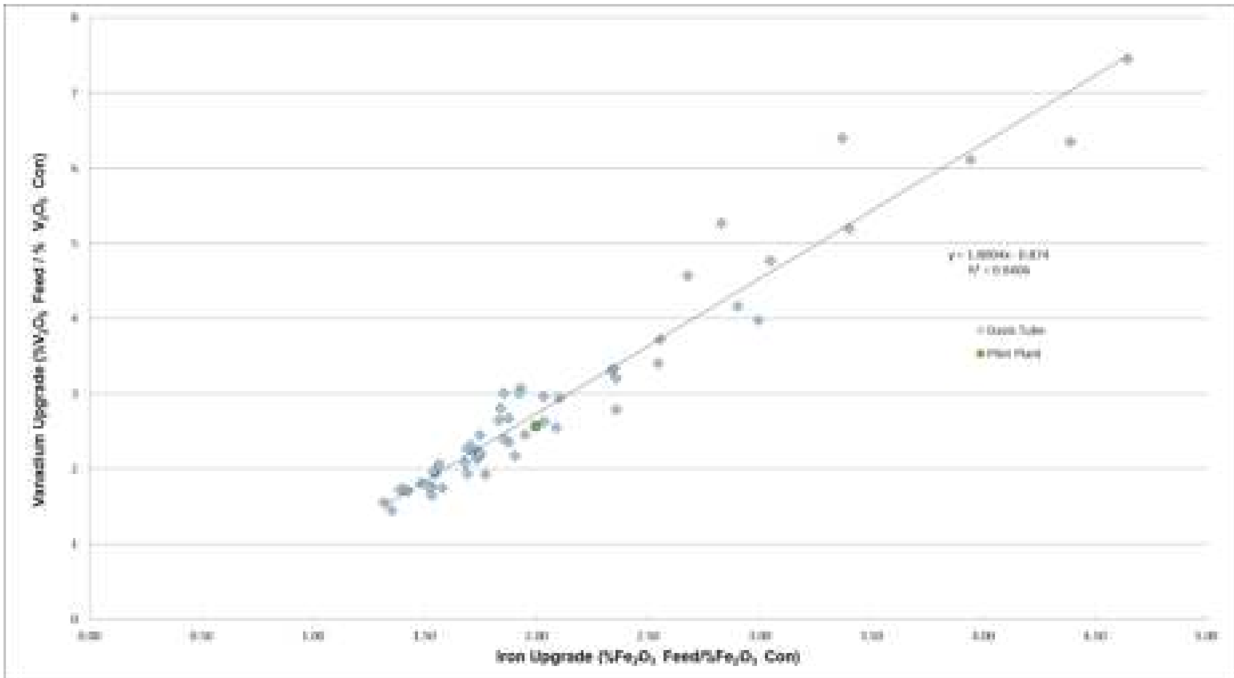


Figure 13-8: Vanadium upgrade versus iron upgrade correlation for ULS Zone

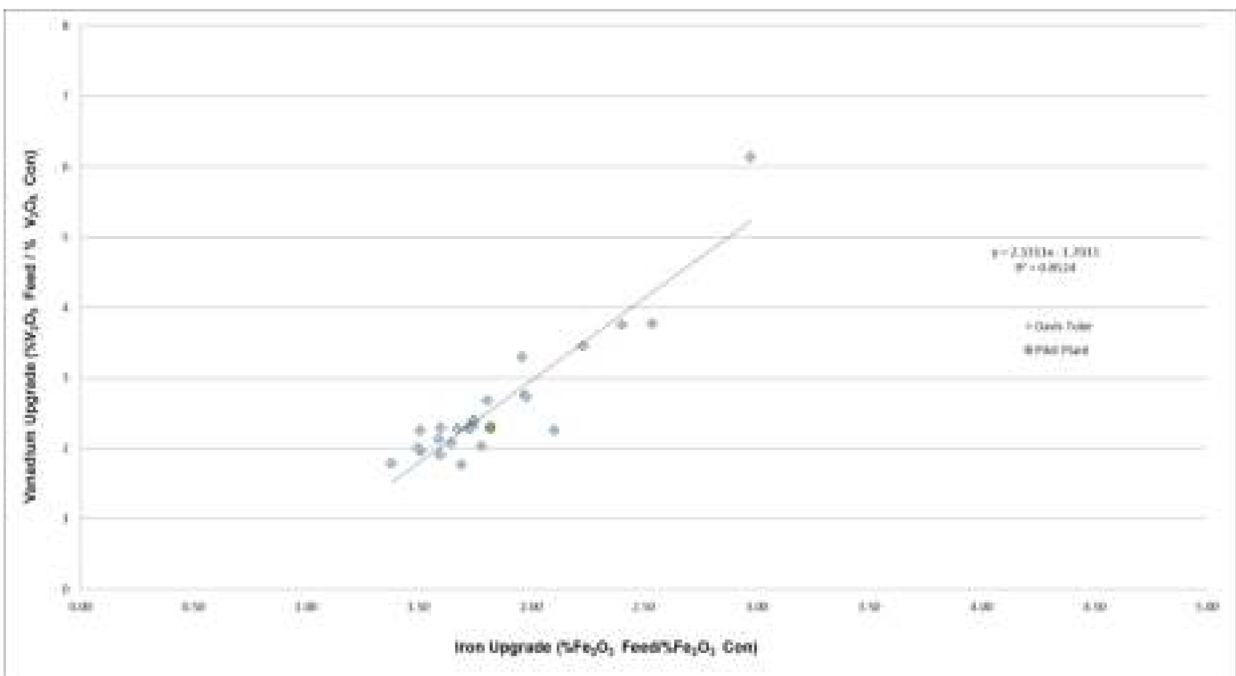


Figure 13-9: Vanadium upgrade versus iron upgrade correlation for TITAN Zone

The amount by which the grade of an element increases from the feed to the concentrate is considered to be its upgrading factor. In the case of the four lithologies we see a strong correlation between the upgrading factors of iron and vanadium as can be seen in Figure 13-6, Figure 13-7, Figure 13-8 and Figure 13-9. As was the case for weight recovery correlation equations, pilot plant testwork confirms the correlation between the iron and vanadium upgrading factors.

13.1.3.3 Development of a Magnetic Separation Flowsheet (Southwest)

Preliminary testwork was conducted at COREM in order to develop a magnetic separation process flowsheet. The tests consisted of low-intensity magnetic separation tests (LIMS), DTA, and mineralogical analyses of selected feed and product samples. Preliminary testing included the flotation of pyrrhotite in order to obtain a final sulphur grade of less than 0.5%. It is currently understood that this target concentrate grade is no longer required as the sulphur content will be managed in BlackRock's secondary transformation plant.

The magnetite concentrate product specifications are the following:

- Logical utilization of the site geotechnical characteristics.
- Best process flow of materials inside the plant;
- FeT grade = 62 - 65%;
- Combined SiO2 + Al2O3 grade < 3%;
- Lowest TiO2 grade possible;
- Maximum P grade = 0.06%;
- V2O5 grade > 1.1%;
- Impurity ratio $\frac{(MgO + CaO)}{(SiO_2 + Al_2O_3)} < 0.3$

Scoping Magnetic Separation Tests

Initial magnetic separation tests were conducted to obtain scoping results before proceeding with the main testwork program. A cobbing test was conducted on a sub-sample that was ground to a P_{80} of 300 µm, followed by a cleaning stage performed at two different grind sizes. The cleaner LIMS tests were done using a DTA on samples with P_{80} values of 53 µm and 75 µm. The scoping test procedure is illustrated in Figure 13-10.

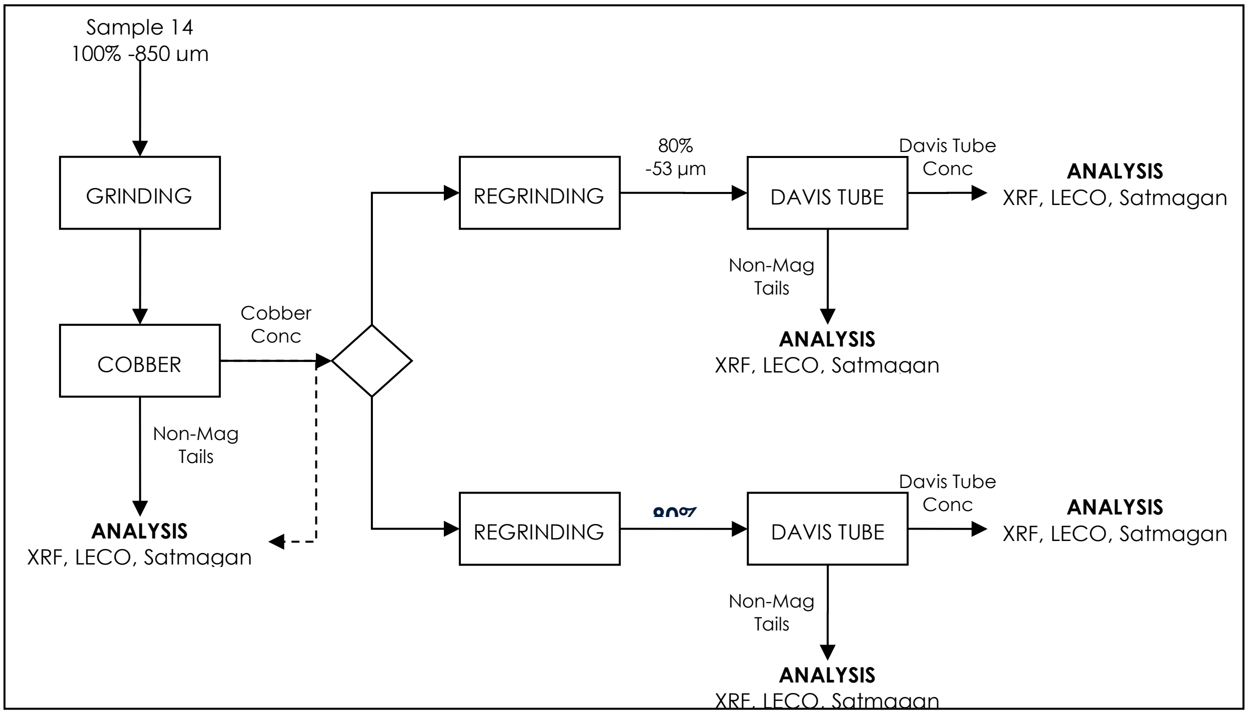


Figure 13-10: Scoping test procedure for magnetic separation on sample 14

Based on the results of the scoping testwork program, the number of required magnetic separation stages was determined in addition to the required liberation and grind sizes.

Bench-scale Flowsheet Verifications

Magnetic separation tests including cobbing, cleaning, and re-cleaning were performed on five composite samples from the Southwest deposit. The optimized feed sizes to the cobber and cleaner LIMS were previously determined in earlier testwork. Results of complete magnetic separation tests for each composite are presented in Table 13-8.

Table 13-8: Summary of Overall Magnetic Separation Test Results

Sample No.	Concentrate Product	Weight Recovery (%)	Grade in Magnetic Product (%)						Cumulative Distribution in Magnetic Product (%)	
			Mag. (1)	Mag Fe (2)	Fe_T	TiO_2	$SiO_2 + Al_2O_3$	V_2O_5	Mag.	Fe_T
6	Cobber	42.8	53.5	38.7	51.4	11.0	11.53	0.97	95.9	69.7
	Cleaner	31.0	78.2	56.6	62.6	8.6	2.27	1.28	101.5	61.4
	Re-cleaner	30.7	75.3	54.5	63.1	8.5	1.89	1.27	96.6	61.3
10	Cobber	43.5	49.6	35.9	50.5	13.0	11.21	0.89	93.2	66.1
	Cleaner.	29.1	78.5	56.8	62.8	8.2	2.54	1.25	98.5	54.8
	Re-cleaner	28.6	77.5	56.1	63.3	7.8	2.06	1.25	95.7	54.4
14	Cobber	39.8	49.4	35.7	51.4	15.8	8.97	0.85	98.9	68.7
	Cleaner	29.1	68.6	49.6	59.6	14.0	1.64	1.05	100.4	58.3
	Re-cleaner	28.7	66.6	48.2	59.8	13.8	1.35	1.05	96.4	57.8
18	Cobber	49.6	48.6	35.2	50.7	13.0	11.09	0.91	92.7	69.8
	Cleaner	35.6	74.0	53.5	60.8	10.6	2.58	1.2	101.2	59.9
	Re-cleaner	35.2	71.4	51.7	61.4	10.4	2.29	1.2	96.6	59.9
20	Cobber	49.5	48.2	34.9	50.0	12.2	12.70	0.89	92.3	68.8
	Cleaner	34.7	75.4	54.6	61.4	8.9	3.04	1.21	101.3	59.3
	Re-cleaner	34.3	73.0	52.8	62.1	8.7	2.61	1.22	96.8	59.1
Average	Cobber	45.0	49.9	36.1	50.8	13.0	11.10	0.90	94.6	68.6
	Cleaner	31.9	74.9	54.2	61.4	10.1	2.41	1.20	100.6	58.7
	Re-cleaner	31.5	72.8	52.7	61.9	9.8	2.04	1.20	96.4	58.5

Note: Some values of magnetite distribution in the magnetic product are > 100% due to the ± 2.5% accuracy of the Satmagan readings and non-reconciliation of the results with respect to cleaner concentrate recovery.

(1) Satmagan Measurement

(2) Mag Fe was calculated assuming Satmagan yields Fe_3O_4

The final Fe_T grades obtained in the concentrates produced after cobbing, cleaning, and re-cleaning, were performed and ranged from 59.8% to a maximum of 63.3%. The Fe_T grades failed to consistently reach the desired range of 62 - 65% level due to the relatively high levels of TiO_2 in the final concentrates that varied from 7.8% to 13.8%. As TiO_2 is present in a solid solution with magnetite, further reductions in titania levels would not be possible without sustaining simultaneous reductions in Fe_T grade and recovery. The vanadium concentration in the final product averaged 1.20% (reported as V_2O_5). The average magnetic iron recovery for the five bench-scale samples was 96%.

Each of the complete magnetic separation tests achieved all product specification targets regarding impurity levels; these included combined SiO_2 + Al_2O_3 grades of less than 3%, maximum P grade 0.06%, and ratios of (CaO + MgO)/(SiO_2 + Al_2O_3) of less than 0.3.

13.1.3.4 Confirmatory Pilot Plant Testwork for the Southwest Zone

Sample Preparation

Two pilot campaigns were run for the Southwest Pit. The first pilot plant (PP1) used, approximately 400 kg of bulk trench samples. The second pilot plant (PP2) used 20 tonnes of drill core samples with over 7% Satmagan. All material used for the pilot plants were representative of the pit.

Methodology

The pilot plant methodology can be seen in Figure 13-11 and Figure 13-12. The target grind size for the regrinding stage was set to 1 mm to reduce the load on the primary grinding stage of the process. The circuits were fully assayed for density, chemical information and particle sizes.

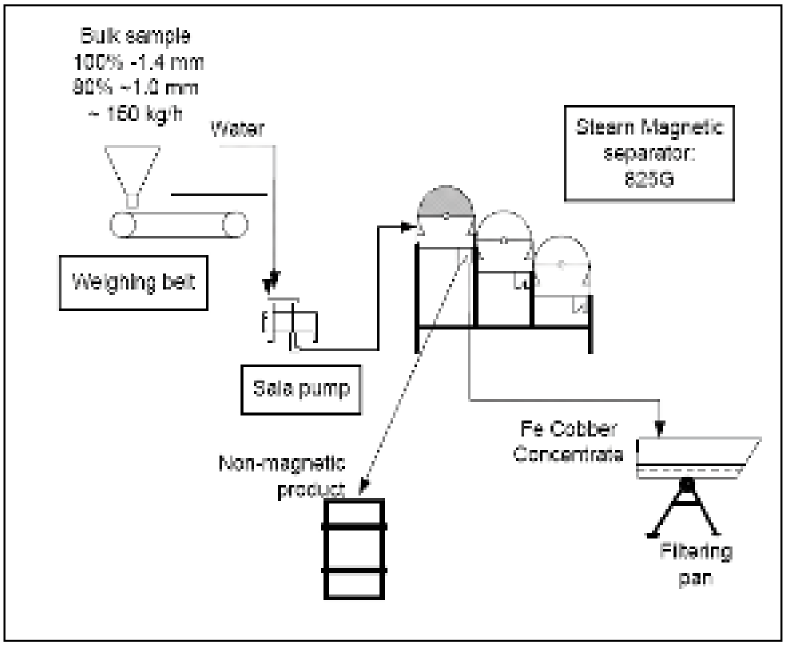


Figure 13-11: Pilot plant cobber stage flowsheet

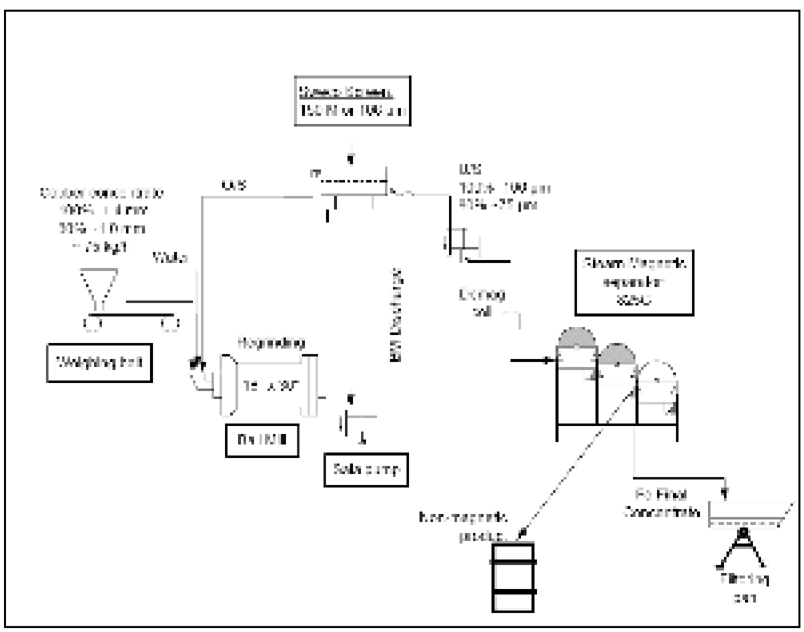


Figure 13-12: Pilot plant re-cleaner stage (w/ closed ball mill circuit) flowsheet

Results and Conclusions

The feed material used for the two pilot plants had Satmagan levels close to the bench-scale testwork average, as seen in Table 13-9. When compared to the bench-scale tests, the pilot plant results also indicated very similar silica, alumina and titania levels.

Table 13-9: Summary of chemical properties of the bulkhead sample and average of the five individual samples

Sample	Magnetite	Fe_T (%)	SiO_2 (%)	Al_2O_3 (%)	MgO (%)	TiO_2 (%)	S (%)
PP1	21.4	32.4	22.4	11.10	3.48	9.17	0.19
PP2	26.6	35.8	17.2	10.3	2.8	10.6	0.2
Avg. Bench	23.0	33.0	21.8	10.98	3.24	9.76	0.14

A summary of the results obtained from the pilot plant can be seen in Table 13-10 and Table 13-11. The results shown here are the balanced results obtained using the BILMAT™ software.

Table 13-10: Analysis results from the pilot plant vs. bench-scale

Product	Analysis (%)					
	Avg. Bench Head	Avg. Bench Final Conc.	PP1 Head	PP1 Final Conc.	PP2 Head	PP2 Final Conc.
Weight	100	31.5	100	30.1	100	36.1
Magnetite (1)	23.0	72.76	21.4	70.5	26.6	70.6
Mag Fe (2)	16.6	52.6	15.5	51.0	19.2	51.1
Fe_T	33.0	62.0	32.4	61.7	35.8	61.2
SiO_2	21.8	1.29	22.4	1.5	17.2	1.2
Al_2O_3	10.98	0.75	11.10	0.9	10.3	0.9
TiO_2	9.76	9.83	9.17	9.41	10.6	10.2
V_2O_5	0.51	1.12	0.48	1.18	0.61	1.2
S	0.14	0.01	0.19	0.17	0.2	0.08

Notes:

(1) Satmagan Measurement

(2) Mag Fe was calculated assuming Satmagan yields Fe_3O_4

Table 13-11: Distribution pilot plant testing vs. bench-scale

Product	Distribution (%)		
	Avg. Bench Final Conc.	PP1 Final Conc.	PP2 Final Conc.
Weight	31.5	30.1	36.1
Magnetite/Mag Fe	96.4	96.9	95.9
Fe_T	58.5	57.4	61.4

Although the two pilot plants were performed with a coarser feed than the bench-scale tests; the recovery of magnetic iron remained constant at the 96% range.

During both pilot plant trials, the final grade was close to the 62% Fe_T target. The final grades of 61.7 and 61.2 (slightly lower than 62%) are likely due to the concentrations of entrained ilmenite, as well as elemental titanium and vanadium found in solid solution within the magnetite crystal lattice. It is believed that it will be possible to obtain the target of 62% Fe_T in the plant, however, it greatly depends on the residual TiO_2 levels.

13.1.3.5 Confirmatory Testwork for the Armitage Pit

The Armitage Pit material was tested for metallurgical similarities to the Southwest Pit. Bench-scale and pilot scale tests were performed on Armitage material to evaluate the material's response to the flowsheet developed for the Southwest Pit. As previously seen in the correlation equations, Armitage DTA data exhibit a nearly identical response in terms of Satmagan and weight recovery.

Bench-scale Testwork

Forty-eight ~400 kg barrels of drill core from the Armitage deposit, representing run-of-mine ore from within the predefined mine pit, were separated into three equal portions for bench-scale testwork and the remaining material was used for pilot testwork.

Chemical Characterization

The Armitage head grades were compared to those of the Southwest bench-scale and pilot scale testwork campaigns. The chemical compositions were similar in terms of the major constituents. The three sub-composites formed from Armitage showed a wide range of head Satmagan that encompasses the average Satmagan for the Southwest bench-scale tests. Armitage also demonstrated a similar relationship between $Fe_T/TiO_2/V_2O_5$ levels, where higher Fe_T levels are followed by higher TiO_2 and V_2O_5 levels.

Results and Discussion

Table 13-12 and Table 13-13 show the concentrate grades and recoveries for the three Armitage bench-scale tests. Two of the three tests yielded Fe_T concentrate grades of 62% and above; however, sample 1 yielded a lower-grade concentrate than expected. Nonetheless, the test performed on the blend of the three samples had very similar characteristics to Armitage Bench 1 and achieved a good concentrate grade, with 62.3% Fe_T and a 94.2% recovery of magnetic iron.

Table 13-12: Concentrate grades for Southwest Bench-scale tests (average) and the three Armitage Composites

Product	Concentrate Analysis (%)				
	Avg. Southwest Bench Head	Armitage Bench 1	Armitage Bench 2	Armitage Bench 3	Armitage Blend Coarse
Weight	30.1	18.8	23.6	28.1	19.8
Magnetite (1)	70.5	69.51	78.08	82.09	79.19
Mag Fe (2)	51.0	50.3	56.5	59.4	57.3
Fe_T	61.7	59.2	62.0	63.2	62.3
SiO_2	1.5	2.14	1.25	0.84	1.32
Al_2O_3	0.9	1.3	0.8	0.65	0.87
TiO_2	9.41	11.5	11.2	10.1	10.6
V_2O_5	1.18	1.02	1.16	1.32	1.2
S	0.17	0.57	0.12	0.10	0.31

Notes :

(1) Satmagan Measurement

(2) Mag Fe was calculated assuming Satmagan yields Fe_3O_4

Table 13-13: Armitage bench-scale testwork distributions compared to Southwest

Product	Concentrate Distribution (%)				
	Avg. Southwest Bench Head	Armitage Bench 1	Armitage Bench 2	Armitage Bench 3	Armitage Blend Coarse
Weight	30.1	18.8	23.6	28.1	19.8
Magnetite/Mag Fe	96.4	92.5	95.3	95.4	94.2
Fe_T	58.5	43.4	50.9	55	48

The optimized flotation conditions used for the Southwest Pit were used to conduct two tests on the Armitage composite concentrate to reduce the S level to below 0.05%. The flotation tests demonstrated that the pyrrhotite flotation circuit developed for the Southwest Pit is also suitable for the Armitage.

13.1.3.6 Confirmatory Pilot Plant Testwork

13.1.3.6.1 Sample Selection and Preparation

The material used for the Armitage pilot plant was derived from the same material that was used for the bench-scale tests.

13.1.3.6.2 Methodology

The pilot plant performed on the Armitage material used the same procedure as the pilot plant test conducted for Southwest material at COREM with the addition of a flotation stage. The flowsheet diagram from the SGS report (SGS *12468-016* 2013) can be found in Figure 13-13. Note that the pilot plant takes into account a flotation of pyrrhotite in the magnetite concentration circuit. It was originally planned to lower the sulphur content of the concentrate via flotation. However, it has been decided to handle the excess sulphur in the smelting process.

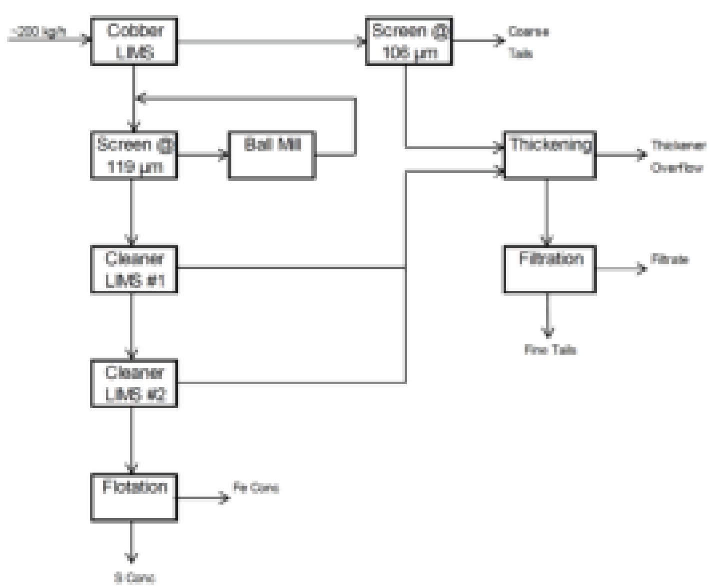


Figure 13-13: Magnetite beneficiation flowsheet used for Armitage Pilot Plant (SGS 12468-016 2013)

13.1.3.6.3 Results and Conclusions for Process Design Criteria

Based on the results of the Armitage pilot plant test, the following points were raised for further investigation:

- Whether the recovery of magnetic iron is achieved at the same level as the Southwest and Armitage bench-scale tests;
- Whether or not the product from both deposits meets the chemical specifications;
- Whether the partition coefficients behave as expected.

Table 13-14: Balanced concentrate assay for Armitage Pilot Plant (PP)

Product	Concentrate Analysis (%)
Magnetite (1)	81.4
Mag Fe (2)	58.9
FeT	62.2
SiO_2	1.25
TiO_2	11.3
S	0.05
Al_2O_3 (3)	0.88
V_2O_5 (3)	1.17

Notes :
(1) Satmagan Measurement
(2) Mag Fe was calculated assuming Satmagan yields Fe_3O_4
(3) Al_2O_3 and V_2O_5 grades are averages calculated over the course of the pilot campaign, whereas the others are balanced data using Bilmat

Table 13-14 shows that the targets for the concentrate quality are met: a grade of 62% Fe and less than 3% Al_2O_3 + SiO_2. The recorded weight, magnetic iron and FeT recoveries were 24.8%, 95.2% and 51.1%, respectively. This indicates that the pilot plant magnetite beneficiation efficiency for the Armitage deposit is approximately the same as for the Armitage bench-scale tests and Southwest bench-scale and pilot tests.

The single peculiarity of the testwork on Armitage is that the weight recovery was lower than expected, given the correlation equations obtained from the relationship between Satmagan and weight recovery for the DTA using all of the bench and pilot testwork performed on the Southwest and Armitage Pits. The bench-scale and pilot tests for the Armitage Pit have slightly lower weight recoveries when compared to the DTA tests, even though they meet the grade specifications and have good magnetic iron recoveries.

Upon further investigation, the material in the Armitage testwork performed at SGS may have produced more fines than in previous testwork done at COREM. This change in particle size may have resulted in the lower overall weight recovery seen in the Armitage data. This hypothesis was tested by submitting a sample of the raw ore from the Armitage Pit to COREM for preparation.

Following the same procedure used for the Southwest Pit, a repeat of the crushing stages was performed at COREM for the Armitage Pit and a coarser product was obtained. This produced a theoretically identical feed to that used by SGS for the Armitage pilot plant. As expected, an excessive creation of fines was produced during crushing. The end result did not affect the recovery of magnetic iron, nor did it affect the final FeT grade; however, there was a decrease in the final weight recovered and finer tailings were produced. This effect is a result of a change in partition coefficients due to the over-grinding.

Thus, the Armitage pilot plant serves as a confirmation of the process and recoveries, but adjustments will be required to optimize the mass partition coefficients used for plant design.

13.1.4 Equipment Specific and Additional Testwork

13.1.4.1 Filtration Testwork

Filtration testwork was carried out by multiple suppliers in order to determine the sizing and design criteria using vacuum filtration. The moisture achieved via filtration testwork with vacuum filtration was 8 wt% (without steam). Results are given in Table 13-15.

Table 13-15: Filtration results summary

Parameter	Disc Filter		Drum Filter	
Feed Solids (wt%)	55	60	55	60
Cake Thickness (mm)	13.5	20.5	9.5	10.5
Dry Cake Weight (kg/m^2)	32.4	49.2	23.1	25.2
Cycle Time (min)	2.2	3.3	1.1	1.2
Cake Moisture (wt%)	8	8	8	8
Filtration Rate (kg/m^2/h)	720	720	1040	1040

The recommended feed solids concentration is 60-wt% as this reaches target cake moisture at a lower moisture correlation factor. Although the drum filters can produce a higher filtration rate than disc filters at comparable cake moisture, the drum filter has to operate with thinner filter cakes and these may be more difficult to release from the media. The drum filter filtration rate can also reduce considerably if the filter feed solids concentration drops since the filter cake formation zone is much less than a disc filter. Through testwork, the minimum acceptable feed percent solids were found to be 55%.

13.1.4.2 Sedimentation testwork

Flocculant screening was performed prior to the sedimentation testwork. Each supplier performing testwork was then asked to use the chosen flocculant to ensure a standardization of the testwork. Sedimentation testwork was performed by three labs/suppliers to determine key parameters for sizing of the tailing's thickener. Sedimentation testwork results are summarized in Table 13-16.

Table 13-16: Sedimentation testwork results

		1	2	3
Feed P_{80}	Microns	66	80	53.6
Feed % solids	% wt.	8-9	12	8
Thickener Flux	m^2/tpd	0.0325	0.025	0.0706
Rise Rate	m^3/m^2/d	348	454	148.56
Underflow Solid Density	% wt.	65	66-71	50
Flocculant Dosage	g/t	20-30	15-20	18-24
Overflow Clarity	ppm	88	<100	<100

13.1.5 Concentrator Discussions and Recommendations

Extensive investigations have been conducted on the behavior of the magnetite beneficiation flowsheet. The effect of different feed grades, cobber F_{80} and grindability effects have been investigated, and in all cases, the circuit provides stable recoveries of magnetic Fe (94-97% recovery) and stable grades (61-63%). The magnetite circuit involves two stages of magnetic separation with a regrinding stage in between. As such, there is not much risk in terms of the flowsheet. The potential for a pebble crusher is a potential risk item in terms of mineral processing. To mitigate this risk, the SAG mill has been sized conservatively and space has been left for a pebble crusher in the event that the SAG does not reach the target throughput.

13.2 Metallurgical Plant Test Work

13.2.1 Sample Selection

Tenova received a 5-tonne sample of VTM concentrate sample was delivered in 3 lots, each mixed sufficiently to ensure uniformity of the coarse material. A subsample of each lot was taken and blended to create the master sample used by Corem for subsequent testwork.

13.2.2 Physical Characterization

Characterization tests were carried out to determine the physical properties, chemical composition and particle size distribution of the 5-tonne VTM concentrate sample received by Tenova. The characterization tests yield information required for the sizing of key equipment in the metallurgical plant, as well as information that is critical for determining wear life of certain equipment parts. Table 13-17 shows the particle size distribution of each lot.

Table 13-17: Particle size distribution of each lot

Fraction		% passing		
µm	mesh	Lot 1	Lot 2	Lot 3
425	35	99.0	99.2	99.2
300	48	98.6	98.9	98.9
212	65	97.6	98.0	98.3
150	100	95.6	96.0	96.3
106	150	90.3	90.8	90.9
75	200	78.2	78.7	78.5
53	270	62.5	62.8	62.8
45	325	54.8	55.2	54.9

XRF technology was utilized to perform the chemical analysis. In addition, iron content and Satmagan were measured by $K_2Cr_2O_7$ titration following the reaction of iron with various compounds, Table 13-18 shows the chemical composition of each lot. Table 13-19 shows the chemical composition of the blend of the three lots of VTM concentrate.

Table 13-18: Chemical composition of each lot

	Lot 1	Lot 2	Lot 3
% SiO_2	1.6	1.5	1.6
% Al_2O_3	1.17	1.15	1.19
% Fe_{tot}	61.9	61.9	61.8
% MgO	0.40	0.40	0.41
% CaO	0.21	0.22	0.23
% Na_2O	< 0.10	< 0.10	< 0.10
% K_2O	< 0.01	< 0.01	< 0.01
% TiO_2	9.05	8.92	8.97
% MnO	0.18	0.18	0.18
% P_2O_5	< 0.01	< 0.01	< 0.01
% Cr_2O_3	0.10	0.11	0.11
% V_2O_5	1.39	1.39	1.38
% ZrO2	< 0.02	< 0.02	< 0.02
% ZnO	0.03	0.04	0.03
% LOI	-2.66	-2.92	-2.88
% S_{tot}	0.03	0.03	0.03
% C_{tot}	<0.05	<0.05	0.05
% FeO	31.3	31.4	31.1
Satmagan (%)	69.4	70.9	70.7

Table 13-19: Chemical composition of the blend of the three VTM concentrate lots

Mineral	Mass (%)
FeTi oxides	96.3
Titanomagnetite	*49.2*
Magnetite	*41.8*
Ilmenite	5.3
Chlorite group	2.3
Pyroxenes/Amphiboles	0.6
Plagioclase	0.1
Epidote	0.4
Others*	0.3
Total	100.0

13.2.2.1 Density Measurements

Bulk density measurement was conducted according to ISO/FDIS 3852: Iron ores for blast furnaces and direct reduction feedstocks – determination of bulk density. The bulk density of two 150 kg samples, consisting of a near-equal blend of the three lots, was measured. Table 13-20 shows the bulk density of the two samples. Moisture was measured at 6.7% H_2O.

Table 13-20: Bulk density of the VTM concentrate blend

	Mass (kg)	Volume (l)	Bulk density (t/m^3)
Test 1	87.0	52	1.67
Test 2	86.5	52	1.66
AVG	86.8	52	1.67

13.2.2.2 Thermal Gravimetric Analysis

Thermal gravimetric analysis was performed on the blend of the three lots. The rate at which temperature changed during the test was 10 °C/min up to a maximum temperature of 1300 °C. Figure 13-14 depicts the differential thermal gravimetric results.

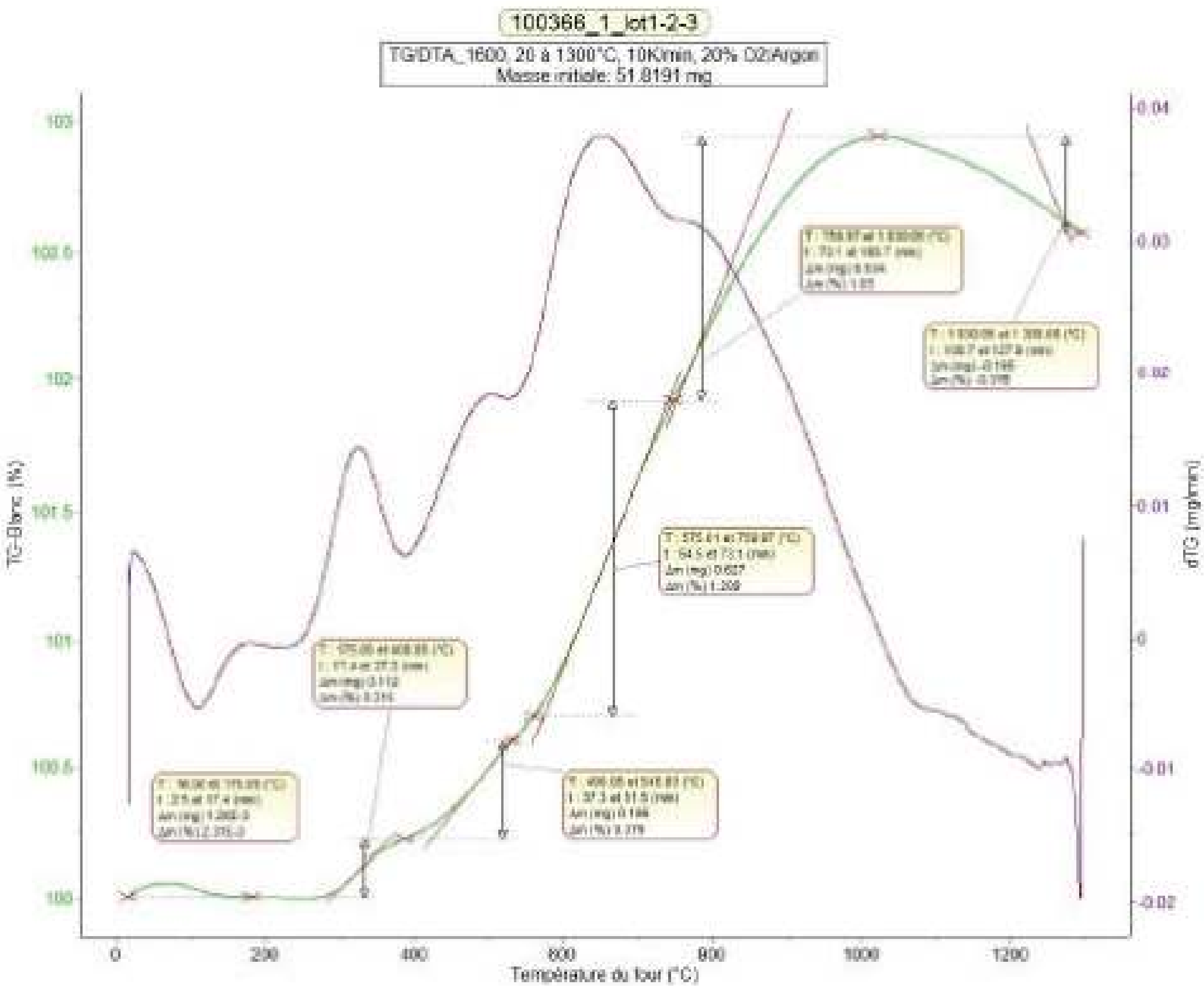


Figure 13-14: Differential thermal gravimetric results

The green line represents the change of the weight of the sample as a function of increasing temperature. A significant gain of weight (2.82%) is observed up to 1030 °C, primarily due to the oxidation of magnetite to hematite. Above 1030 °C, a drop in weight (0.38%) occurred. The loss of weight is attributed to oxidation state changes at high temperature.

13.2.3 Pelletizing Test Work

13.2.3.1 Objective

Corem Laboratory, in Quebec City, was selected to perform tests to determine the VTM concentrate suitability for pelletizing as well as to produce a series of samples for subsequent testing in HYL Direct Reduction commercial reactor located at the Ternium Plant in Monterrey, Mexico.

13.2.3.2 Pelletizing Test Flowsheet – Corem Quebec City

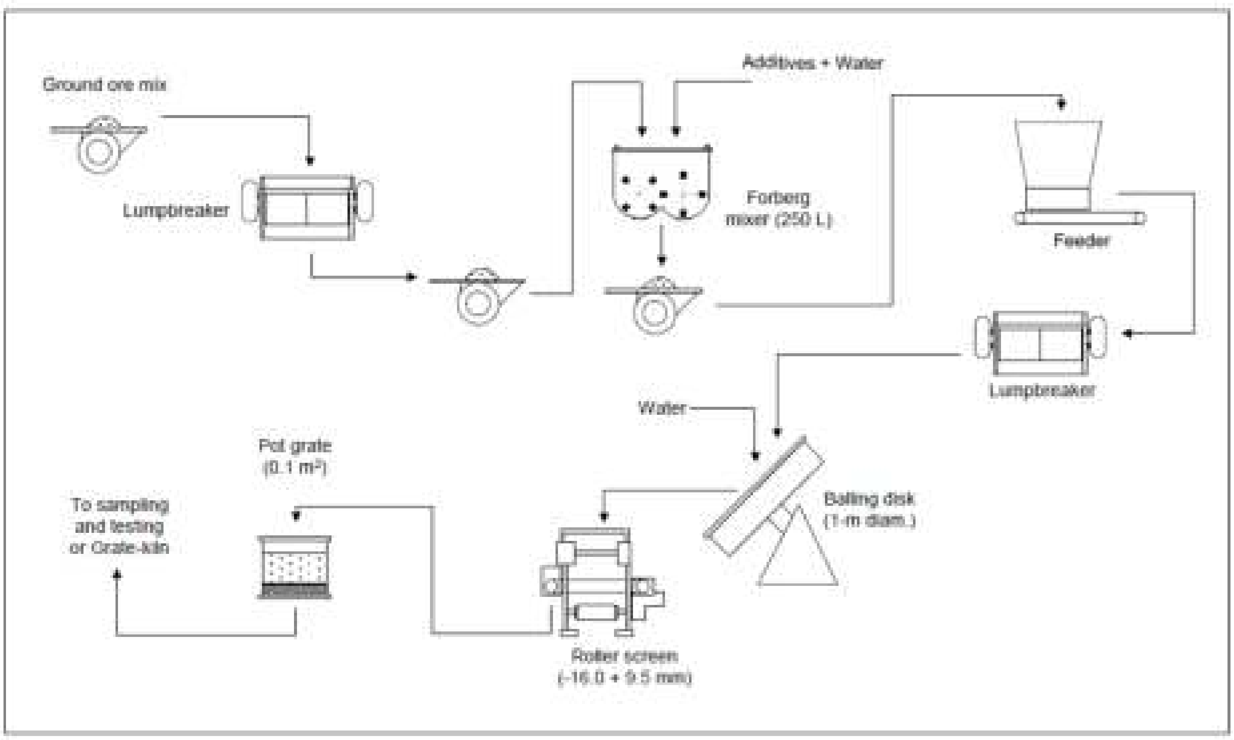


Figure 13-15: Pelletizing test flowsheet – Corem Quebec City

13.2.3.3 Balling Tests

The purpose of performing laboratory balling tests was to determine how the characteristic of green balls, or unfired pellets, are influenced by the amount of bentonite binder, moisture content and average particle size.

13.2.3.3.1 Sample Preparation

The unfired pellets examined were prepared from the received VTM concentrate sample and a ground sample of the received VTM concentrate. The VTM concentrate as received, was coarse with a blaine of 1290 cm^2/g and 55% of the material below 45 microns. The ground concentrate sample was prepared with 75 kg of dry solids in an 18" x 36" mill at 45 RPM. The mill contained 280 kg of grinding balls (-1"1/2" +3/4"). The grinding time was set to 17 minutes to achieve the target 1750 cm^2/g. The reground concentrate sample was finer with a blaine of 1730 cm^2/g and 70% of the material below 45 microns.

13.2.3.3.2 Methodology

A 90 kg dry batch of bentonite and limestone were blended and grinded to pelletizing fineness. Bentonite, water, and 2.5 kg of iron ore was mixed in a Hobart mixer for one minute to form pellets. Then, the pellets are screened at 1.65 mm and transferred gradually in increments of 400 g to a balling tire while water is sprayed to form green balls. The final addition of pellet feed to the balling tire weighed 700 g. The resulting green balls were removed from the balling tire. The green balls greater than 6.3 mm in size were returned to the balling tire and the remaining size fractions are added gradually to ensure the production of a narrow size distribution of green balls. The final green balls (-12.5+10 mm) were hardened in the balling tire for one minute.

A total of 45 green balls were tested; fifteen green balls were used for drop resistance measure from a height of 45.7 cm and thirty green balls were used for compressive strength measure. Half the green balls used to measure compressive strength were dried for two hours in 105 °C and the water content of the other half was measured. Two bentonite addition rates, 1% and 1.2%, were tested on each half. Moisture was varied from 7.7% to 9.4%.

13.2.3.3.3 Results and Conclusions

It was concluded that 1% bentonite binder provides adequate strength, and no further grinding is required which is atypical for magnetite and hematite concentrates. Table 13-21 shows the laboratory results of the balling tire test.

Table 13-21: Laboratory results of the balling tire test

Balling Unit	Test Series	Balling #	Pellet feed				Green Ball Quality		
			%<45 µm*	Blaine*	Bentonite	Moisture	Drop #	Wet Strength	Dry Strength
			%	cm²/g	% (kg bentonite/kg concentrate)	% (kg H2O/kg H2O + kg solid)	-	kg	kg
Tire	Concentrate as received	B1a	55.7	1410	1.0	8.1	6	1.3	7.2
		B1b	55.7	1410	1.0	8.5	10	1.4	8.3
		B1c	55.7	1410	1.0	9.3	11	1.2	7.0
		B2d	55.8	1415	1.2	7.8	5	1.2	8.3
		B2a	55.8	1415	1.2	8.5	7	1.0	9.6
		B2b	55.8	1415	1.2	9.2	20	1.4	9.3
Tire	Concentrate reground	B3d	70.0	1830	1.0	7.7	4	1.2	6.5
		B3b	70.0	1830	1.0	8.8	12	1.3	8.0
		B3a	70.0	1830	1.0	9.4	17	1.5	8.4
		B4a	70.0	1835	1.2	8.4	6	1.2	8.2
		B4b	70.0	1835	1.2	8.9	16	1.4	10.9
		B4c	70.0	1835	1.2	9.4	20	1.6	12.4
Typical targets for green balls quality							6 - 12	1.5 - 2.0	4 - 10

Increasing the bentonite addition rate to 1.2% did not significantly improve the green ball quality of the green balls, coarse and fine. Moisture ranges from 8.5% to 9.2% and 8.5% to 9.4% was deemed acceptable for the as-received concentrate and reground concentrate, respectively. The green ball characteristics were best with a moisture of 8.9% ±0.2%.

13.2.3.4 Basket Test

13.2.3.4.1 Sample Preparation

Following the completion of the balling tests, green ball pellets are dried in an oven at 105 °C for 12 hours prior to the basket test to ensure reproducibility.

13.2.3.4.2 Methodology

A basket test was performed to determine the magnitude in which average particle effects fired pellet quality.

The basket test uses an insert in the pot grate that enables green balls of different characteristics to be fired under identical conditions. For this test, two identical green ball samples for the as-received VTM Concentrate grind and two identical samples for the reground material were used to demonstrate the reproducibility of the test methods. The four samples weighing approximately 2.5 kg each was placed into four of six sections of a basket. Regular firing cycles were used to fire the baskets. Three thermocouples are used to ensure sufficient firing.

13.2.3.4.3 Results and Conclusions

The conclusion drawn was that concentrate regrinding to finer particle size distributions did not significantly improve the quality of the fired pellet. A firing temperature of 1250˚C proved appropriate to attain adequate strength. Table 13-22 shows the basket test results.

Table 13-22: Basket test results

Firing #			Basket P-1A			
Balling #			B1	B2	B3	B4
Pellet Feed	Physical and Chemistry	Concentrate	As received		Reground	
		%-325 mesh**	55.8		70.0	
		Blaine**	1415		1835	
		$\%SiO_2$*	2.2		2.2	
		$\%CaO / \%SiO_2$*	0.80		0.80	
		%MgO*	0.44		0.44	
Green balls	Moisture	%	9.0	9.0	9.0	9.0
	Drop	-	17	17	20	20
	Wet strength	kg/pel.	1.7	1.6	2.0	2.2
	Dry strength	kg/pel.	9.5	10.3	13.3	13.1
Firing	Strategy		Tenova			
	T° peak firing	(°C)	1250			
Fired Pellets	Compression (kg/pel.)	Mean	375	380	387	375
		Std	122	99	75	62
		% -140	0	0	0	0
		% -90	0	0	0	0
	Mini-Tumble	% +6.3 mm	95.8	95.7	97.2	97.1
		% -0.5 mm	4.2	4.3	2.8	2.9
	Porosity	%	27.4	27.7	26.0	26.7
	Satmagan	%	0.2	0.2	0.2	0.2
	FeO	%	0.6	0.6	0.6	0.5
	R180	% Red.	90.9	-	-	-
		% $Fe_{met.}$	63.9	-	-	-
		% Fe_{tot}	77.0	-	-	-
		% Met. *	83.0	-	-	-
		% Met. **	87.0	-	-	-

13.2.3.5 Pot-grate firings

13.2.3.5.1 Sample Preparation

Approximately 450 kg of concentrate was mixed with additives, binder and limestone fluxing agent, in a 250 mL Forberg mixer. Water was added to attain the green ball moisture target. The mixed material was fed to a lump breaker, then to a one-meter diameter pelletizing disc. Three size fractions of green balls are collected from the discharge: +16 mm, -16+9.5 mm, and -9.5mm. The -16+9.5 mm size fraction was used for the firing tests.

13.2.3.5.2 Methodology

The objective of pot-grate firings was to confirm basket test results as well as validate furnace configuration and conditions to achieve a minimum CCS of 225 kg/pellet. Table 13-23 shows the furnace configuration used in this project. Results from the pot-grate tests should be used for plant scale-up.

Table 13-23: Furnace configuration and conditions

Zone	%	Length (m)	Area (m^2)	WB (#)	T° (°C)	Time (min)
Up-Draft Drying	12.5	4	16	2.0	260	6.0
Down Draft Drying	12.5	4	16	2.0	260	6.0
Pre-Heat (RAMP T°)	18.8	6	24	3.0	260 to 1040	9.0
Firing	15.6	5	20	2.5	1290*	7.5
After Firing	6.3	2	8	1.0	1290*	3.0
Cooling	34.4	11	44	5.5	Amb	16.5
Total	100	32	128	16.0	-	48.0

The firing pot used has a surface area of 0.10 m^2 and a capacity to load up to 70 cm of green balls. Oxygen was added to help reach firing temperatures. Seventeen pot-grate firing tests were performed in this project: Six preliminary tests, eight for the production of a 320 kg sample, and three for the production of a 115 kg sample. The pellets along the side walls of the pot-grate were rejected; only the pellets in the center were used for analysis. The physical properties of the fired pellets were studied at different firing temperatures in series.

Scanning electron microscopy (SEM) and optical microscopy technologies were utilized to determine the mineralogy of the fired pellets.

13.2.3.5.3 Results and Conclusions

The fired pellets were determined to have the proper physical characteristics and strength necessary for reduction tests under commercial conditions, according to the following standard procedures listed in Table 13-24.

Table 13-24: Standard procedures for physical characterization

Determination		Procedure
Physical properties	Tumble and abrasion indices	ISO 3271
	Compressive strength (CCS)	ISO 4700
	Porosity	COREM
	Mini-Tumble	COREM
DR properties	R180	COREM
	LTD HYL	HYL
	Reducibility HYL 950°C	HYL
BF properties	Swelling	ISO 4698

The six preliminary pot-grate tests results in Table 13-25.

show the effect of pellet basicity and firing temperature on the physical properties of the fired pellets. Cold compressive strength was observed to be greater with increased firing temperature and basicity ratio (CaO/SiO_2 ratio).

Table 13-25: Effect of pellet basicity and firing temperature on the physical properties of the fired pellets

	TEST #		P-1A	P-1B	P-1C	P-2A	P-2B	P-2C
	DATE		2016-10-03			2016-10-05		
	DESCRIPTION		BRM V-Titanomagnetite Concentrate as received; 1.0% Bentonite 58.8% Fetot, 2.1% SiO_2, 1.7% CaO, 8.5% TiO_2, 1.3 %V_2O_5 (B2=0.8)			BRM V-Titanomagnetite Concentrate as received; 1.0% Bentonite 59.3% Fetot, 2.0% SiO_2, 0.8% CaO, 8.6% TiO_2, 1.3 %V_2O_5 (B2=0.4)		
MIX OF CONC.	%-325M		55.0			55.0		
	BLAINE (cm2/g)		1290			1290		
GREEN BALLS	%H2O (POT LOADING)		8.6	8.7	8.6	8.7	8.7	8.7
	DROPS (POT LOADING)		13	13	17	18	15	15
	WET (kg) (POT LOADING)		1.7	1.7	2.1	1.7	1.8	1.6
	DRY (kg) (POT LOADING)		9.2	8.3	9.0	10.4	10.4	10.4
FIRING	STRATEGY		Tenova + Metso	Tenova + Metso	Tenova + Metso	Tenova + Metso	Tenova + Metso	Tenova + Metso
	GAS TEMP. (°C)		1175	1230	1260	1230	1260	1290
	T° IN TOP OF BED (°C)		1228	1259	1303	1269	1287	1325
	GRATE FACTOR (t FP/day/m2)		26.9	26.1	26.8	27.1	27.1	27.1
FIRED PELLETS	TUMBLE ISO	% +6.3 mm	93.8	94.3	95.1	92.1	93.5	93.6
		% -0.5 mm	4.9	4.5	4.1	6.0	5.0	5.0
	STRENGTH (kg/pel)	AVRG	281	290	332	269	262	303
		STD DEV	136	105	129	130	114	146
	PELLETS SIZE (%-16.0 +12.5 / %-12.5+9.5mm)		19.1 / 77.1	-	-	22.3 / 99.8	-	-
	POROSITY	%	27.8	27.7	27.7	25.6	25.7	25.5
	SATMAGAN	% (T/M/B)	< 0.2 / < 0.2 / 0.2	<0.2 / <0.2 / 0.6	< 0.2 / < 0.2 / 0.5	< 0.2 / < 0.2 / 0.3	< 0.2 / < 0.2 / < 0.2	< 0.2 / < 0.2 / 0.3
	FeO	%	0.6	-	0.8	0.6	-	0.7

13.2.3.6 Additional Pellet Test Results

Sticking index, reducibility, and swelling tests were performed on the fired pellets. Results in Table 13-26 indicate that the sticking tendency and swelling tendency are low which are favorable characteristics for the HYL direct reduction process. On the contrary, a lower than anticipated reducibility suggests that the HYL reactor will have to operate at a higher operating temperature or gas cycle rate than the commercial reactor operated by Ternium in Monterrey Mexico to achieve the same metallization rate.

Table 13-26: Additional pellet test results

Pot-grate P-2A, 2B and 2C			Typical Standards
Sticking HYL	% sticking	5.4	30 maximum
	% reduction	86.9	90%
	K (x10^2) min^{-1}	3.0	4.0 minimum
Reducibility HYL 950 deg C	% reduction	93.3	95%
ISO Swelling	%Vol	10.8	20 maximum
	% reduction	59.4	

13.2.4 Tenova HYL Test Work

13.2.4.1 Objective

Tenova HYL, in Mexico, was selected to carry out testwork to determine the VTM concentrate reducibility under commercial conditions.

13.2.4.2 Bag Tests for Iron Ore Pellet Samples

13.2.4.2.1 Sample Preparation

Two 15 kg samples of fired pellets produced from the preliminary pot-grate tests, with a basicity ratio of 0.8 and 0.4, were prepared. Although the basicity ratio does not appear to influence the metallization of the pellets based on the R180 results, it does impact the low temperature disintegration according to the LTD HYL test results (see Table 13-27). For this reason, Tenova selected a basicity of 0.4 to be the chemistry to produce the 320 kg and 115 kg samples used in subsequent pot-grate firing tests. In addition, flux addition experimentation during smelting testwork would be limited with a higher basicity sample. The R180 results obtained during the preliminary pot-grate tests confirm the results obtained during the basket tests.

Table 13-27: Effect of basicity ratio on metallization and low temperature disintegration of pellets

		Pot-grate	
		P-1A+P-1B+P-1C	P-2A+P-2B+P-2C
B2		0.87	0.43
Kg shipped		15	15
Date shipped		2016-10-14	2016-10-14
Tumble ISO		94.4	93.1
CCS (kg/pel)		301	278
R180	% Red	91.0	91.9
	% Fe_{met}	65.3	63.1
	% Fe_{tot}	76.4	76.3
	% Met. *	85.5	82.7
	% Met. **	87.1	88.4
LTD HYL	% Unbroken	41.1	16.5
	%+6.3mm	66.9	22.3
	%-3.15mm	23.5	69.9

It was observed that the 320 kg sample produced has similar properties to the 15 kg sample (B2=0.43) produced from the preliminary tests (see Table 13-28). The firing conditions for the production of both the 320 kg and 115 kg samples were equivalent to the P-2C preliminary pot-grate test firing conditions.

Table 13-28: Pot-grate firing test

		Second sample from pot-grate series 3 and 4
B2		0.40
Kg shipped		320
Date shipped		2016-11-29
Tumble ISO		92.7
CCS (kg/pel)		266
R180	% Red.	N/A
	% $Fe_{met.}$	N/A
	% Fe_{tot}	N/A
	% Met. *	N/A
	% Met. **	N/A
LTD HYL	% Unbroken	N/A
	%+6.3mm	N/A
	%-3.15mm	N/A

13.2.4.2.2 Preliminary Bag Test

This was a comparative test between two formulations of pellets received from COREM (identified as BRM0.4 and BRM0.8 based on their binary basicity) and a reference pellet from Ternium Las Encinas (identified as Sample X). A first bag test was done on January 17 to observe the behavior of the prepared pellets samples for the first time in an actual Direct Reduction reactor.

The table below shows the composition of both pellet formulations as reported by COREM, in Canada.

Table 13-29: Composition of both pellet formulations as reported by COREM, in Canada

B2		0.84	0.42
% SiO_2	%	2.0	1.9
% Al_2O_3	%	1.24	1.28
% Fetot	%	59.2	59.6
% FeO	%	0.64	0.60
% MgO	%	0.40	0.38

B2		0.84	0.42
% CaO	%	1.67	0.80
% Na_2O	%	< 0.10	< 0.10
% K2O	%	0.01	0.01
% TiO_2	%	8.53	8.69
% MnO	%	0.18	0.17
% P2O5	%	< 0.01	< 0.01
% Cr_2O_3	%	0.11	0.1
% V_2O_5	%	1.30	1.35
% ZrO2	%	< 0.02	< 0.02
% ZnO	%	0.03	0.03
% LOI	%	-0.12	-0.04

It is important to point out that this composition indicates a total Iron content of around 59% for both pellet samples when DR grade pellets usually have values above 65%. In the composition there appears to be a significant amount of Titanium and Vanadium in the form of oxides, which are regularly practically absent in DR grade pellets also.

For the comparative bag test, a total of 100 bags were prepared. Each bag consisted of three separated sections each one containing a different pellet sample. The bags were fed into the DR Reactor by the charge system and later recovered in the discharge system. Figure 13-16 shows an example of the prepared test bags.

Figure 13-16: Example of the prepared test bags

The reduced pellets from the recovered bags were gathered into composites separated by pellet types to measure particle size (degradation), % of metallization and % of totalcarbon content. Visually, the pellets from BRM 0.4 and 0.8 look overall unbroken for most of the sample even after having passed through the DR reactor. Figure 13-17 shows a comparison of the pellets available in each compartment of one test bag after the test.

Figure 13-17: Comparison of the pellets available in each compartment of one test bag after the test

The results regarding particle size after reduction (fines generation) showed that the BRM pellet samples exhibit low degradation with particles lower than 1/4" being from 3-5% in both the 0.4 and 0.8 samples which had similar behavior. Overall, the results were good showing a moderate to low tendency for fines generation. The Sample X pellet gave a high percentage of particles below 1/4", and this is an expected result from this pellet. Table 13-30 shows the results for the three pellets evaluated.

Table 13-30: Evaluation results for the three pellets

Pellet	+1/4 (%)	-1/4 (%)
BRM0.4	95.4	4.6
Sample X	83.7	16.3
BRM0.8	96.4	3.6

The results regarding the percentage of metallization for both types of BRM pellets, either 0.4 or 0.8 basicity, showed very similar behaviors, with an overall metallization around 90%. The carbon content for BRM samples is also very similar, from 3.5 to 3.7% overall. This is a moderate to high result in the case of ZR process. The Sample X pellet results showed 93.7% metallization and 4.7% carbon content, which are expected for this pellet. Table 13-31 gives a summary of these results.

Table 13-31: Metallization and carbon content of the BRM pellets

Pellet	%Met	%C	%S
BRM0.4	90.5	3.7	0.0043
Sample X	93.7	4.7	0.0056
BRM0.8	90.5	3.5	0.0066

The average metallization and total carbon content of the production of the date of the test were 94.3% and 4.0%, respectively.

Since the results present very little variation in the behavior of the BRM0.4 and the BRM0.8 pellet in the DR reactor, the larger sample for the 200 kg production campaign was done with the BRM0.8 formulation for its higher basicity.

13.2.4.2.3 DRI Production Campaign General Information

The iron ore pellet samples of BRM0.8 from COREM were received in two shipments comprise several plastic containers, each varying from 16 to 25 kg of pellets.

The pellets were visually examined to observe if the samples were suitable for the test. The pellets had very well-rounded shapes with no visible cracks. They had greyish coloration and some pieces had irregular not-rounded shapes. The samples looked suitable to proceed with the bag tests (see Figure 13-18).

Figure 13-18: Pellet samples for bag tests

For the overall campaign, a total of 12 bag tests were done at different dates from the months of January to the month of March, depending on DR Reactor availability. For each bag test performed, 150 bags were prepared and fed into the 3M5 Reactor shaft in Ternium Facilities through the atmospheric bin at the top of the charging system. Each test bag consisted of two compartments of around 125 g of BRM pellets for a total of 250 g per test bag. The test bags prepared for this campaign are 2.5 times larger than the ones usually used for these kinds of tests (of around 100 g of content). Figure 13-19 shows a representation of the sample contents inside each test bag.

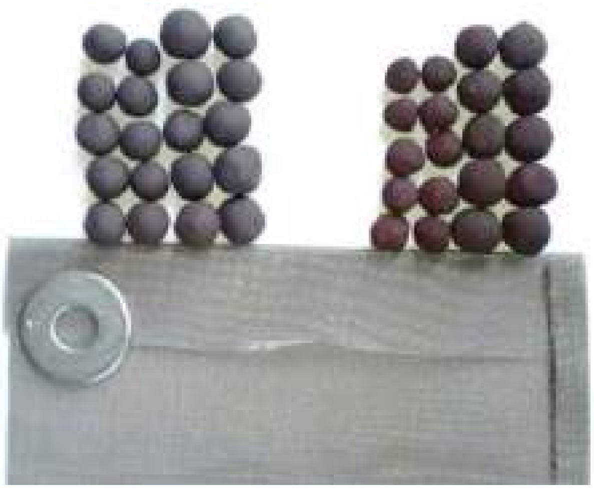


Figure 13-19: Representation of the sample contents inside each test bag

Test bags containing BRM pellets were identified by means of a round shape steel piece used as a tag with a consecutive number engraved; this is done to identify the overall process residence time of the test bags. Please refer to Figure 13-20 below for reference.

Figure 13-20: Tag identification

The tests were carried out on the programmed dates from 12:30a.m. up to 5:30p.m. for a total test time of 17 hours from charging to recollection of the test bags. All mesh-made bags were fed into the charging system of the DR Reactor shaft at a rate of seven bags each 15 minutes. Then, the total feeding time was 5 hours 30 minutes. The bags were recovered as much as possible in the cold DRI discharge conveyor. The time to retrieve the bags was from 7:30a.m. to 5:30p.m. in each test. It is important to point out, just a percentage of the original 150bags are recovered, and from this percentage, some present some damage where the content might be either lost or it could be mixed with some foreign material, making necessary to discard such test bags. The average effective recovery percentage per bag test for the whole campaign was around 70%.

It is important also to point out that, during some tests, the DR Plant could experience some transitory conditions which would obligate to discard the whole test since the conditions for the reduction did not remain homogenous during the testing time.

13.2.4.2.4 Bag Collection and Classification

The following after each bag test, the test bags recovered from the DR process were registered in a list by their corresponding sample identification tag to check their residence time. After this, the samples were reintegrated into their respective again and separated respectively into halves using a sample cutter; a small sample taken from one half was crushed, grinded and homogenized for quality analysis. Each quality analysis took from two to three days depending on the laboratory workload.

Quality analysis consisted in determining total iron (FeT) and metallic iron content (FeM) in order to calculate the percent of Metallization (%Met) as well as total Carbon and Sulfur content of the prepared sample.

13.2.4.2.5 Bag Test Results

The total test bags that were recovered from the reactor discharge system varied slightly from one test to another but remaining within a 70% recovery of the original 150 bags at each test. Figure 13-21 shows an example of the appearance of the test bags after reduction. The material inside shifted into a dark grey color characteristic of reduced iron for the case of both types of pellets. Most of the original content looked to be with little physical damage.

Figure 13-21: Appearance of the test bags after reduction

The average residence time for the test bags in the overall campaign was around 9.3 hours. It is important to mention that this is the overall residence time and the reduction time of the pellet is a fraction within this residence time. This residence time corresponds to the 3M5 Plant from charging to discharging of the test bags and will vary in other DR Plants depending on the size of the Reactor and the production rate.

Table 13-32 below shows the average quality results of the overall samples obtained in the campaign.

Table 13-32: Average Quality Results of the Overall Samples Obtained in the Campaign

Pellet	%Met	%C	%S
BRM	90	3.58	0.0069

The carbon content of the BRM pellet was 3.58%; these values are usually achievable only by ENERGIRON ZR process. The carbon content in the case of the reduced samples is attributed to the carbon deposited during the DR process in the reduction zone of the Reactor furnace, which is primarily found in the form of Iron Carbide or Cementite (Fe_3C).

Nevertheless, it is important to point out that the usual values of Carbon content in DR pellets that pass through the ENERGIRON ZR process have around 3.0 to 4.0% of Carbon; thus, for the case of both samples, the amount of Carbon deposited is within the values normally expected from the ZR DR process.

As shown in Table 13-32, the average metallization percentage is 90.00% for the whole campaign; which is a good result, although it could be improved. Sample X Iron Ore pellets used in 3M5 Reactor usually reach a metallization percent of 93-94% which is a result of tuning up the process conditions of the DR Reactor for this particular iron ore pellet.

The average sulfur content of the reduced pellets is 0.0069% for the whole campaign. However, if some of this sulfur goes along with the exhaust gas into the reduction gas circuit, it would be removed in the CO_2 absorption system before it gets recirculated into the Reactor shaft again.

In Table 13-33, a summary for the number of tests performed, the dates they were performed and the result in kilograms of DRI obtained in each test is presented. The campaign began on January 24th with the first bag test and the last bag test was performed on March 15.

Table 13-33: Summary of the Tests Performed

Campaign bag test no.	Date performed	Result (kg of DRI)	Comments
01	24/01/2017	18.57	Beginning of Campaign
02	26/01/2017	18.07	
03	09/02/2017	Discarded	Transitory Condition in the Reactor
04	20/02/2017	20.97	Restart after a DR Plant programmed shutdown
05	22/02/2017	20.75	
06	28/02/2017	15.38	
07	02/03/2017	20.74	
08	06/03/2017	18.73	
09	08/03/2017	19.65	
10	10/03/2017	18.56	
11	13/03/2017	17.82	
12	15/03/2017	18.68	End of Campaign

It is important to point out that, a period of at least one day after a bag test is required before performing the following one. At the same time, the presence of transitory conditions, like in the case of the test performed on February 9th, would affect the resulting sample, thus they had to be discarded from the rest to avoid variation in the procedure. Also, availability of the DR Plant is subject to events such as shutdown of the same for programmed maintenance purposes.

The main DR plant average conditions during the entire campaign are summarized in Table 13-34.

Table 13-34: DR Plant average campaign conditions

Met	[%]	94.21
C	[%]	3.66
Production	[TonDRI/h]	94.5
FER	[NCM/TonDRI]	1,385
TER	[°C]	1,090
INGR	[NCM/TonDRI]	88
INGE	[NCM/TonDRI]	108
FEE	[NCM/TonDRI]	470

FER stands for the flow of reducing gas at the inlet of the DR Reactor, TER is the temperature at the inlet of the reactor, INGR is the inlet of natural gas to the reduction circuit, INGE is the inlet of natural gas at the cooling circuit and FEE is the inlet flow of cooling gas to the reactor.

The overall collected sample of DRI from BRM iron ore pellet was around 207 kg. This DRI was appropriately packed and shipped to Tenova Pyromet in order to be used in further testing regarding melt shop processing.

13.2.4.3 Conclusions

All the results reported in this document were obtained by testing the samples of BRM iron ore pellets in a single comparative test as well as a campaign comprised by 12 bag tests performed at different dates. Any further interpretation and/or extrapolation of these results will depend on how representative the evaluated samples are of the actual pellet to be used in the intended process, as well as the process conditions.

- The Bags Test results showed that the BRM pellets, under the actual DR plant conditions of 3M5 DR Reactor, reach a percentage of metallization of 90.00, In case of higher metallization values are required, a proper tuning of the plant conditions is possible;
- The carbon deposited into the BRM pellets was 3.58, which is a normal result expected in the process; this is a highly desired property for later steps in the steelmaking process;
- In an overall conclusion, after the performed bags test, we consider/classify BRM pellets received as suitable to be used in ENERGIRON ZR DR Plant; a 200 kg sample of the DRI obtained was prepared and shipped to Tenova Pyromet for further analyses.

13.2.4.4 Conclusion of the additional test on the VTM pellets July 2018

A series of follow-up tests performed by Tenova to confirm the viability of P2 pellets (pellet formulation used in the Feasibility Study) and to explore the possibility to optimize reducibility and minimize degradation of DRI by changing the amount and type of fluxes added during the pelletizing process. These tests were also performed to confirm or optimize the design included in the 2017 Feasibility Study.

Reducibility testing was completed on three different pellet compositions; P1, P2 (FS baseline composition) and P3, and a pellet using a high Blaine P2 composition designated as P4. Test conditions were specifically modified to BRM's Energiron ZR pressure, temperature and gas composition. Tenova's also processed these pellets using standard bag testing at a full-scale commercial plant, to better understand the VTM pellet behavior and limitations in the plant designed for BRM VTM ore and polymetallic business plan. HYL LTD testing protocol was also performed on these three pellet compositions.

The results of this testing and optimization program demonstrated:

- Bag Tests at Ternium confirmed very good results relative to other pellets – especially the P3 pellet composition, which reached 93.4% metallization;
- HYL Infinite Time Reduction test showed a reduction of 97.2% for the same pellet composition of the Feasibility Study (P2) and 99.5% reduction with the P3 pellet composition, indicating no absolute barriers to reduction of the iron oxides present in the matrix;
- HYL standard Reducibility @950°C performed at LESA laboratory showed very good reducibility for the P3 pellet and suitable reducibility for the P2 pellet;
- HYL standard LTD testing performed at LESA laboratory confirms that this test does not properly represent fines generation for this type of ore. On the other hand, both, the bag tests and ISO 11257:2015 indicate low fines generation for all the VTM pellets tested;
- The ISO 11258:2015 (DR90) testing showed low reduction for all pellet formulations. This is due to the low temperature, pressure and H2 partial pressure employed by this method in contrast with the much higher levels used in the ZR process. HYL relies on representative bag testing, and HYL standard reducibility testing run under conditions that more closely emulate BRM's ZR Energiron design conditions;
- Regrinding of the VTM concentrate to finer than 1850 cm^2/g specific surface area prior to pelletizing did not result in any improvement of the metallurgical properties of the pellet.

13.2.5 Tenova Pyromet Test Work

13.2.5.1 Objective

The direct reduction iron (DRI) prepared by Tenova HYL, in Mexico, was used the prepare multiple products including pig iron, titanium and vanadium slags. Tenova was able to define various assumptions and design calculations for the development of the BRM process flow sheet based on the results from testing these products.

Under the direct supervision and project management of Tenova Pyromet, all pyro metallurgical and vanadium slag processing test work was conducted by Mintek in South Africa in accordance with the details outlined in the report issued by Mintek (Bench scale demonstration of the production of FeV from Blackrock Chibougamau titaniferous magnetite).

The primary purpose of this test work was to investigate the smelting behaviour and particularly when the pig iron produced from smelting is subjected to oxidation (converting) step to recover the vanadium to a slag phase. The slag was processed further to recover vanadium as V_2O_5. In order to accomplish this sufficient vanadium, bearing pig iron must be produced. For this purpose, the smelting stage was conducted using the DC arc furnace. Based on the characteristics of the laboratory scale produced V_2O_5 sample, synthesized V_2O_5 and V_2O_3 was processed to produce 80-FeV in a tilting 100 kVA DC arc furnace.

The test work objectives were to confirm normal industrial process parameters for the following:

- A direct reduced iron (DRI) product produced by Tenova HYL
- The DRI used as a precursor for the production of:
  - Pig iron;
  - High purity pig iron (HPPI);
  - Titanium rich slag;
  - Vanadium bearing slag;
  - Ammonium metavanadate (AMV);
  - Vanadium pentoxide (V_2O_5);
  - Ferrovanadium (FeV80); and
  - Treatment of various waste streams.

Vanadium extraction from pig iron convertor slag is well understood with decades of demonstrated technical and commercial viability in various locations globally. A team of industry experts in the field of vanadium technology-assisted Tenova in developing a process flow sheet tailored specifically for the needs of BRM. A program of laboratory and bench scale test work was developed to confirm various assumptions and design calculations for the BRM process flow sheet. In Tenova's view, the testwork executed is sufficient to meet the FS requirements.

The DRI was smelted in a bench scale Direct Current (DC) arc furnace to produce a Vanadium bearing pig iron and a titanium rich slag. The V-bearing pig iron was further processed in a converter to recover a vanadium rich slag and a high purity pig iron. The V rich slag was subsequently treated by roast-leach process in order to solubilize the vanadium and thereafter precipitate the vanadium as Ammonium Metavanadate (AMV). The AMV was calcined to Vanadium Pentoxide (V_2O_5) flake, for the subsequent alumina-thermic processing to ferro-vanadium (FeV_{80}). Prior to the precipitation of AMV, the silica content, a potential contaminant, was reduced to acceptable levels.

The waste streams from the roast-leach, de-silication and AMV precipitation were evaluated by process monitoring and chemical analyses.

The procedures for test work consisted of the following:

- DRI smelting;
- V-slag conversion from Pig iron;
- V-slag roasting and leaching;
- De-silication of V solution;
- AMV precipitation;
- V_2O_5 flake production;
- FeV_{80} production.

13.2.5.2 DRI Smelting

Smelting in the DC arc furnace was preceded by laboratory scale smelting tests of the DRI pellets in an induction furnace.

It should be noted the feasibility study was based on a predicted DRI analysis which had 94% metallization and 4.32% C, 10.78% TiO_2 and 1.57% V_2O_5 compared to the DRI analysed at Mintek. The metallic Fe content of the DRI used in the test work indicates 90% metallization, the complete analysis is shown in Table 13-35.

Table 13-35: Laboratory Scale Smelting Tests of the DRI Pellets

DRI	%
Fe	67.75
FeO	9.06
Fe_2O_3	1.22
TiO_2	11.08
V_2O_5	1.62
SiO_2	2.57
Al_2O_3	1.65
CaO	1.55
Cr_2O_3	0.15
MgO	0.4
C	3.24

*Note: This analysis was used in the mass balance calculations as the baseline

13.2.5.2.1 Laboratory scale smelting

The laboratory scale DRI smelting was conducted in an induction furnace to establish the following conditions for the laboratory tests:

- The Fe and V recoveries were highest at 1650°C;
- Smelting could be performed with no flux and no additional C at 1650°C.

A good slag and metal separation were achieved indicating a workable slag presence during smelting. The outcome of the test work was as follows:

- Metal analyses used as a base to estimate the recoveries. The average chemical analysis of the metals from the tests are shown in the table below;
- V in metal of 1.08% corresponds to 96% V recovery. This is significantly higher than the 90% recovery assumed in mass balance calculations with 1.02% V content in metal.

Table 13-36 shows the average metal composition achieved during laboratory scale smelting.

Table 13-36: Average Metal Composition

Average Metal Composition	%
Fe	93
C	2.58
Si	1.97
Cr	0.34
Mn	0.76
Ti	0.39
V	1.08

13.2.5.2.2 Bench scale smelting

The objective of the bench scale smelting test work was to:

- Confirm selected lab-scale parameters;
- Produce sufficient amount of vanadium (V) bearing pig iron for subsequent tests;
- Produce a titanium bearing slag.

The smelting campaign was conducted in a 100 kVA DC furnace operating at 30 kW over a period of two days. A carbon content of 3.4% in the metal was targeted for maximum vanadium extraction without significant reduction of titanium and silicon to the pig iron.

The carbon content in the recipes was established at 4.5% to the DRI feed to achieve the desired V recoveries and to produce the desired C content of 3.4% in the pig iron.

The pig iron and slag analysis were confirmed, see Table 13-37 and Table 13-38:

- Pig iron analysis:
  - The V recovery achieved in the test work was 96%. This is significantly better than 88% recovery assumed in the FS design mass balance calculations;
  - There is a good correlation between the actual and the calculated analysis for slag analysis.

Table 13-37: Pig Iron Analysis

	Mass balance [%]	Test average [%]
Fe	94.55	93.78
V	1.02	1.08
C	3.4	3.39
Cr	0.11	0.31
Ti	0.4	0.3
Si	0.4	0.42

- Slag analysis:
  - The low TiO_2 content in the slag is attributed to the dilution caused by contamination from the crucible;
  - High MgO contents in the slag is from the MgO crucible;
  - High FeO is attributed to the entrainment of metallic iron in the slag.

Table 13-38: Mass Balance Slag Analysis

	Mass balance slag analysis [%]	Test average [%]
TiO_2	61.22	39.0
V_2O_5	0.95	0.8
Al_2O_3	9.7	14.5
SiO_2	11.3	6.7
CaO	9.1	4.6
MgO	5.76	23.2
Cr_2O_3	0.1	0.3
FeO	0.68	10.8
MnO	1	0.5

Note: A recalculation of slag composition to remove the effect of MgO crucible contamination shows a predicted analysis of 64.2% TiO_2 and 1.4% V_2O_5 which correlates favourably with the design mass balance baseline values.

13.2.5.2.3 Converting to Produce Vanadium Slag

The main objectives of converting test work were to:

- Maximize selective oxidation of vanadium in the pig iron and recovery into the slag phase;
- Produce granulated high purity pig iron.

The bench scale vanadium (V) converting test work was conducted by using hematite as the oxidizer and carbon as reductant and carburizer. The temperature in the induction furnace was maintained between 1380°C and 1450°C.

A measured 87.2 kg of V bearing pig iron was processed and 3.9 kg of V-slag was recovered. Table 13-39 shows the metal analysis was achieved.

Table 13-39: Metal Analysis

	Final specification [%]	Test [%]
C	3.5 to 4.5	3.59
Si	0.5 max	<0.003
Mn	0.05 max	<0.001
S	0.02 max	<0.0199
Ti	-	<0.0005
V	0.05	0.13
FE	95 - 96	95.5

Note the potential contaminants such as Manganese (Mn), Titanium (Ti), and Sulphur (S) are well below levels that would be damaging to steel making, allowing the HPPI product to be sold to steel producers wishing to dilute impurities from their own processes with steel scrap.

Comparing the vanadium contents of the starting metal and final metal, 88% of the vanadium has reported into the slag. Due to the scale limitations of the test work, the vanadium concentrations in the final metal could not be reduced below 0.13% V. In the actual operation, the vanadium concentration will be well below 0.1% V. The V-slag analysis completed in shown in Table 13-40.

Table 13-40: Vanadium Slag Analysis

	Market Baseline [%]	Test [%]
V_2O_5	23 min	19.1
FeO	35 - 40	45.9
SiO_2	15 max	11.8
Al_2O_3	-	5.6
MgO	5 max	8
CaO	4 max	0.7
Cr_2O_3	4 max	2.8
MnO	5 max	2.2
TiO_2	12 max	5.4

The higher levels of FeO resulted in the dilution of the V_2O_5 content in the V-slag.

13.2.5.3 Roast-leach Process for Vanadium Recovery from Slag

The objective of the roasting stage was to convert vanadium in the slag to the water-solublesodium metavanadate ($NaVO_3$) according to the general reaction:

$Na_2CO_3 + V_2O_5 \rightarrow 2NaVO_3 + CO_2$

The parameters that were established for the best roasting conditions for the vanadium conversion to the water leachable phase included roasting temperature, roasting time, alternative sodium salt blends, alternative slag to carrier ratios, and effect of alumina addition to the roaster feed.

The laboratory scale test work successfully established the best conditions for the bench scale tests.

- Roasting at 800°C for 90 minutes;
- Salt mixture of Na_2CO_3 & Na_2SO_4, compared to Na_2CO_3 only and Na_2CO_3 and NaCl;
- V_2O_5 concentration of 13% in the roaster feed was found to be optimal;
- Positive results in terms of V recoveries were obtained with an alumina addition in the roaster feed.

The V-slag bench scale test work was processed under the conditions presented in Table 13-41.

Table 13-41: V-slag Bench Scale Test Work Conditions

Parameter	Conditions
Roasting temperature	800 deg C
Roasting time in minutes	120
Roasting reagent	Na2CO3
V_2O_5 in feed	13%
Roasting reagent	Na2CO3

The roast was first cooled to 200°C and subsequently leached in water for 1 hour before clarification.

The clarified vanadium solution analysis is shown in Table 13-42.

Table 13-42: Clarified Vanadium Solution Analysis

	Mass balance [g/l]	Concentration [g/l]
V	30-40	39.8
Si	0.05	0.428
Cr	-	3.53
Al	-	0.93

From the analysis above, desilication of the concentrated vanadium solution is required prior to the precipitation of AMV.

The solid residue analysis is presented in Table 13-43.

Table 13-43: Solid Residue Analysis

	Mass balance [g/l]	Washed residue [%]
V_2O_5	0.77	0.66
Fe_2O_3	46.47	49.95
Na_2O	16.14	8.25
Cr_2O_3	1.95	0.82
Al_2O_3	-	5.65
SiO_2	14	12.94
CaO	-	1.61
MgO	-	7.11
TiO_2	11.15	7.62
MnO	1.39	3.03

The V_2O_5 concentration in the solid residue (tailings) from the test work was considerably lower than the predicted value in the mass balance for baseline, indicating a higher than expected yield. This result was achieved with 13% V_2O_5 in the roaster feed versus 6% V_2O_5 roaster feed head grade assumed in the design mass balance calculations, indicating possibility of operating with reduced roaster carrier recycle load. Reduced recycle load would significantly lower roaster capital requirements and reduce operating costs.

13.2.5.3.1 De-silication of V solution

The objective of this process step is to remove Si contained in the leach solution prior to the AMV precipitation to less than 0.05 g/ L Si.

The analyses of the V solution before and after de-silication is shown in the Table 13-44. The results indicate the de-silication test was successful with 0.013g/L Si remaining in the solution after the test.

Table 13-44: Analyses of the V Solution before and after de-silication

	Before de-silication [g/l]	After de-silication [g/l]
Si	0.43	0.013
V	39.8	19.7
Cr	3.5	3.8
Al	0.93	0.32

The analysis of the residue collected from de-silication is shown in Table 13-45.

Table 13-45: Analyses of the residue collected from de-silication

Washed residue	[%]
SiO_2	34.4
V_2O_5	1.21
Cr_2O_3	0.92
Al_2O_3	27.6

13.2.5.3.2 AMV precipitation

The objective was to precipitate AMV from the de-silicated liquor produced. The analysis of AMV is presented in Table 13-46.

Table 13-46: AMV Analysis

AMV	[%]
V_2O_5	77.3
Cr_2O_3	1.5
Al_2O_3	0.91
SiO_2	1.28

The main objective of producing AMV from the de-silicate liquor was achieved.

13.2.5.3.3 V_2O_5 Flake Production

The main objective was to de-ammoniate the AMV to V_2O_5 and subsequently fuse the V_2O_5 to flake. Table 13-47 shows the composition of the V_2O flake.

Table 13-47: V2O5 Flake composition

	Aim [%]	V_2O_5 flake [%]
V_2O_5	>98.0	81.52
SiO_2	-	2.52
Al_2O_3	-	0.56
Cr_2O_3	-	4.5
FeO	-	11.66
MnO	-	2.23
NiO	-	3.31

The main objective of producing a V_2O_5 flake was achieved.

The main reason for not achieving the 98% V_2O_5 specification was due to the contamination caused by the metal crucible used during the fusion process (see Figure 13-22).

Figure 13-22: Contamination from the metal crucible

13.2.5.4 Aluminothermic Reduction and Smelting Test

The primary objective of the overall project was to demonstrate the production of FeV_{80} from the BRM V-slag. Thus, the V_2O_5 flakes produced from the BRM material would have to be subjected to aluminothermic reduction to produce the FeV_{80}. However, the typical FeV production procedure entails the use of a minimum of 10 kg of the V_2O_5 flakes which could not be attained from the small quantity BRM material. Hence, a third party V_2O_5 flakes were used to demonstrate the FeV_{80} production commercial viability. The FeV_{80} analysis is presented in Table 13-48.

Table 13-48: FeV_{80} Analysis

FeV_{80}	[%]
Al	0.59
Cr	0.26
Fe	21.9
Mn	0.25
Si	0.32
V	76.6

Figure 13-23 is an actual visual of the FeV_{80} produced.

Figure 13-23: Actual visual of the FeV80 produced

13.2.5.5 Titanium Slag Digestion

Digestion tests were performed to evaluate the titanium slag produced from the smelting of the DRI in the OSBF electrical furnace. The goal of the tests was to confirm the digestibility of the slag and to compare with other slags in the market.

BlackRock engaged Symphony Trade and Investment 43 (Pty) Ltd, a South African company, to perform digestion tests to compare with existing slag on the market. The tests were performed at Mintek laboratory in South Africa. The experiments from various titania slag concentrations comprised of the following: grinding and sulfuric acid digestion of the slag, water leaching and digestion of the cake, clarification of the leaching solution.

The tests showed the digestion of the slag through a sulphate process to be an effective route for the recovering of TiO_2 and compared very well to high-grade TiO_2 slag produced on the market. Although BlackRock titanium slag tested was not as per the final specifications expected, the slag still provided the necessary technical validation that it can be digested to recover 79% of the TiO_2 into solution.

13.2.5.6 Conclusion

The test work has demonstrated the technical viability of the process flowsheet proposed in the feasibility study which includes smelting of DRI, conversion of V-slag, solubilization of vanadium, de-silication, precipitation of AMV, de-ammoniation and fusion of AMV to V_2O_5 flake, and production of FeV_{80} from V_2O_5 flake.

- The V recovery achieved in the test work was 96%. This is significantly higher than the 90% estimation in the mass balance. There is a good correlation between the actual and the calculated analysis for slag analysis;
- In the converting tests comparing the vanadium contents of the starting metal and final metal, 88% of the vanadium has reported into the slag. Due to the scale limitations of the test work, the vanadium concentrations in the final metal could not be reduced below 0.13% V. In the actual operation, the vanadium concentration is expected to be well below 0.1% V corresponding to V recoveries of higher than 90%;
- Optimum roasting conditions were found to be 800 °C for 120 minutes. Furthermore, V_2O_5 concentration of 13% in the roaster feed was found to produce better results than 6% V_2O_5 feed. This means the amount of carrier material, envisaged in the process design, recirculated to dilute the V_2O_5 concentration in the feed can be reduced drastically;
- Target V concentration in the leach solution of 30 – 40 g/L was achieved easily;
- The V_2O_5 concentration in the solid residue (tailings) from the test work was considerably lower than the predicted value in the mass balance for baseline, indicating a higher than expected yield. This result was achieved with 13% V_2O_5 in the roaster feed versus 6% V_2O_5 in the actual plant design;
- De-silication tests were successful in reducing the Si concentration in the leach solution from 0.43 g/L to 0.013 g/L, which is significantly lower than the target value of 0.05 g/L;
- V_2O_5 concentration in the de-silication residue was 1.2% against the target value of 2% in the baseline indicating better than expected V losses into the waste;
- The main objective of producing AMV from the de-silicated liquor was achieved;
- The main objective of producing a V_2O_5 flake was achieved;
- The main objective of producing a FeV_{80} alloy was achieved.

14. Mineral Resource Estimates

14.1. Historical Resource Database

The BlackRock property is divided into two deposits; the Southwest deposit (drillhole designator SW-drill panel#-hole#; designated SW many of the figures in this section) and the Armitage deposit (drillhole designator AE-drill panel#-hole#). SGS completed an initial resource estimation of the Southwest deposit in the summer of 2010. At that time, the focus was on magnetite only, and estimates of the $\%Fe_{Mag}$ were interpolated in approximately 240,000 blocks 10 x 5 x 10 m filling a mineralized envelope with outlines interpreted on drill sections.

Values of the $\%Fe_{Mag}$ were derived analytically from Satmagan, or in a limited number of cases, estimated from WRA ($\%Fe_T$ derived from Fe_2O_3) using linear regression relations, for approximately 2,700 3 m core hole or trench samples (540). A total of 49 core holes totalling 12,441 m were available. They were drilled on northwest-southeast panels, 100 m apart from each other on the northeast part of the deposit (panels 1600, 1700, 1800 sequentially through 2600, with three holes on each panel), and 200 m apart from each other on the southwest part of the deposit (sections 100 through 1400, with two holes on each panel).

An initial resource estimation of Armitage was also completed by SGS in the spring of 2011. It too was restricted to magnetite, with estimates of $\%Fe_{Mag}$ of approximately 290,000 blocks 10 x 5 x 10 m filling an interpreted mineralized envelope. Estimates derived from approximately 2,600 3 m core hole samples. A total of 28 core holes totalling 8,093 m were available. They were drilled on 14 NNW-SSE panels, 200 m apart from each other.

14.2. 2013-2014 Resource Database

This resource update is based on a drillhole database, dated February 11, 2013 (File *DH4 FEB 11 2013.xlsx*). Some Satmagan data were updated and a few density values were added in February 2014 (File: *Sat correction for SGS feb 2014.xlsx* of February 24,2014). No additional drilling, sampling and or no other analytical work has been done on the data since 2014.

14.2.1. Southwest Deposit

The Southwest resource update is based on data from up to 115 holes representing 66 more than in the 2010 study (see Table 14-1 below).

Table 14-1: All drilling in the Southwest Deposit

Drillhole	Drillhole	Drillhole	Met Hole	Met Hole	Met Hole
SW-01-01	SW-01-02	**SW-01-03**			
SW-02-01	SW-02-02	**SW-02-03**			
SW-03-01	**SW-03-02**	**SW-03-03**			
SW-04-01	SW-04-02	**SW-04-03**			
SW-05-01	**SW-05-02**	**SW-05-03**			
SW-06-01	SW-06-02	**SW-06-03**	**SW-BK-06**		
SW-07-01	**SW-07-02**	**SW-07-03**			
SW-08-01	SW-08-02	**SW-08-03**	**SW-BK-08**		
SW-09-01	**SW-09-02**	**SW-09-03**			
SW-10-01	SW-10-02	**SW-10-03**	**SW-BK-10**	**SW-BK-10B**	
SW-11-01	**SW-11-02**	**SW-11-03**			
SW-12-01	SW-12-02	**SW-12-03**			
SW-13-01	**SW-13-02**	**SW-13-03**			
SW-14-01	SW-14-02	**SW-14-03**	**SW-BK-14**	**SW-BK-14B**	**SW-BK-14C**
SW-15-01	**SW-15-02**	**SW-15-03**			
SW-16-01	SW-16-02	SW-16-03	**SW-BK-16**		
SW-17-01	SW-17-02	SW-17-03			
SW-17+50-01					
SW-18-01	SW-18-02	SW-18-03	**SW-BK-18**	**SW-BK-18B**	**SW-BK-18C**
SW-18+50-01					
SW-19-01	SW-19-02	SW-19-03			
SW-19+50-01					
SW-20-01	SW-20-02	SW-20-03	**SW-BK-20**		
SW-20+50-01					
SW-21-01	SW-21-02	SW-21-03	**SW-BK-21**	**SW-BK-21B**	
SW-22-01	SW-22-02	SW-22-03	**SW-BK-22**	**SW-BK-22B**	**SW-BK-22C**
SW-23-01	SW-23-02	SW-23-03			
SW-24-01	SW-24-02	SW-24-03			
SW-25-01	SW-25-02	SW-25-03			
SW-26-01	SW-26-02	SW-26-03			
GeoTech	**TG-FW-1**	**TG-FW-2**	**TG-HW-1**	**TG-HW-2**	
Misc.	**MRN-09**	**MRN-10**	**MKBY-08**	**MKBY-09**	**SWPT**

Note: Black font represents information used in the 2010 resource estimate, **Red Bold** represents the drilling data added since 2010 and used for the current resource model, and **Blue Bold** represents the vertical holes drilled for bulk pilot testing.

The following is a summary of the new drillhole representation in the database:

a. A third NQ hole on panels 100, 200, 400, 600, 800, 1000, 1200 and 1400 (eight new holes: SW-01-03…SW-14-03);

b. Three new NQ holes on drill panels 300, 500, 700, 900, 1100, 1300 and 1500, with no previous drilling (total 21 new holes: SW-03-10…SW-15-03);

c. Step-out NQ holes on drill panels 1750, 1850, 1950 and 2050 (four new holes: SW-17+50-01…SW-20+50-01);

d. Vertical HQ metallurgical bulk sample holes on drill panels 600, 800, 1000 (2), 1200 (3), 1400 (3), 1600, 1800 (3), 2000, 2100 (2) and 2200 (3) (20 new holes: SW-BK-06…SW-BK-22C);

e. NQ holes to test footwall and hanging wall of the mineralization for pit wall stability studies (eight new holes: TG-FW-1…TG-HW-4);

f. Miscellaneous holes (five historical holes MRN-09 and 10, MKBY-08 and 09, SWPT (BlackRock condemnation)).

There is lithologic information available along all new holes, except (f). There are assay data available (but not continuous) along all new holes, except (d) and (f).

Trench data from Southwest has not changed since 2010. Those data are not used in the derivation of the current Southwest resource model; they are not detailed in this report. The main reason for their exclusion is that a third hole (SW-xx-03) has now been drilled on most Southwest drill panels in order to document the basal portion of the Southwest deposit that previously relied upon the trenches. The drill sampling is more robust and less prone to possible sampling bias or surface weathering effects.

Southwest holes are almost exclusively dipping 45° to the N310, with three holes separated by 50-100 m (most generally 100 m) on each of the 26 drill panels. The drill panels themselves (from 100 to 2600) are nominally separated by 100 m (see Figure 14-1 to Figure 14-5).

For Southwest, there are up to 4,877 assay intervals totalling 14,592 m. Of these intervals, 4,835 are 3 m long (the remaining 42 are all less than 3 m). Of the 4,877 intervals, 533 were taken from the ten trenches used in the 2010 study, with the balance of 4,344 representing data taken from the core holes.

Statistics on available assay data in those 4,344 intervals are found in Table 14-2. Most intervals (94%) have a %Satmagan. There is also a large proportion of intervals (85.5%) with WRA results including %Fe_2O_3, %TiO_2, %V_2O_5, %Al_2O_3 and %P_2O_5. Sulphur (%S) is available in about the same proportion of intervals (74%). Density (from pycnometer) is available in 12% of samples.

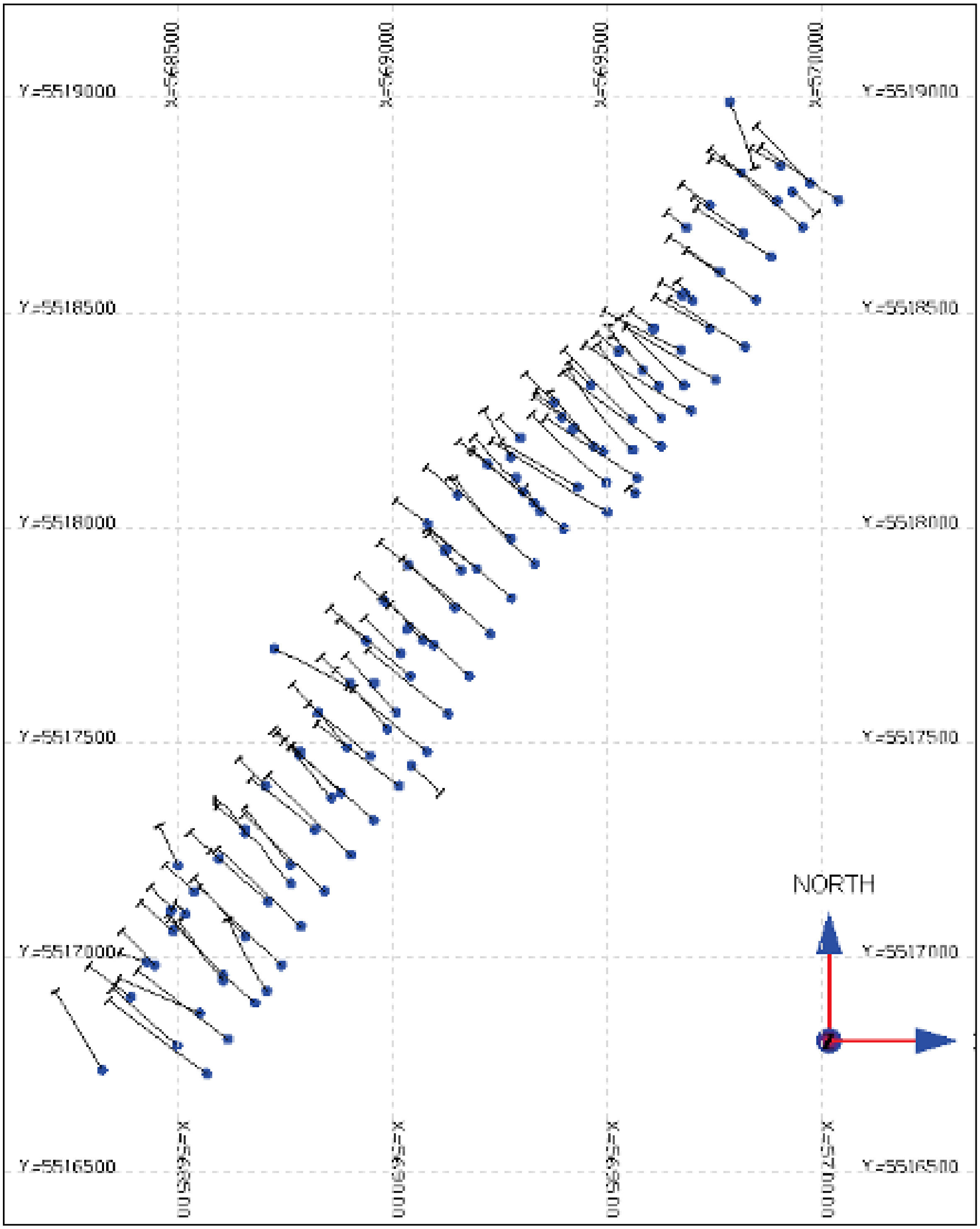


Figure 14-1: Plan view of Southwest Holes

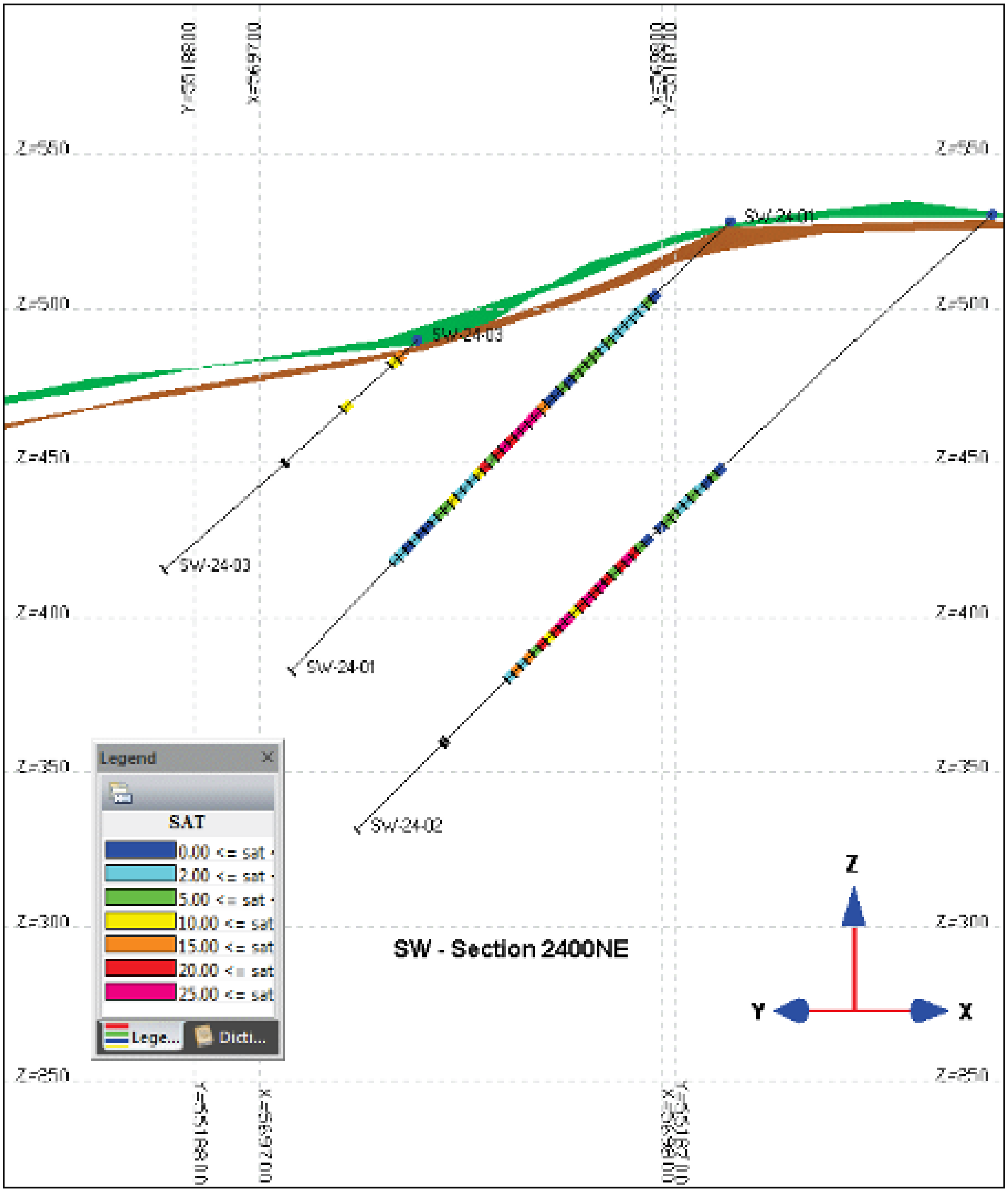


Figure 14-2: Representative Southwest drill panels with DH assay data

Top surfaces: green = topography, brown = overburden/bedrock contact.

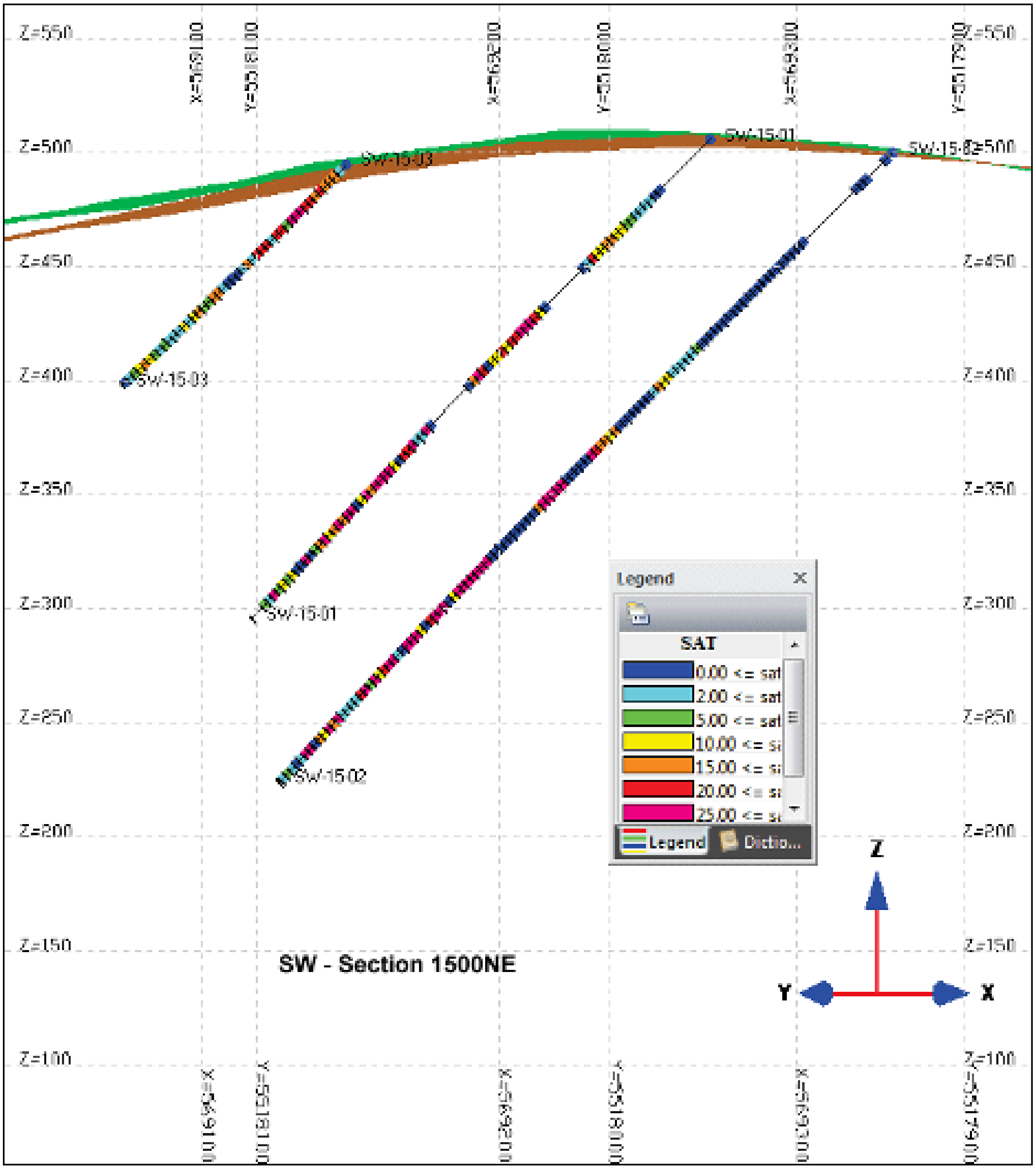


Figure 14-3: Representative Southwest drill panels with DH assay data

Top surfaces: green = topography, brown = overburden/bedrock contact.

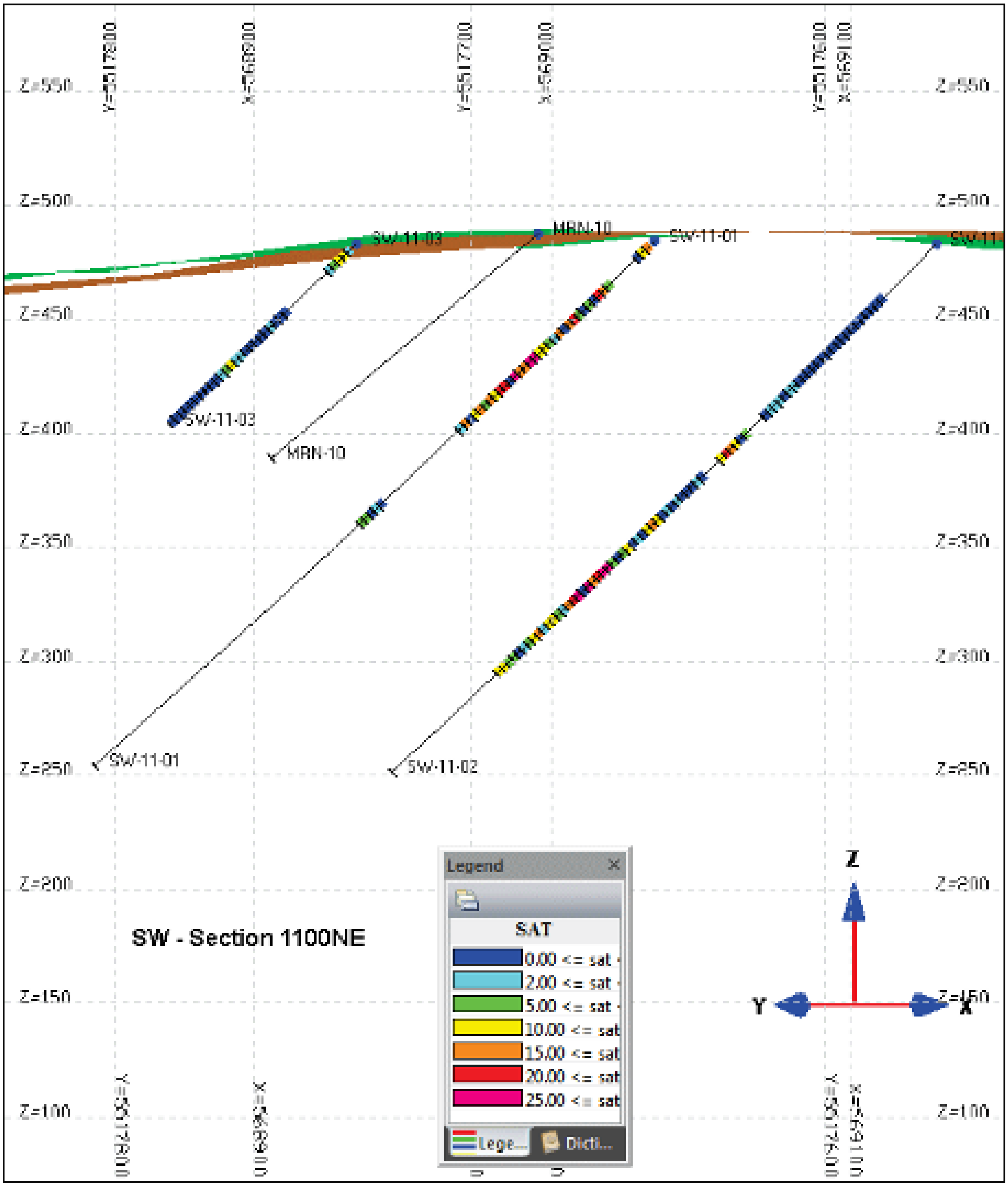


Figure 14-4: Representative Southwest drill panels with DH assay data

Top surfaces: green = topography, brown = overburden/bedrock contact.

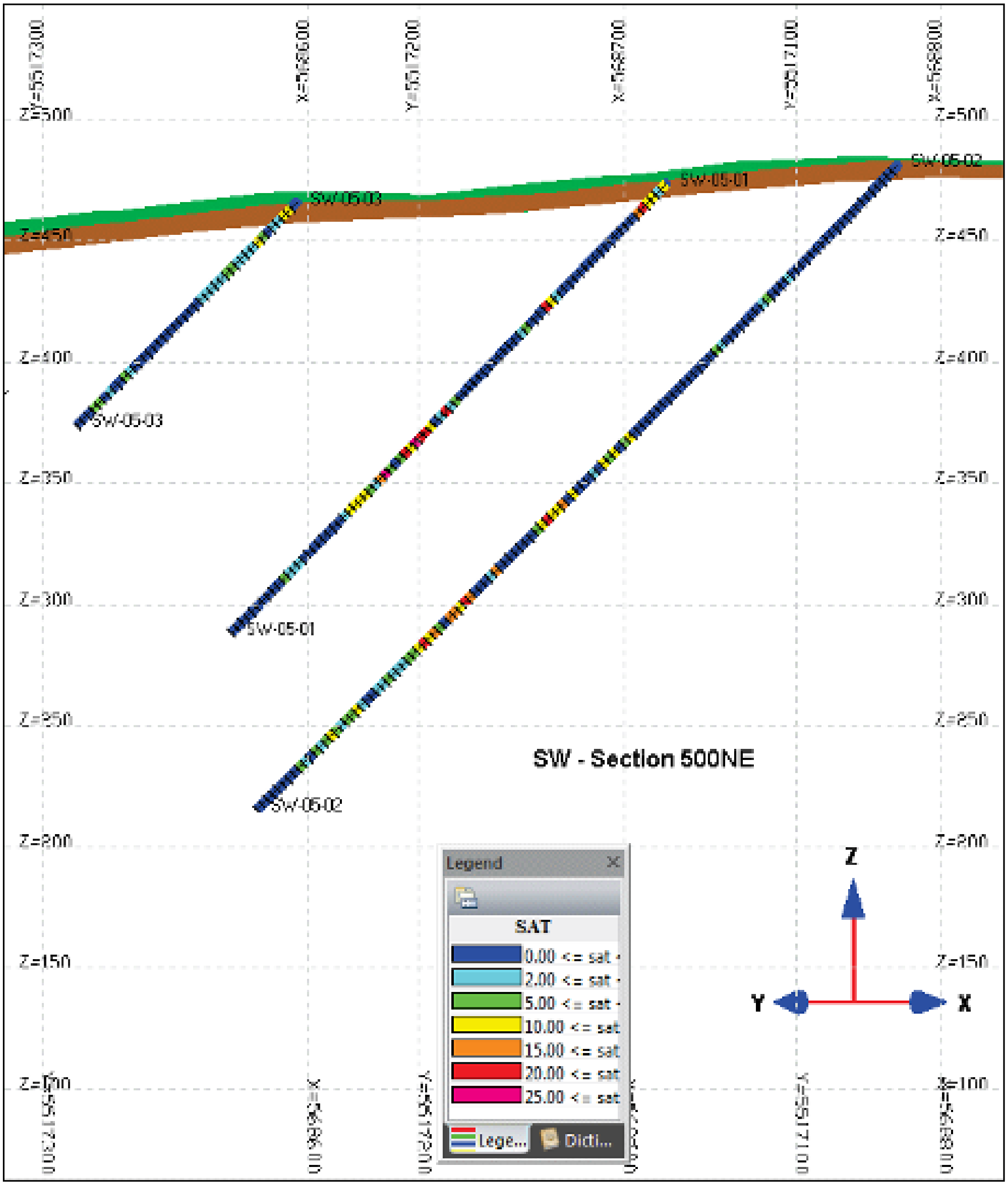


Figure 14-5: Representative Southwest drill panels with DH assay data

Top surfaces: green = topography, brown = overburden/bedrock contact.

Table 14-2: Statistics of assay interval data in the Southwest drillholes

Variable	Number	Min.	Max.	Average	%CV
SW core holes					
Length (m)	4,344	0.5	4.5	3.0	5.4
%Satmagan	**4,092**	**0**	**42.2**	**10.5**	**101.4**
Density (g/cm3)	**697**	**2.68**	**4.63**	**3.41**	**9.8**
$\%SiO_2$	3,712	6.2	73.6	33.9	32.7
$\%Al_2O_3$	**3,713**	**5.6**	**29.4**	**15.1**	**29.2**
$\%Fe_2O_3$	**3,713**	**2.8**	**68.9**	**28.7**	**51.8**
%CaO	3,713	0.9	16.4	7.7	32.4
%MgO	3,713	0.8	16.4	3.8	49.8
$\%Na_2O$	3,713	0.04	9.5	1.9	52.6
$\%K_2O$	3,713	0.0	2.6	0.25	112
$\%Cr_2O_3$	3,713	0	0.33	0.05	105
$\%TiO_2$	**3,713**	**0.24**	**16.5**	**5.0**	**72.1**
%MnO	3,713	0.02	0.58	0.21	37.6
$\%P_2O_5$	**3,713**	**0**	**1**	**0.050**	**135**
%SrO	3,232	0	0.11	0.019	49.1
%BaO	3,232	0	0.08	0.012	142
%LOI	3,636	0	12.6	2.6	61.2
Total	**3,713**	**94.6**	**101.6**	**99.4**	**0.6**
$\%V_2O_5$[(1)]	**3,712**	**0.01**	**0.96**	**0.28**	**83.4**
%FeO	464	1.6	33.1	15.7	46.1
%S	**3,227**	**0.0**	**6.41[(2)]**	**0.17**	**103.5**

The %CV = coefficient of variation, i.e., standard deviation divided by average [(1)]. Some vanadium data were provided as %V. They have been converted to $\%V_2O_5$ by being multiplied by 1.7852 [(2)]. Obvious outlier: next highest %S is 1.8.

14.2.2. The Armitage Deposit

The Armitage resource update is based on data from up to 107 holes, i.e., 107-28 = 79 more than in the 2011 study. The new holes are summarized in Table 14-3, and a verbal description is provided below:

- A third hole on the even drill panels, i.e., 400, 600, 800, 1000, 1200, 1400, 1600, 1800, 2000, 2200, 2400, 2600 and 2800 (total: 13 new holes: AE-04-03…AE-28-03);

- Three new holes on the odd drill panels 300, 500, 700, 900, 1100, 1300, 1500, 1700, 1900, 2100 and 2700 and drill panel 200, plus two new holes on drill panels 2300 and 2500 (total 40 new holes: AE-02-01…AE-27-03);
- Vertical HQ metallurgical bulk sample holes on drill panels 400, 600 (4), 800 (2), 1000 (2), 1200, 1400, 1600, 1800 (2), 2000, 2200 (2), 2400, 2600 (2) and 2800 (21 new holes: AE-BK-04….AE-BK-28);
- Miscellaneous holes (five new holes MKBY-11, 12, 13 and 14, plus CD-6).

There are lithologic data along all new holes, except (d). There are assay data (but not continuous) along all new holes, except (c) and (d).

Armitage holes are typically dipping 45° to the N340, with three holes (except drill panels 2300 and 2500 with only two holes) separated by approximately 100 m (sometimes with as little as 50 m spacing where deviation from the drill design was strongest) on each of the 27 drill panels (from 200 to 2800). The drill panels are separated by 100 m (Figure 14-3 and Figure 14-4) as well. This grid design is identical to the Southwest, with the exception of a 50 m offset for holes or geotechnical holes. There are two holes drilled 400 m away on the isolated drill panel 3200, that are also in the dataset, but have not been included in the current resource model.

Table 14-3: All drilling in the Armitage Deposit

Drillhole	Drillhole	Drillhole	Met Hole	Met Hole	Met Hole	Met Hole
AE-02-01	**AE-02-02**	**AE-02-03**				
AE-03-01	**AE-03-02**	**AE-03-03**				
AE-04-01	AE-04-02	**AE-04-03**	**AE-BK-04**			
AE-05-01	**AE-05-02**	**AE-05-03**				
AE-06-01	AE-06-02	**AE-06-03**	**AE-BK-06**	**AE-BK-06B**	**AE-BK-06C**	**AE-BK-06D**
AE-07-01	**AE-07-02**	**AE-07-03**				
AE-08-01	AE-08-02	**AE-08-03**	**AE-BK-08**	**AE-BK-08B**		
AE-09-01	**AE-09-02**	**AE-09-03**				
AE-10-01	AE-10-02	**AE-10-03**	**AE-BK-10**	**AE-BK-10B**		
AE-11-01	**AE-11-02**	**AE-11-03**				
AE-12-01	AE-12-02	**AE-12-03**	**AE-BK-12**			
AE-13-01	**AE-13-02**	**AE-13-03**				
AE-14-01	AE-14-02	**AE-14-03**	**AE-BK-14**			
AE-15-01	**AE-15-02**	**AE-15-03**				
AE-16-01	AE-16-02	**AE-16-03**	**AE-BK-16**			

Drillhole	Drillhole	Drillhole	Met Hole	Met Hole	Met Hole	Met Hole
AE-17-01	**AE-17-02**	**AE-17-03**				
AE-18-01	AE-18-02	**AE-18-03**	**AE-BK-18**	**AE-BK-18B**		
AE-19-01	**AE-19-02**	**AE-19-03**				
AE-20-01	AE-20-02	**AE-20-03**	**AE-BK-20**			
AE-21-01	**AE-21-02**	**AE-21-03**				
AE-22-01	AE-22-02	**AE-22-03**	**AE-BK-22**	**AE-BK-22B**		
AE-23-01	**AE-23-02**					
AE-24-01	AE-24-02	**AE-24-03**	**AE-BK-24**			
AE-25-01	**AE-25-02**					
AE-26-01	AE-26-02	**AE-26-03**	**AE-BK-26**	**AE-BK-26B**		
AE-27-01	**AE-27-02**	**AE-27-03**				
AE-28-01	AE-28-02	**AE-28-03**	**AE-BK-28**			
AE-32-01	AE-32-02					
GeoTech						
Misc.	**MKBY-11**	**MKBY-12**	**MKBY-13**	**MKBY-14**	**CD-06**	

Black font represents information used in the 2011 resource estimate, **Red Bold** represents the drilling data added since 2011 and used for the current resource model, and **Blue Bold** represents the vertical holes drilled for bulk pilot testing. There were no geotechnical holes drilled in the Armitage deposit.

In Armitage, there are up to 3,980 assay intervals, totalling 11,935 m. All except 22 are 3 m long. Statistics on assay data are found in Table 14-4. In this case, assay coverage is almost perfect, with about 99% coverage for all WRA elements and Satmagan in the assayed intervals. The proportion of samples, with a measured density from pycnometer (19%), is about the same as in Southwest (16%). %CV = coefficient of variation, i.e., standard deviation divided by average[1]. Some vanadium data were given as %V. They have been converted to %V_2O_5 by being multiplied by 1.7852. The maximum density value of 5.40 g/cm^3 [2] is an obvious outlier. The next highest density has a value of 4.30 g/cm^3.

Table 14-4: Statistics of assay interval data in the Armitage drillholes

Variable	Number	Min.	Max.	Average	%CV
Armitage core holes					
Length (m)	3,980	1.5	4.0	3.0	2.2
%Satmagan	3,932	0	39.8	9.6	87.7
Density (g/cm^3)	776	2.73	5.40(2)	3.34	8.4
%SiO_2	3,943	11.8	68.1	33.7	26.8
%Al_2O_3	3,943	6.6	29.0	15.2	23.0
%Fe_2O_3	3,943	1.9	59.5	28.3	41.3
%CaO	3,943	2.3	19.9	7.8	26.0
%MgO	3,943	0.5	12.1	3.6	47.7
%Na_2O	3,943	0.06	7.7	1.9	48.9
%K_2O	3,943	0.0	3.9	0.16	163
%Cr_2O_3	3,943	0	0.22	0.04	97.7
%TiO_2	3,943	0.13	13.2	5.0	58.4
%MnO	3,943	0.01	0.40	0.20	35.4
%P_2O_5	3,713	0.007	0.33	0.033	103
%SrO	3,943	0	0.10	0.016	61.8
%BaO	3,943	0	0.07	0.014	110
%LOI	3,943	0.12	13.9	3.2	59.9
Total	**3,943**	**96.0**	**100.4**	**99.2**	**0.5**
%V_2O_5(1)	3,943	0.01	0.94	0.26	71.3
%FeO	0	-	-	-	-
%S	3,944	0.0	1.8	0.26	74.9

14.2.3. PK Hyperspectral Data

Another set of data, which has been used in the derivation of the new resource block models, represents hyperspectral scan values along core holes by Photonic Knowledge (PK) (see Section 9.4.1). This automatic logging technique provides a continuous log of the concentration of the main mineral assemblages (i.e., magnetite, ilmenite, hemo-ilmenite, hornblende, "green waste" and "host waste") along the drill core.

With data composited into 1 m down hole intervals, PK represents the most continuous data coverage of any type on the project. This data was used to help constrain the structural domains in the models, and to pad mineralized intervals where there were missing assay data (mostly in the Southwest deposit) (see Section 14.3).

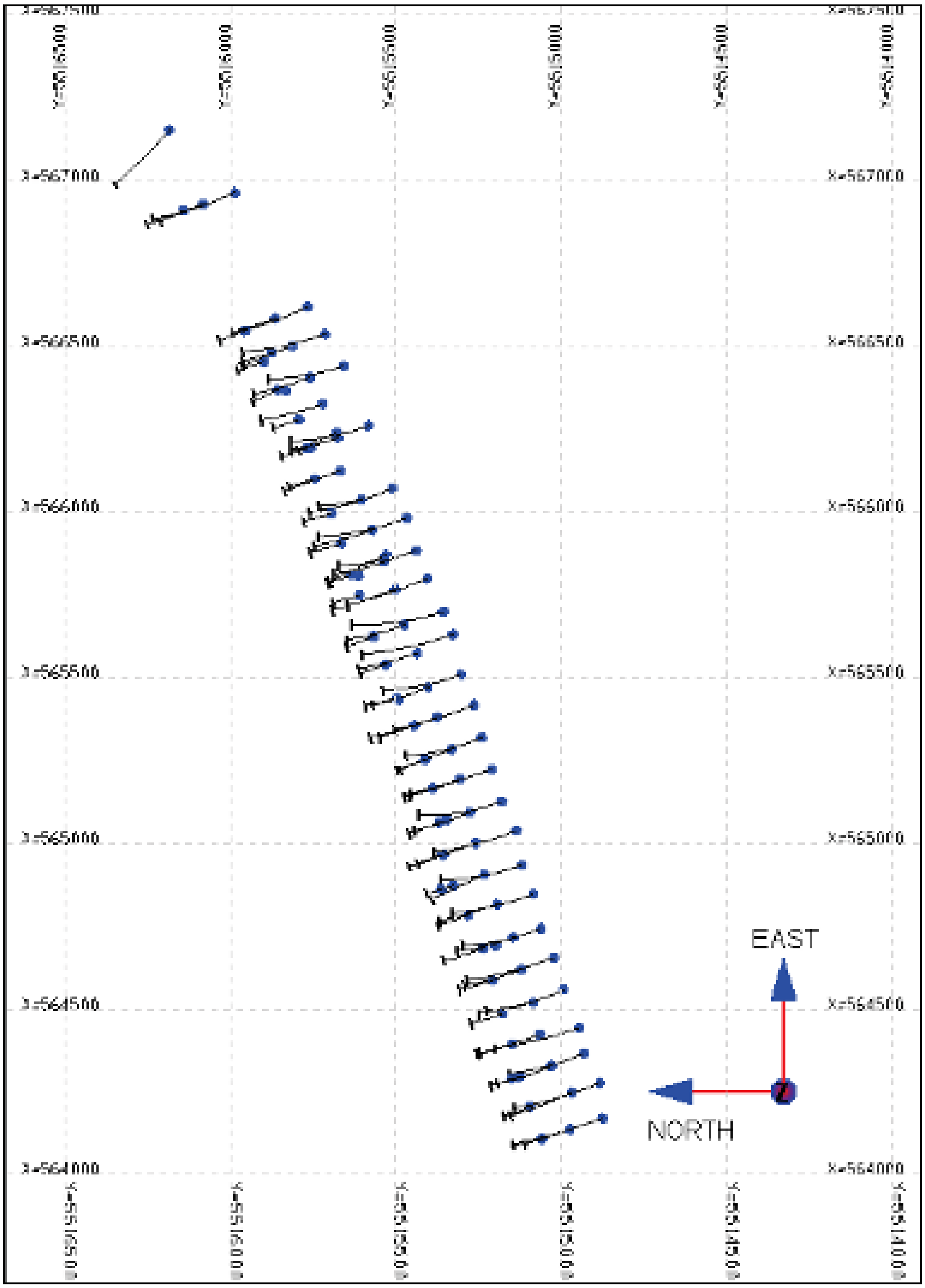


Figure 14-6: Plan view of Armitage holes where each square is 500 m on a side

Note that the plan view is rotated, with north pointing to the left-hand side of the drawing.

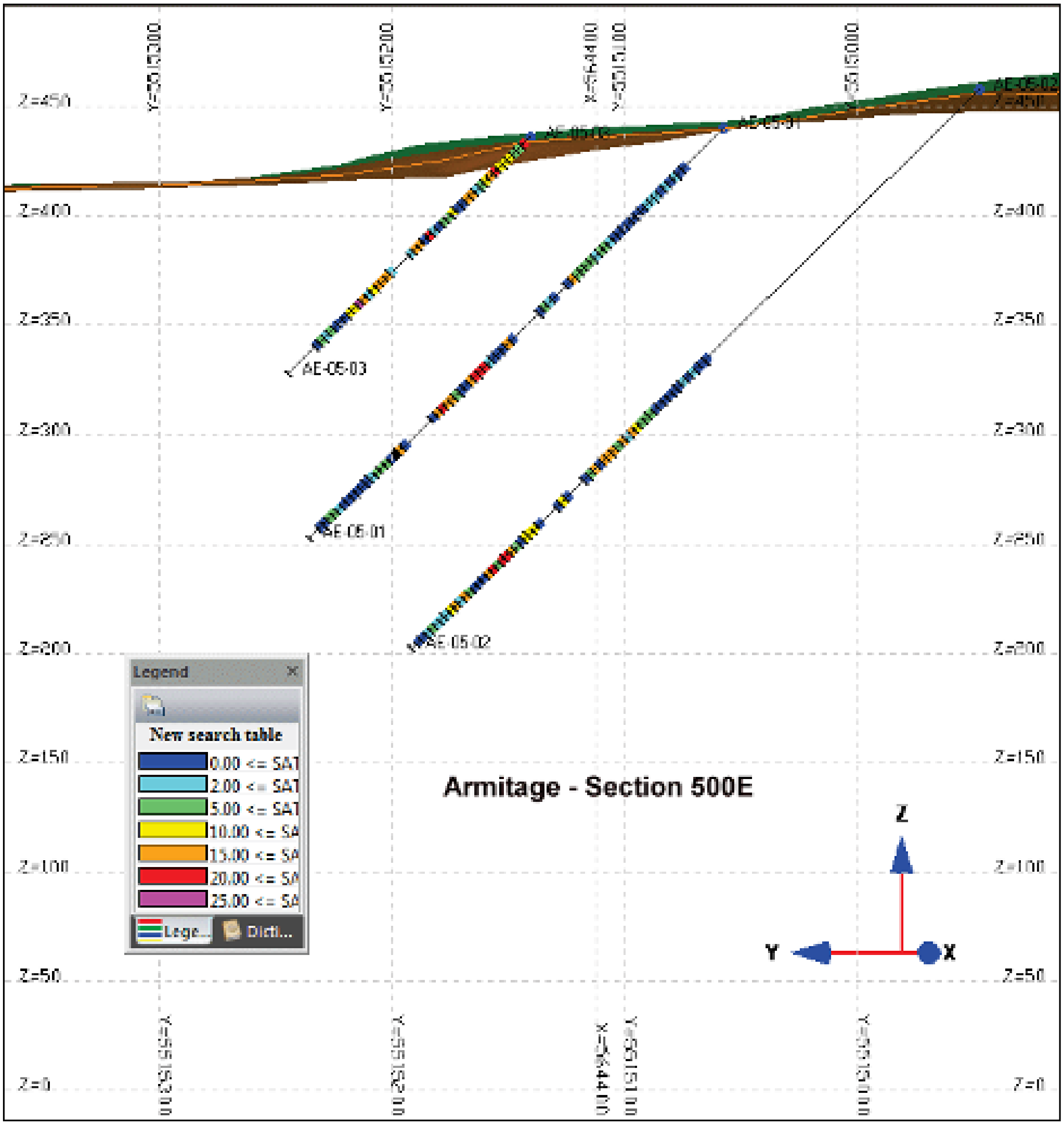


Figure 14-7: Representative Armitage cross-sections with DH assay data, where each horizontal line represents 50 m in elevation; section is facing east

Top surfaces: green = topography, brown = overburden/bedrock contact.

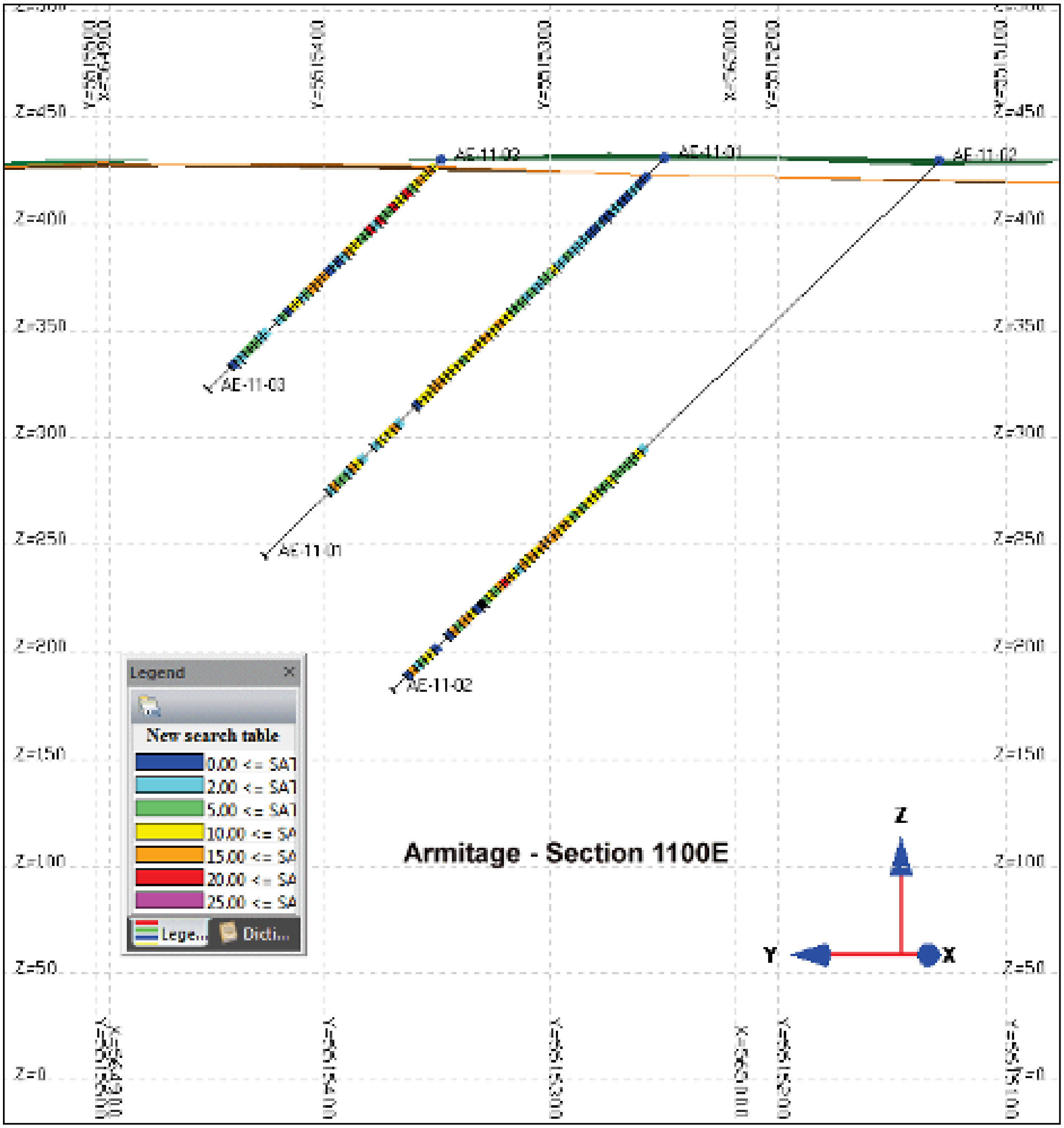


Figure 14-8: Representative Armitage cross-sections with DH assay data, where each horizontal line represents 50 m in elevation; section is facing east

Top surfaces: green = topography, brown = overburden/bedrock contact.

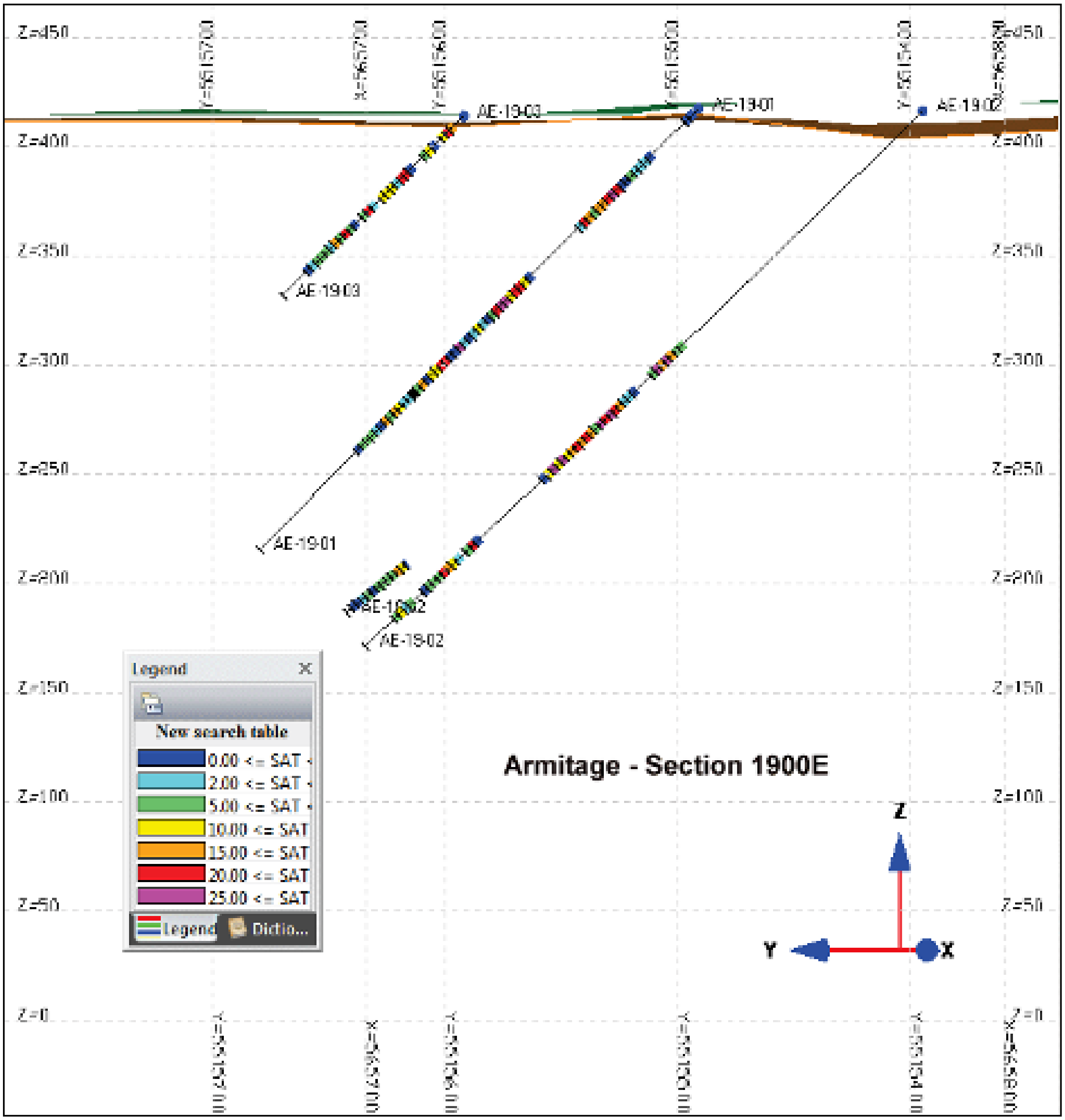


Figure 14-9: Representative Armitage cross-sections with DH assay data, where each horizontal line represents 50 m in elevation; section is facing east

Top surfaces: green = topography, brown = overburden/bedrock contact.

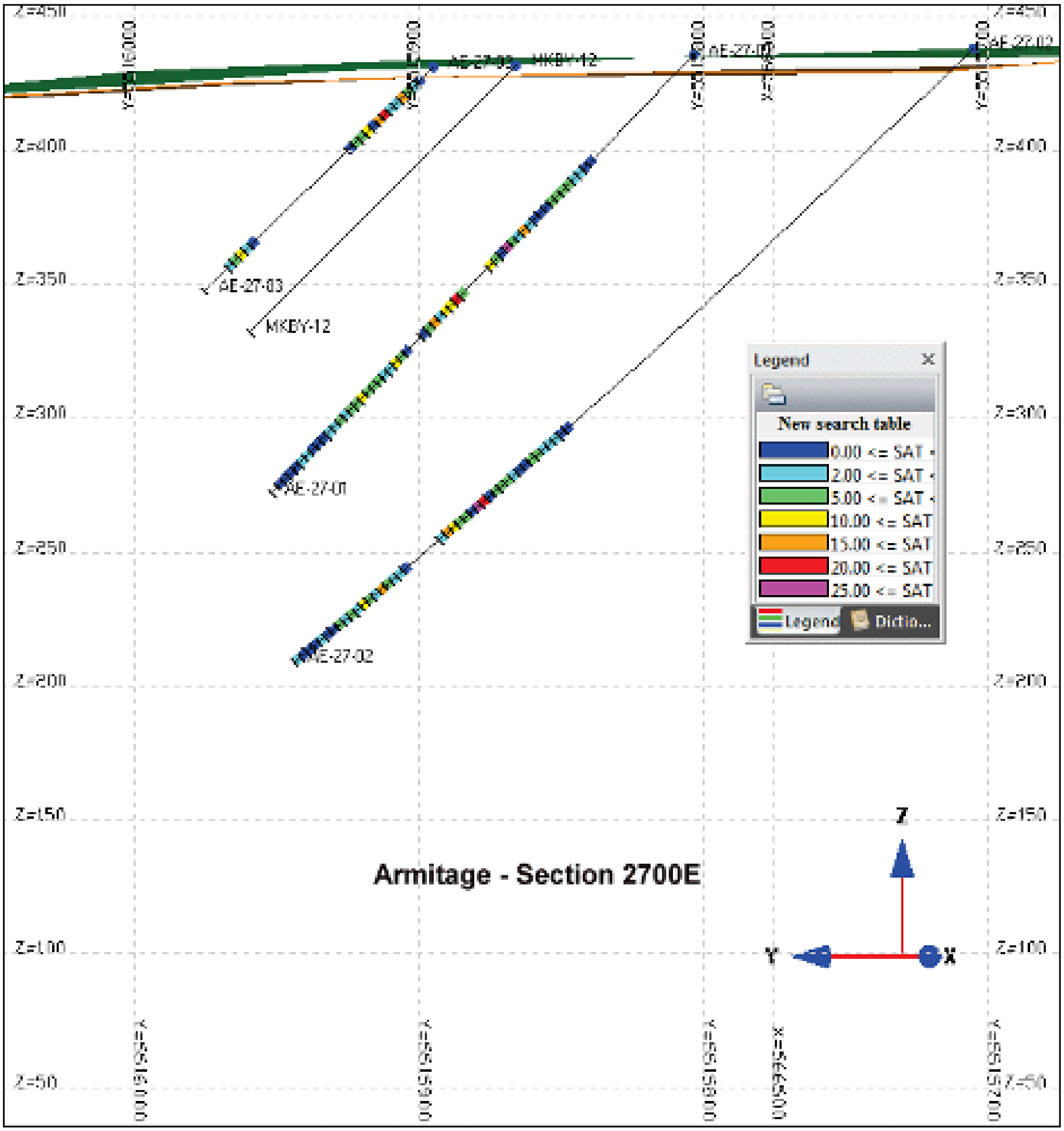


Figure 14-10: Representative Armitage cross-sections with DH assay data, where each horizontal line represents 50 m in elevation; section is facing east

Top surfaces: green = topography, brown = overburden/bedrock contact.

14.2.4. Comparison of Southwest and Armitage

Comparison of statistics for assay data in Southwest and Armitage shows fairly similar averages for major elements, i.e., Fe_2O_3, SiO_2, Al_2O_3, TiO_2, CaO, MgO, Na_2O and MnO. Average Satmagan and density are slightly less in Armitage: 9.1% vs. 9.7% and 3.34 vs. 3.43 g/cm^3, respectively. There are greater relative differences displayed between average concentrations of some of the trace elements; higher S in Armitage (0.26% vs. 0.17%) and higher phosphorus in Southwest (0.050% vs. 0.033%). The PK dataset for Southwest has a total of 20,034 data composites, and the Armitage has a total of 21,608 interval data composites, providing essentially 100% coverage of the resource drillholes.

14.3. Mineralized Envelopes

In the 2010 Southwest and 2011 Armitage SGS-Geostat resource models (see Section 6.4), the extent of the mineralization around drillholes was first controlled by a broad envelope connecting outlines around most assay intervals on each drill section, and then, within that envelope, by a mineralization indicator using low cut-offs of 2% Satmagan. The indicator was coded as 0 (less than 2% Satmagan) or 1 (more than 2% Satmagan) in each sample, and then interpolated as a mineralized fraction in each block of the resource model.

For the current (2013) model, mineralization is constrained by 3D solids based on the new stratigraphic interpretation of the mineralization (see Section 7). Mineralization of both Southwest and Armitage deposits is concentrated in four successive units corresponding to successive stages of magmatic differentiation.

The four units have been named by BlackRock geologists, from the bottom of the stratigraphic section to the top: BCS for Basal Chrome Series, MC for Middle Cumulate Horizon, ULS for Upper Layered Series and Titan for Titaniferous Cap. These four units concentrate most of the high %Satmagan, %TiO_2 and %V_2O_5 assay intervals. According to sample data in drillholes, each unit may be further subdivided into several discrete bands, separated by low-grade material.

The mineralized package has been dislocated by cross-fractures. Lateral and vertical displacements are not significant on most of these fractures. They do, however, separate sections of independent dip along the strike extension of each deposit (Figure 14-7). These structural subdivisions have been termed “domains” (for structural domain) by BlackRock geologists.

The up-dip extension of the mineralized polygons (and resulting solids) terminates against modelled bedrock surface (Figure 14-12 to Figure 14-15) The bedrock contact was constructed by connecting the bedrock contacts interpreted on each drill section from the depth of overburden reported in the drillholes on that section. This surface has been updated with all of the new drill information. It has been verified that collar locations from the surveyed collar coordinates fall on the topographic surface.

For the down-dip, as illustrated in Figure 14-12 to Figure 14-15, the unit outlines are interpreted to extend approximately 200 m from the deepest hole on each drill panel. That 200 m distance is also used to extend solids on both extremities along the strike.

Solids are filled with 10 x 5x 7 m blocks on a regular grid, parallel to sections. The 10 x 5 m section is the same as the one used in the previous model, with the 5 m block thickness necessary to accommodate the low thickness of some of the interpreted mineralized bands. The 7 m (10 m before) corresponds to half of the new suggested bench height of 14 m.

The use of the small block size eliminates the need to use mineralized block fractions (the fraction of mineralized solids in blocks). A block is entirely assigned to a unit if its center is inside a solid of that unit. For top blocks overriding the overburden-bedrock surface, the fraction of block below that surface is computed and taken into account in the calculation of resources from the block model.

14.3.1. Southwest Resource Model

The Southwest model has its origin at X=568,508E and Y=5,516,648N, and an X axis with an azimuth of N 220 (rotation of 130° clockwise). The Southwest grid has up to 288 columns from RX=-2830 (section 2830NE) to RX=+50 (section 50SW), 115 rows from RY=-610 to RY=+120 and 80 benches from Z=0 to Z=570.

In Southwest, we have 325,805 blocks within mineralized solids and with a bedrock fraction, for a total of 112.82 Mm3. Based on small displacements and changes in the strike and/or dip of the stratigraphic units, the model has been subdivided into seven domains, numbered 1 to 7, from southwest to northeast, along the 2,600 m strike length of the deposit (Figure 14-5).

The lateral extension from drill panel 100 on the far left-hand side of Figure 14-11 is terminated into a fault at distances varying between 100-200 m, and is extended a full 200 m beyond drill panel 2600, toward the northeast.

Several examples of the cross-sectional interpretations are presented in Figure 14-12 to Figure 14-15 below. Note that the surface projections are terminated in the block model by the bedrock surface. The "Domains" in the sub-labels in the figure refer to segments of the stratigraphic package (BCS+MC+ULS+Titan) with differing orientations that are separated from each other by (interpreted) fracture zones. There are seven domains in SW.

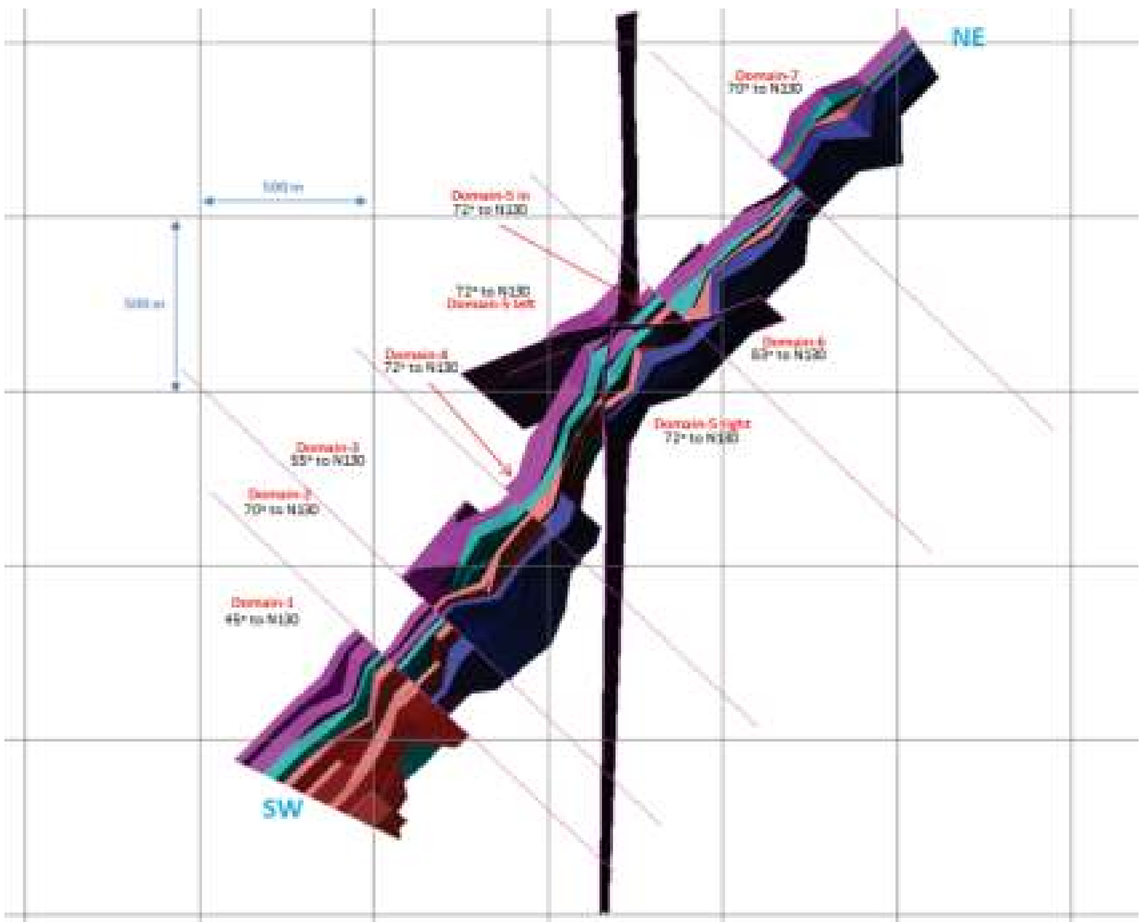


Figure 14-11: Blackrock interpretation of mineralized solids for Southwest

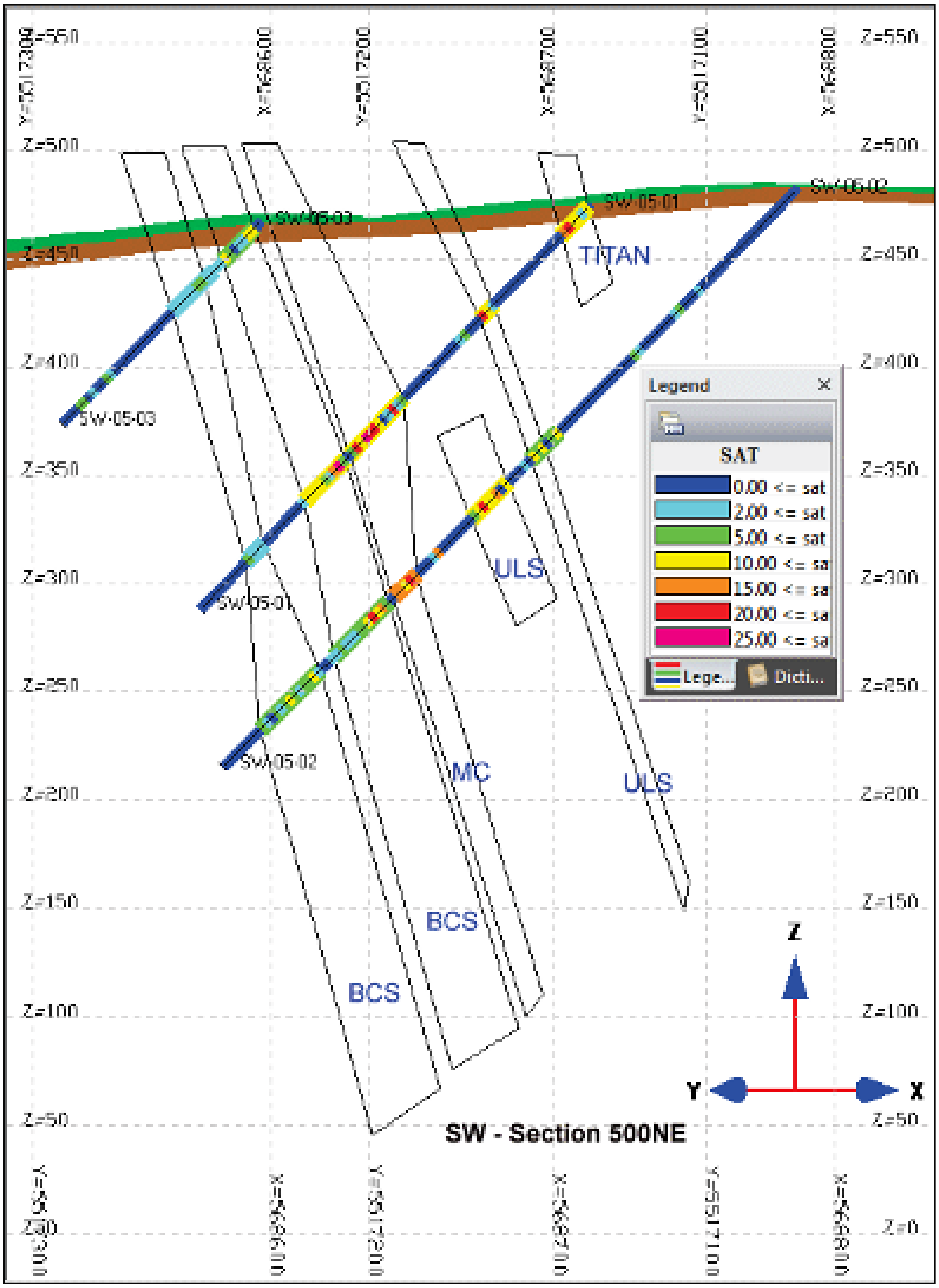


Figure 14-12: Interpretation of mineralized units on Southwest drill panels. Horizontal lines are in 50 m increments. section is viewed looking northeast

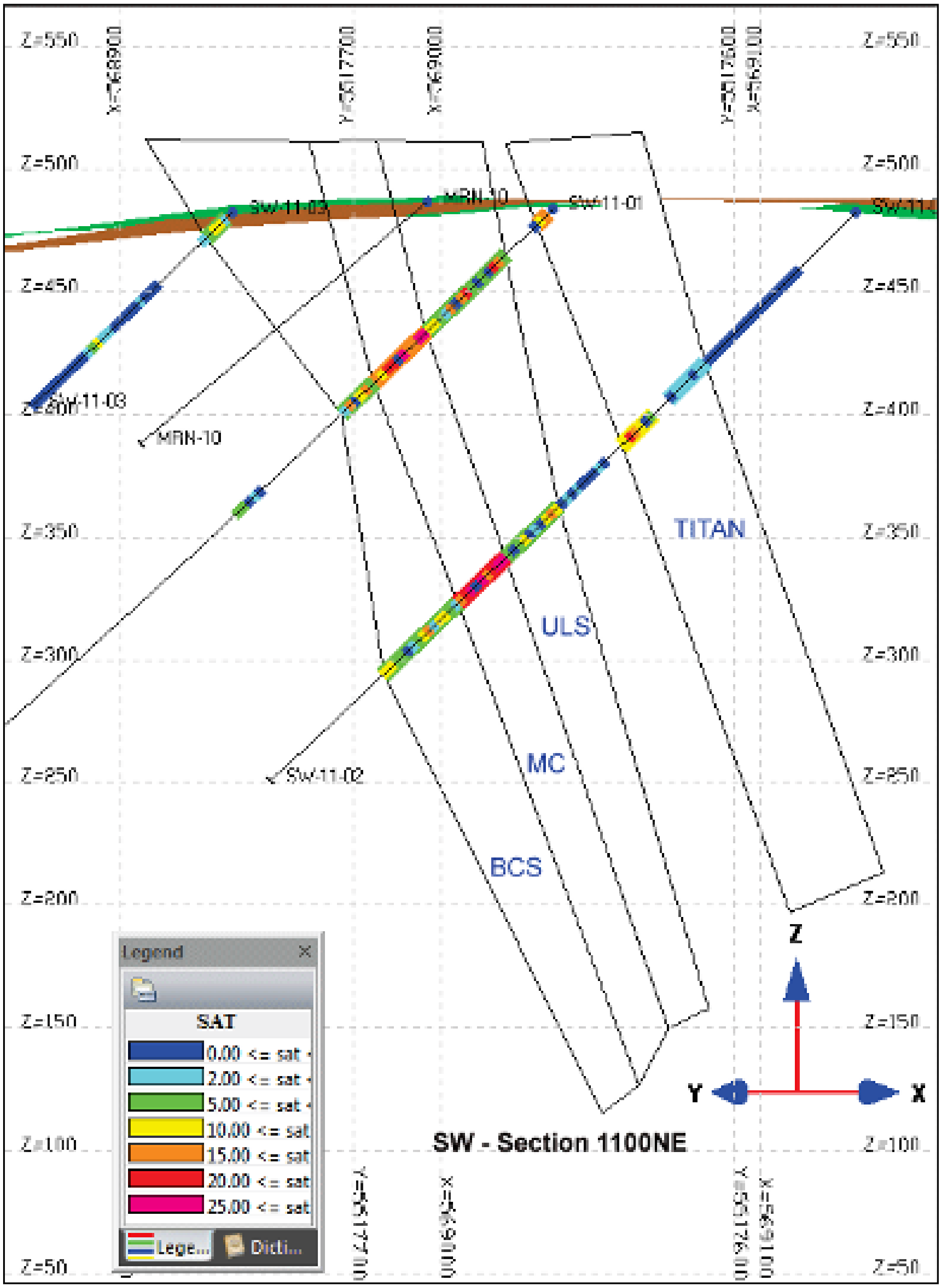


Figure 14-13: Interpretation of mineralized units on Southwest drill panels. Horizontal lines are in 50 m increments. Section is viewed looking northeast

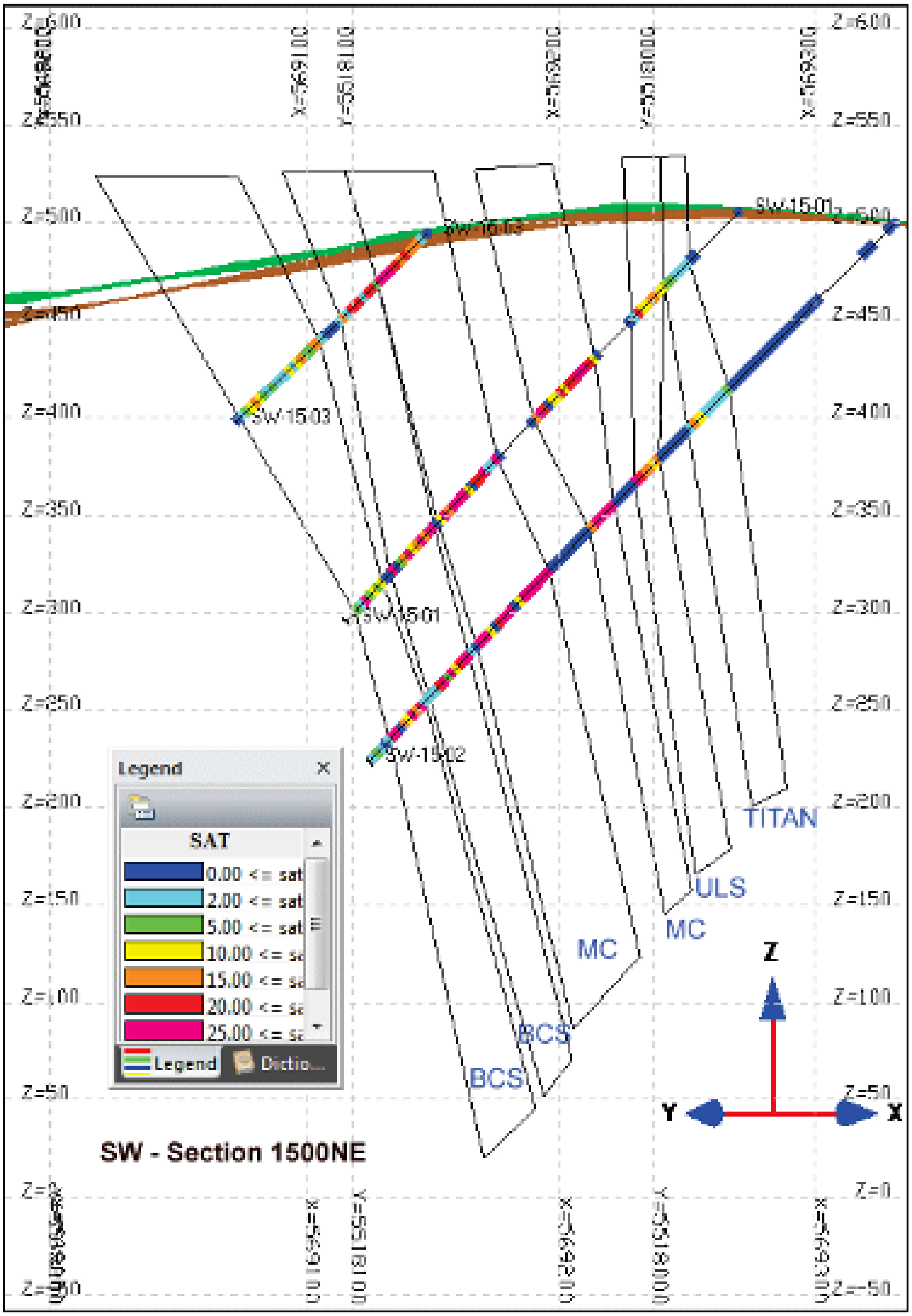


Figure 14-14: Interpretation of mineralized units on Southwest drill panels. Horizontal lines are in 50 m increments. Section is viewed looking northeast

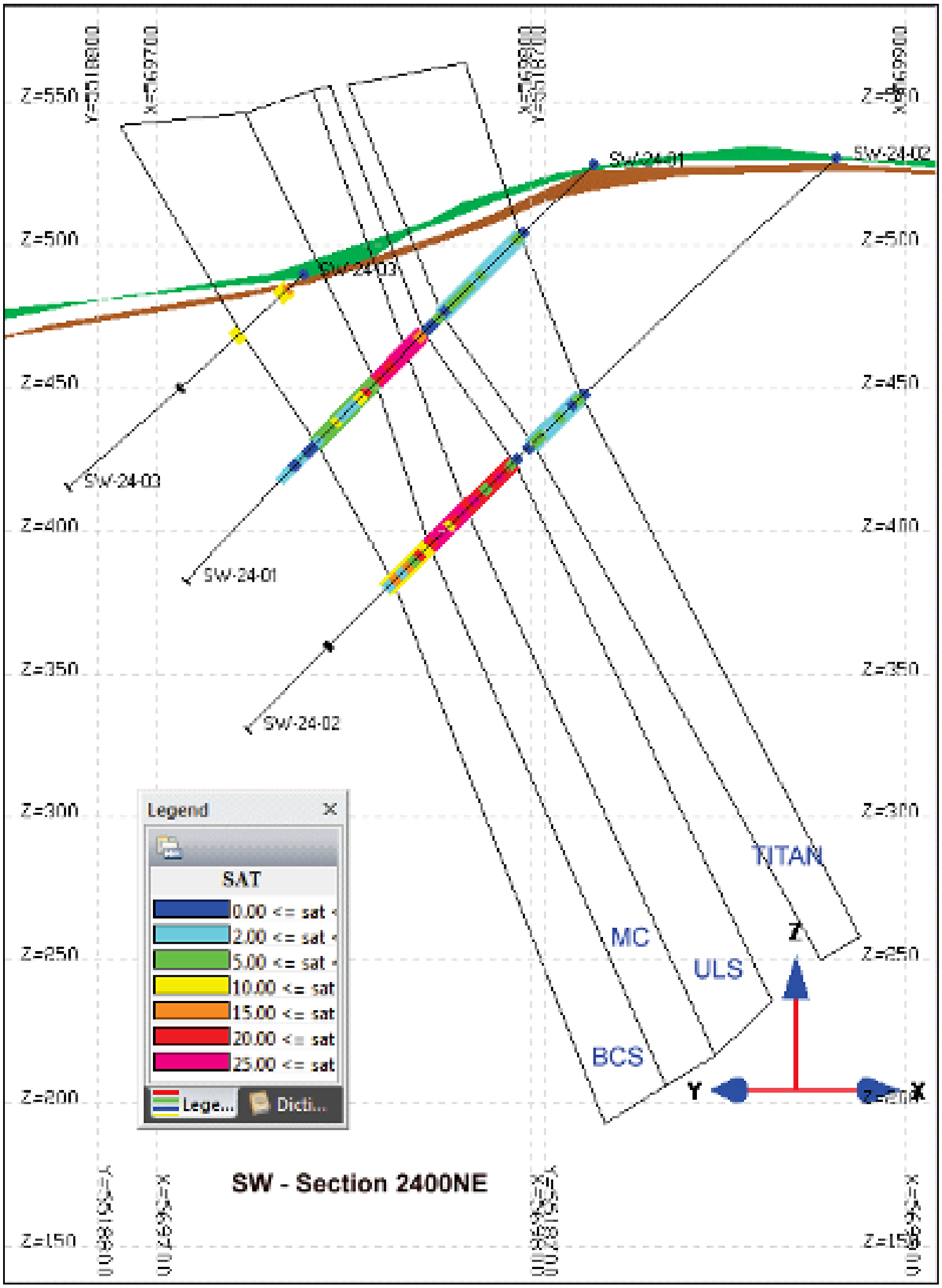


Figure 14-15: Interpretation of mineralized units on Southwest drill panels. Horizontal lines are in 50 m increments. Section is viewed looking northeast

There are also several bench plans illustrating the geologic model presented in Figure 14-7. The domain boundaries are clearly illustrated in these views, as is the geologic continuity of the mineralized stratigraphic units over the 2.9 kilometres represented.

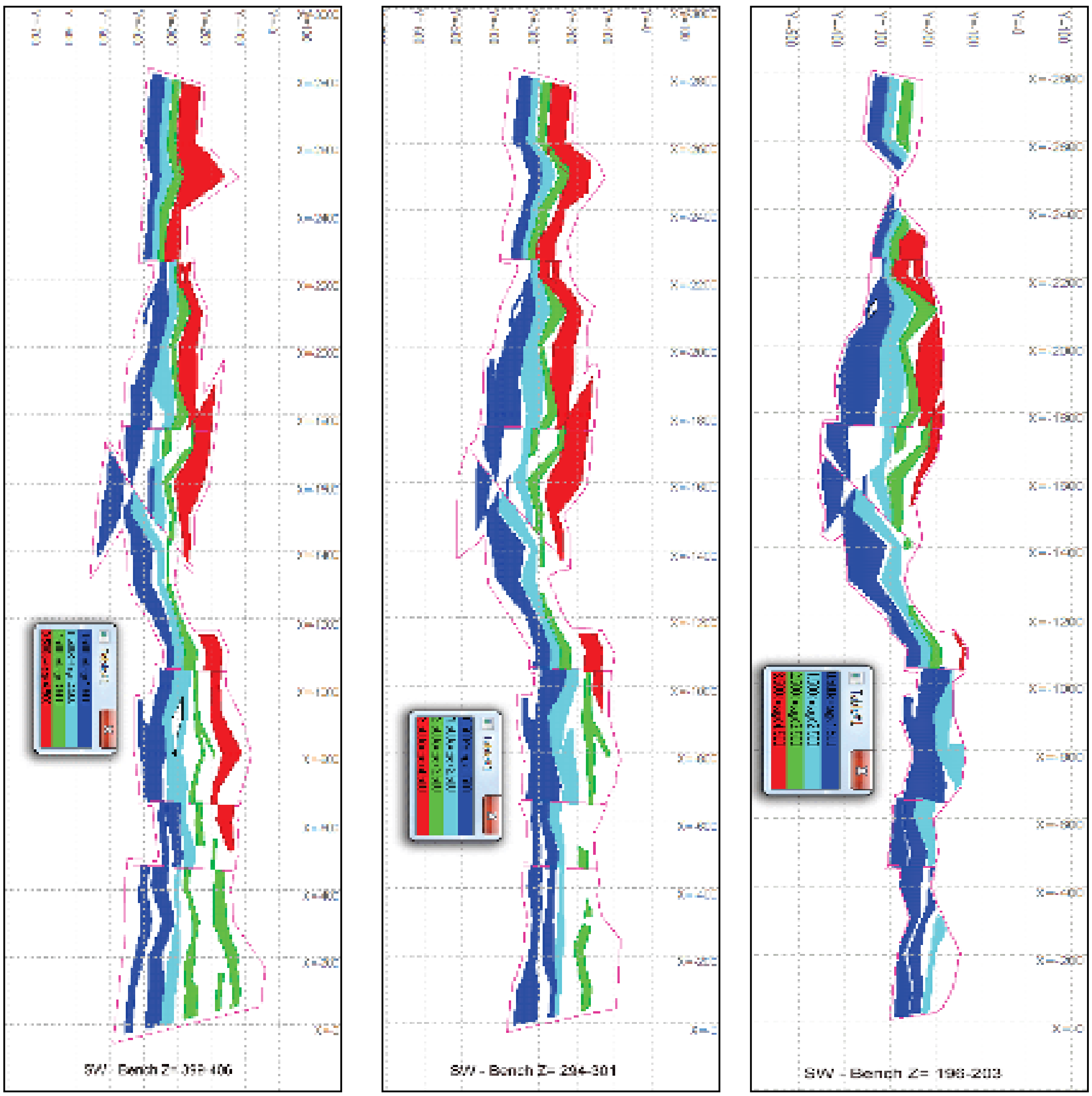


Figure 14-16: Southwest bench plans with blocks in mineralized units

Color according to rock units: dark blue (1) = BCS, light blue (2) = MC, green (3) = ULS and red (4) = Titan.

14.3.2. Armitage Resource Model

Armitage's rotated local reference system has its origin at X=563,984.06E and Y=5,514,793.60N, and an X axis with an azimuth of N-70° (clockwise rotation of 70°). The Armitage grid has up to 295 columns from RX=0 (section 0E) to RX=+2950 (section 2950E), 94 rows from RY=-100 to RY=+370 and 80 benches from Z=-56 to Z=504.

In Armitage, the number of mineralized blocks is 376,169 for a total volume of 130.37 Mm3. Based on small displacements and changes in the strike and/or dip of the stratigraphic units, the model has been subdivided into 13 domains, numbered 1 to 13, from west-southwest to east-northeast, along the 2,700 m strike length of the Armitage deposit (Figure 14-17).

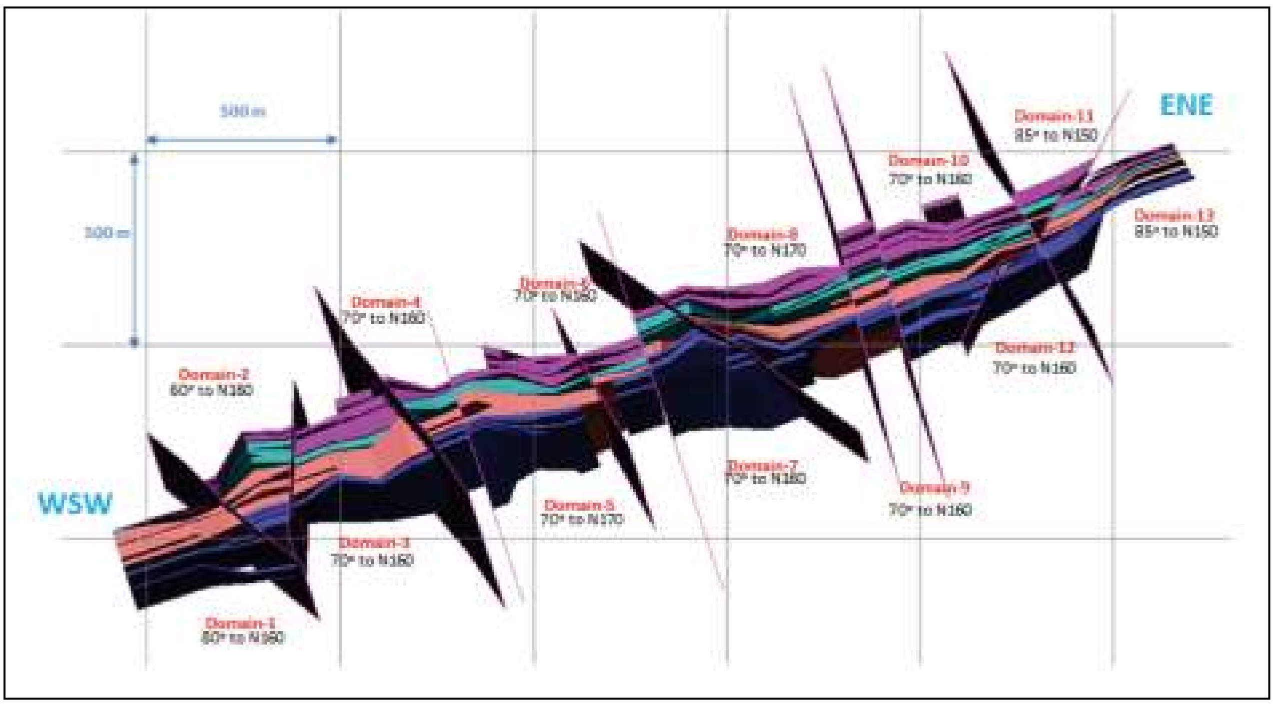


Figure 14-17: Interpretation of mineralized solids for Armitage

"Domains" are sections of the continuous stratigraphic package along strike (BCS+MC+ULS+Titan) that are separated by fractures or faults. The domain boundaries have allowed each subsection to move independently of the adjacent subsections. There are 13 domains in Armitage, with the strike and dip of each subsection annotated in Figure 14-17.

The modelled extension of mineralized stratigraphy is projected 200 m beyond drill panel 200 on the WSW end and drill panel 2800 on the ENE end of Armitage. Note that the last interpreted drill panel included in this resource model is 2800, despite two step-out drillholes on drill panel 3200. No geological modelling was included around this isolated information, as the 400 m distance to the nearest drill panel diminished the quality and value of that interpretation to the overall resource.

Several examples of the cross-sectional interpretations are presented in Figure 14-18 through Figure 14-21 below. Note that the surface projections are terminated in the block model by the bedrock surface.

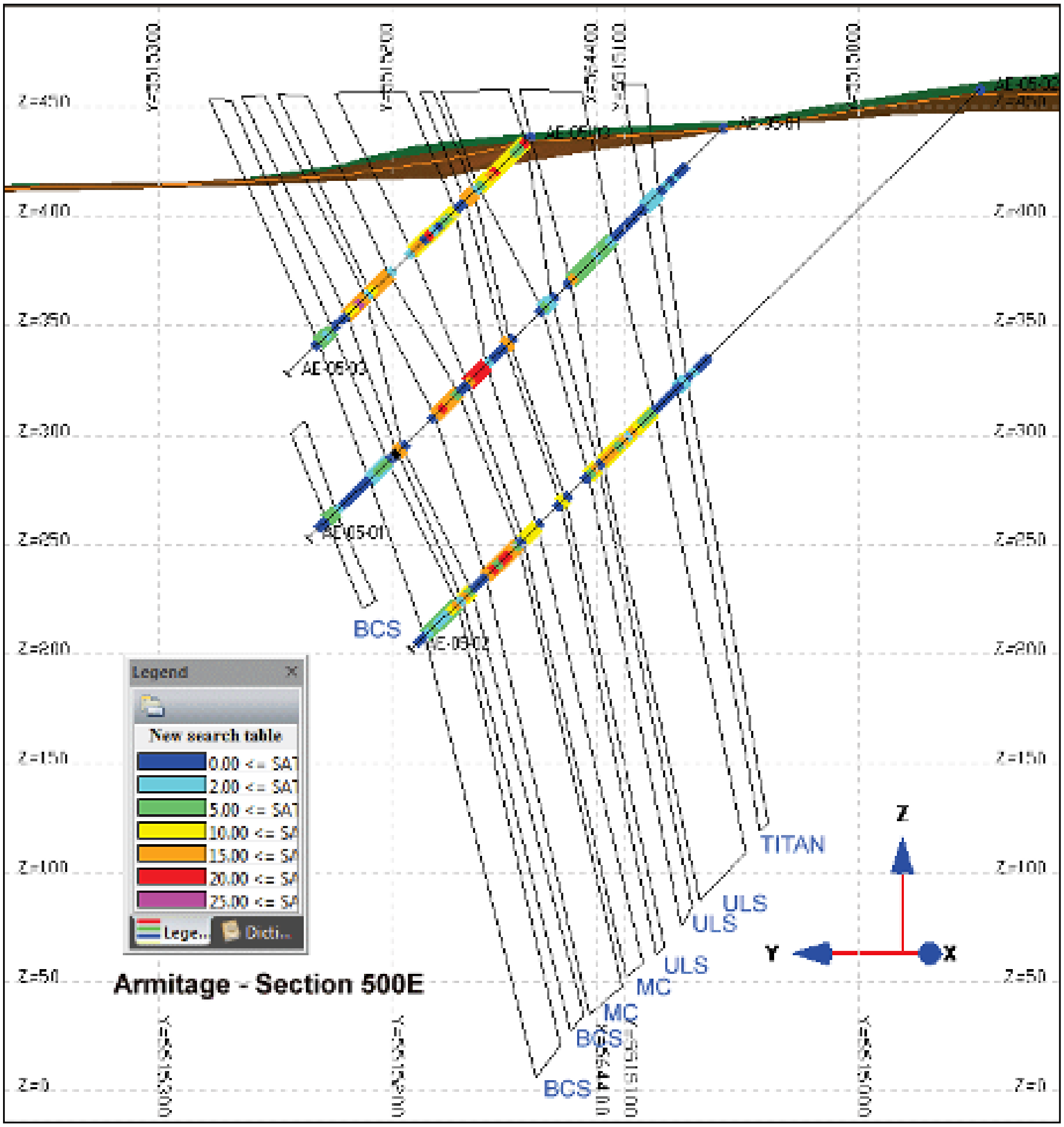


Figure 14-18: Interpretation of mineralized units on Armitage drill panels. Horizontal lines are in 50 m increments. Section is viewed looking northeast

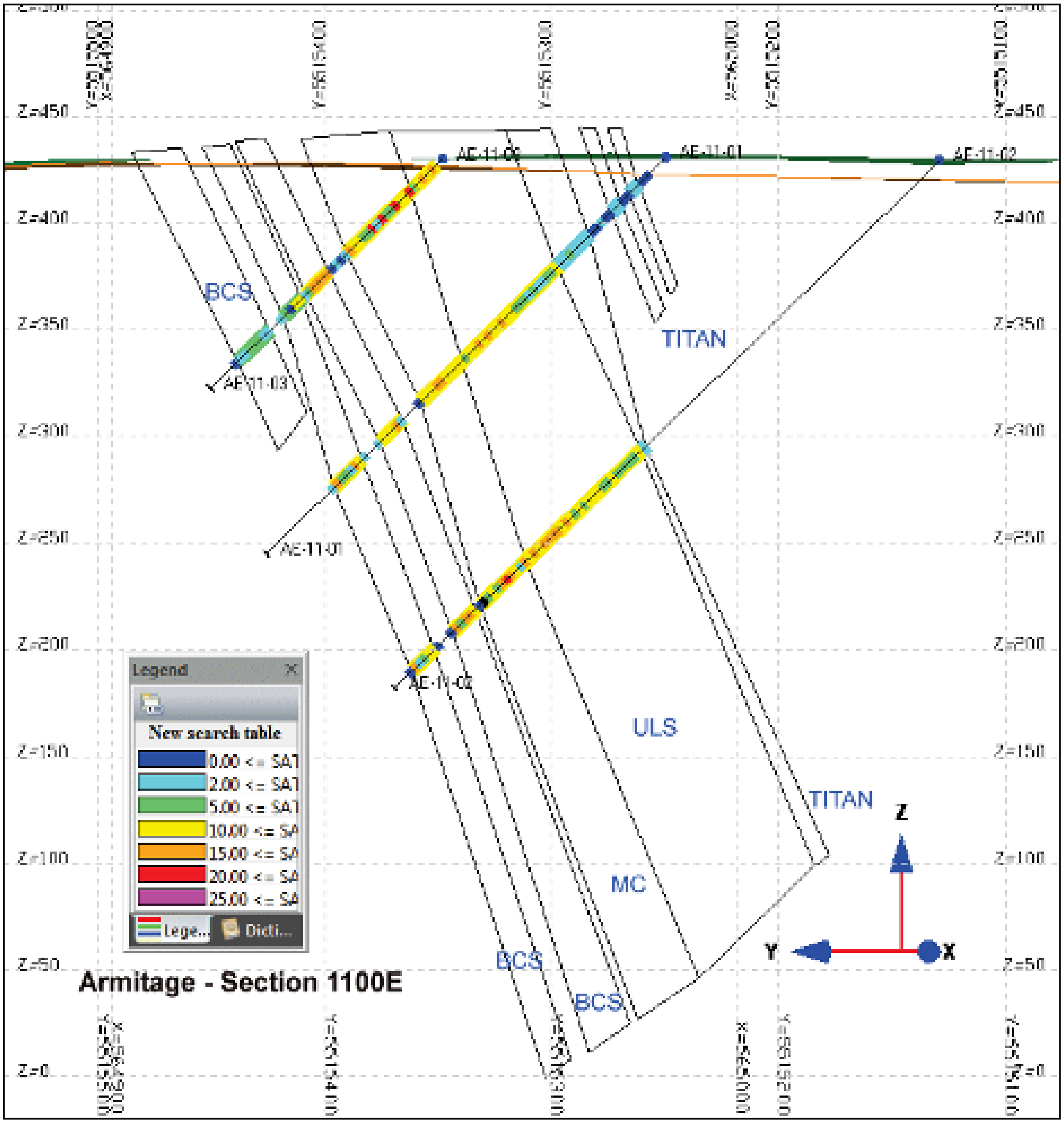


Figure 14-19: Interpretation of mineralized units on Armitage drill panels. Horizontal lines are in 50 m increments. Section is viewed looking northeast

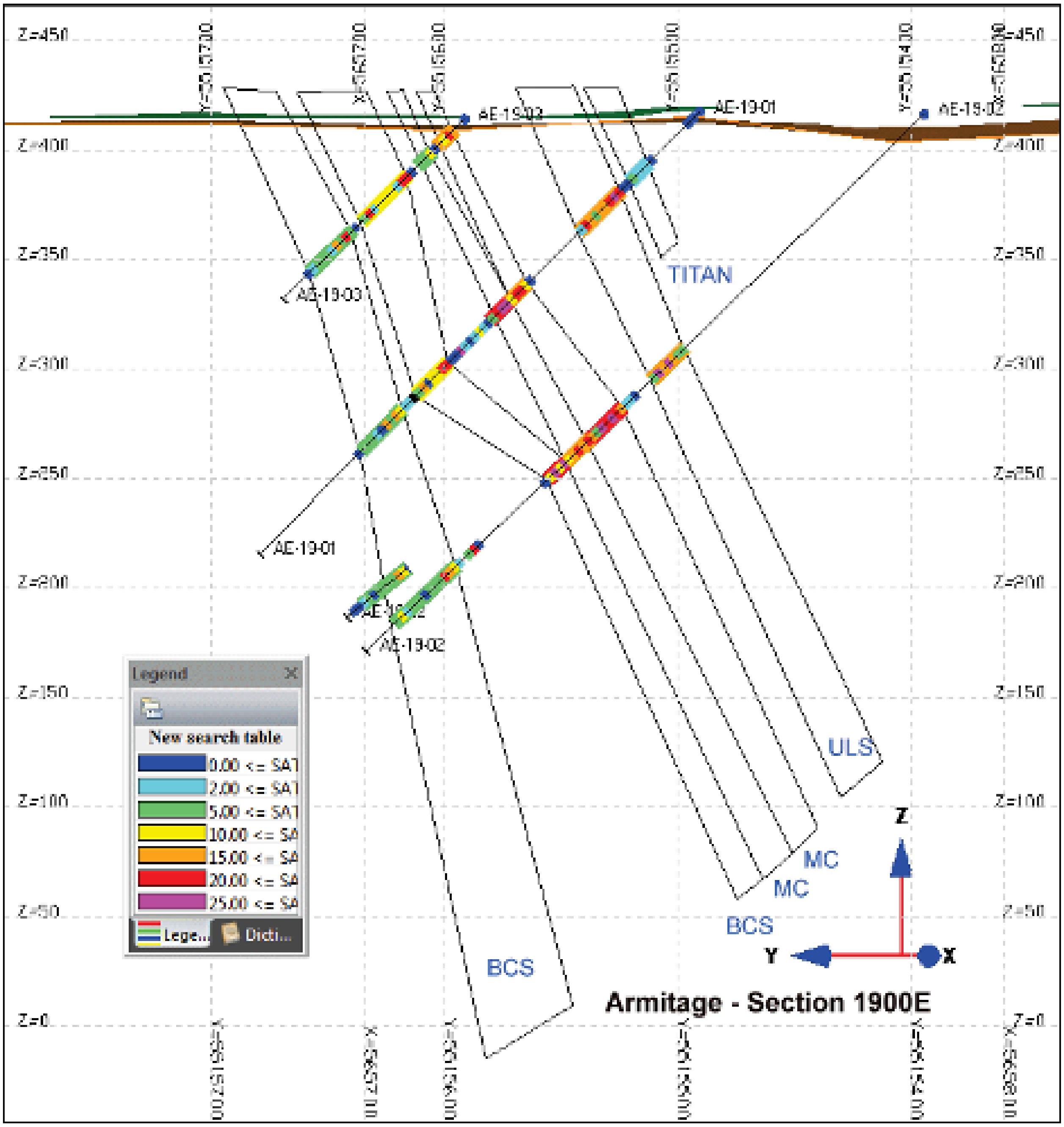


Figure 14-20: Interpretation of mineralized units on Armitage drill panels. Horizontal lines are in 50 m increments. Section is viewed looking northeast

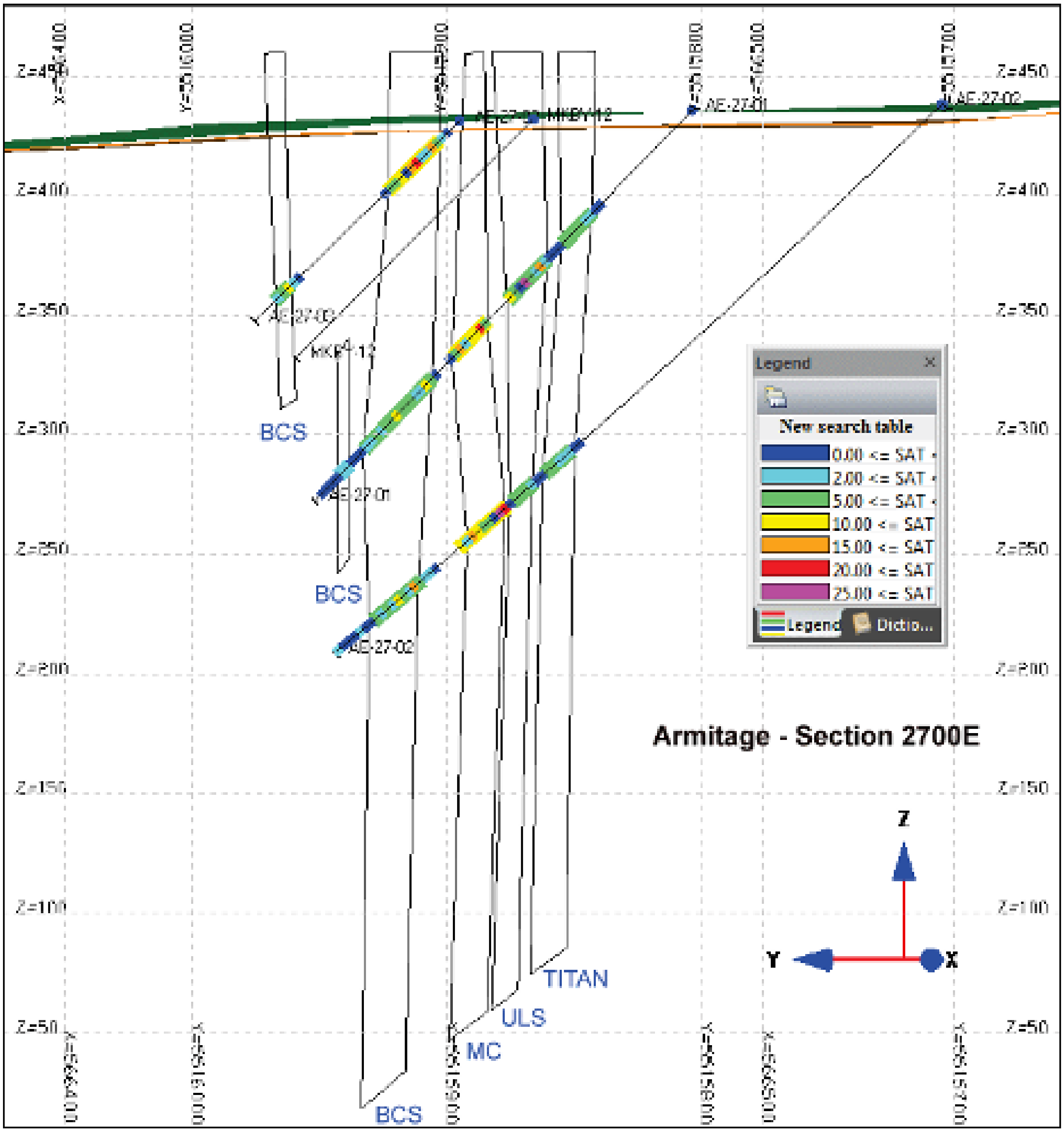


Figure 14-21: Interpretation of mineralized units on Armitage drill panels. Horizontal lines are in 50 m increments. Section is viewed looking northeast

There are also several bench plans illustrating the geologic model presented in Figure 14-10. The domain boundaries are clearly illustrated in these views, as is the geologic continuity of the mineralized stratigraphic units over the 3.0 kilometres represented.

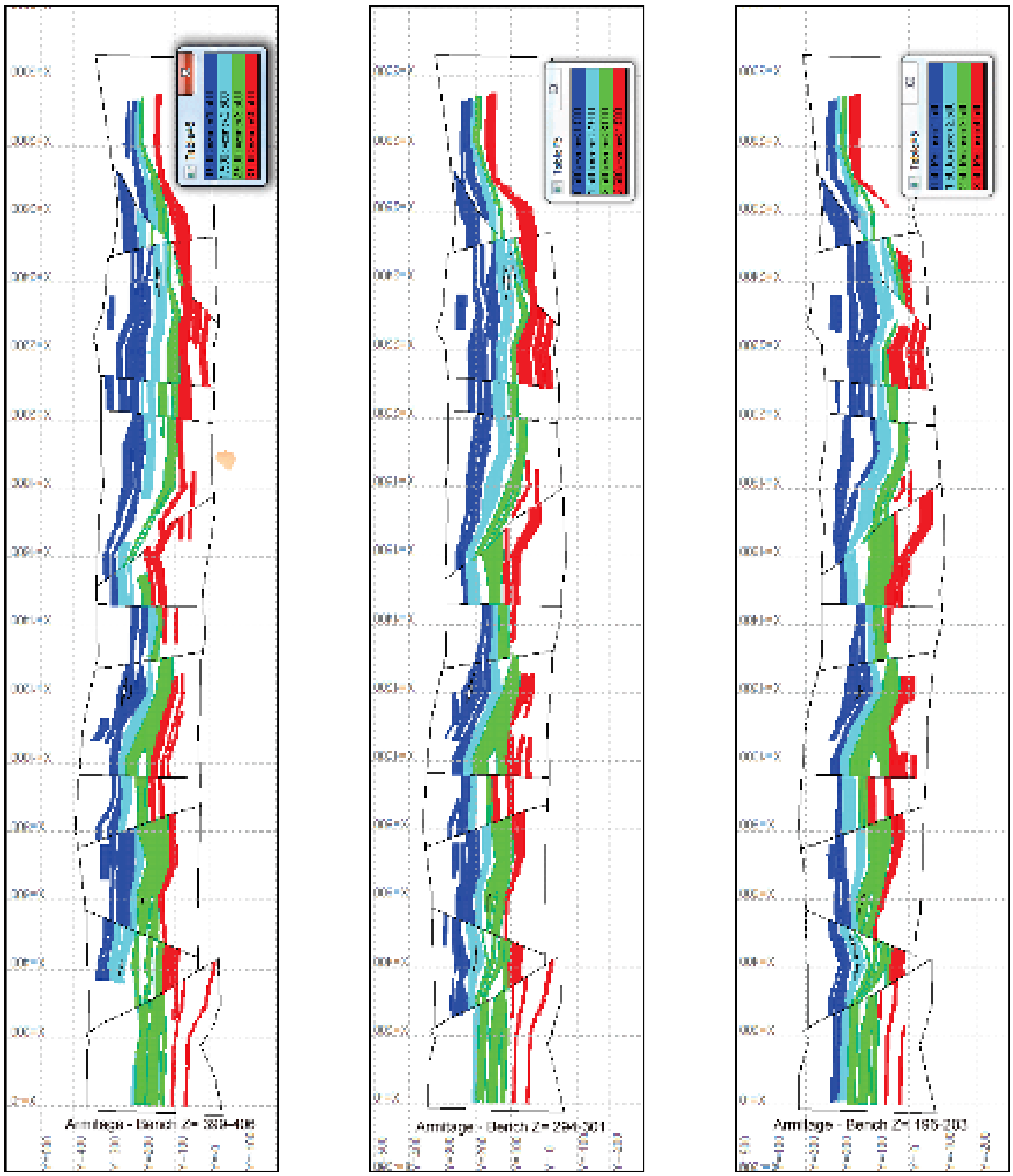


Figure 14-22: Armitage bench plans with blocks in mineralized units

Color according to rock unit: dark blue (1) = BCS, light blue (2) = MC, green (3) = ULS and red (4) = Titan.

14.3.3. Mineralized Volume Comparison

Table 14-5 shows the distribution of blocks and volumes according to units in the two deposits. In both deposits, BCS is the prevalent unit. The ULS unit is almost as important in Armitage, but is least important in Southwest. In both deposits, MC and Titan have about the same volume (between 20 and 25 Mm^3), which is about half that of BCS' in each case.

Table 14-5: Distribution of the mineralized material according to stratigraphic units

Deposit	Unit	# blocks	Volume (Mm^3)
Southwest	BCS	141,493	49.02
Southwest	MC	69,844	24.20
Southwest	ULS	45,432	15.67
Southwest	Titan	69,036	23.93
Southwest	All	325,805	112.82
Armitage	BCS	126,257	43.81
Armitage	MC	72,834	25.29
Armitage	ULS	111,763	38.68
Armitage	Titan	65,315	22.60
Armitage	All	376,169	130.37

14.4. Samples in Mineralized Zones

The grades of blocks within each mineralized solid are interpolated from the grades of samples in the same solid. Given that some drillholes deviate significantly from the center plane of the sections, the selection of samples in solids is done according to "From-To" limits of mineralized intercepts within the drillholes. Those intercepts become the basis for interpreted limits of mineralization on sections (Figure 14-12 to Figure 14-15 for Southwest and Figure 14-18 to Figure 14-21 for Armitage), which in turn is the basis of the mineralized solids.

Table 14-6 and Table 14-7 list the statistics of reported grades for assay intervals in unit intercepts in the two deposits. Four principal grades have been retained, namely %Fe_2O_3, %TiO_2, %V_2O_5 and %Satmagan (i.e., magnetite). Auxiliary grades for %Al_2O_3, %P_2O_5 and %S were also interpolated into the blocks to facilitate tracking of potential penalty items in the ultimate concentrates produced by the mine.

For the principal grades, intervals with missing values in the DH database have been "padded" before block interpolation. This padding is necessary to avoid any bias which could originate from a selective assaying of the most (visually) attractive intervals within limits of intercepts. Accurate padding is made possible by the strong correlation of principal grades in samples of almost all units (Figure 14-23 to Figure 14-38).

Given that %Fe_2O_3, %TiO_2 and %V_2O_5 are always available together, since derived from the WRA by ALS, three types of blank padding were performed: Pad1 = %Satmagan from %Fe_2O_3, Pad2 = %Fe_2O_3 +%TiO_2 +%V_2O_5 from %Satmagan and Pad3 = all four principal grades from PK hyper-spectral values. Numbers of padded samples of each type are shown in Table 14-6 and Table 14-7. They show that the proportion of padded intervals is much higher in Southwest (average 22.4% padded data over the four units), compared to Armitage (only 1.7% on average), which has been assayed in a more systematic manner.

For illustration purposes, here are the details of missing values in the intercepts of the BCS unit in the Southwest deposit. In those intercepts, we have:

a. Up to 1,354 3 m assay intervals;

b. A total of 1,067 of those intervals (79%) have WRA data with %Fe_2O_3, %TiO_2 and % V_2O_5;

c. 1,182 (87%) have Satmagan data;

d. Up to 87 intervals have a missing %Satmagan value derived from existing %Fe_2O_3, using the regression equation at the top of Figure 14-11;

e. We have a large number of those missing Satmagan intervals in holes SW-14-02 (22), SW-10-02 (19), SW-14-01 (13), SW-04-03 (9), SW-12-02 (9) and SW-08-01 (7);

f. Up to 202 intervals have missing %Fe_2O_3, %TiO_2 and % V_2O_5, which can be derived from existing %Satmagan using the regression equations at the top of Figure 14-27, Figure 14-31 and Figure 14-35;

g. We have a large number of those missing WRA data in new in-fill holes TG-FW-3 (30), SW-18+50 (20), SW-19+50 (18), SW-17+50 (17), TG-FW-1 (10) and SW-20+50 (7);

h. There is also missing WRA data in regular holes SW-15-01 (16), SW-15-02 (15), SW-17-02 (15), SW-01-02 (12), SW-19-01 (12), SW-16-01 (8), SW-17-01 (8) and SW-26-03 (6);

i. Up to 85 intervals do not have WRA data nor Satmagan value, but existing PK hyper-spectral data in those intervals allow calculation of reasonable %Fe_2O_3, %TiO_2, %V2O% and %Satmagan for those intervals;

j. Most of these instances are in holes SW-07-01 (18), SW-14-03 (8), SW-08-03 (7), SW-14-01 (5), SW-16-03 (5) and SW-24-03 (6).

The missing data in the mineralized units are the prime target for additional assaying to complete the systematic coverage prior to updating the resource models.

Table 14-6: Statistics of hole assay interval data in the Southwest mineralized units

Variable	Number	Min.	Median	Average	Max.	%CV
BSC (Pad1=87, Pad2=202, Pad3=85)						
Length (m)	1,354	2	3	3	3.75	NA
%SAT Original	1,182	0	10.2	12.6	41.7	72.6
%SAT Final	1,353	0	9.3	12.2	41.7	73.7
%Fe_2O_3 Original	1,067	4.3	27.7	30.2	64.5	38.8
%Fe_2O_3 Final	1,353	4.3	27.0	30.0	64.5	40.0
%TiO_2 Original	1,067	0.3	4.2	4.7	12.6	52.5
%TiO_2 Final	1,353	0.3	4.0	4.7	12.6	53.4
%V_2O_5 Original	1,067	0.02	0.33	0.36	0.91	53.6
%V_2O_5 Final	1,353	0.02	0.32	0.36	0.91	54.7
%Al_2O_3 Original	1,067	7.1	16.6	16.4	26.8	20.8
%P_2O_5 Original	1,067	0	0.020	0.039	0.36	113.1
%S Original(1)	955	0.01	0.09	0.11	1.43	85.6
MC (Pad1=19, Pad2=87, Pad3=12)						
Length (m)	655	3	3	3	4.5	NA
%SAT Original	624	0	25.7	23.6	42.2	39.6
%SAT Final	655	0	25.5	23.4	42.2	40.4
%Fe_2O_3 Original	556	9.5	49.8	47.2	68.9	24.8
%Fe_2O_3 Final	655	9.5	49.8	46.7	68.9	26.2
%TiO_2 Original	556	0.8	9.5	9.3	16.5	33.1
%TiO_2 Final	655	0.8	9.5	9.2	16.5	34.6
%V_2O_5 Original	556	0.03	0.62	0.58	0.96	30.4
%V_2O_5 Final	655	0.03	0.62	0.57	0.96	32.7
%Al_2O_3 Original	556	5.6	11.0	11.6	23.6	27.0
%P_2O_5 Original	556	0	0.020	0.031	0.24	107.5
%S Original	522	0.02	0.14	0.16	1.87	80.6

Variable	Number	Min.	Median	Average	Max.	%CV
ULS (Pad1=14, Pad2=63, Pad3=8)						
Length (m)	433	1	3	3	3	NA
%SAT Original	401	0	17.1	16.5	37.7	59.7
%SAT Final	423	0	16.4	16.1	37.7	61.4
%Fe_2O_3 Original	352	8.7	42.0	39.2	64.7	36.4
%Fe_2O_3 Final	423	8.7	41.8	38.8	64.7	35.3
%TiO_2 Original	352	0.8	9.1	8.5	15.8	45.2
%TiO_2 Final	423	0.8	9.0	8.3	15.8	43.9
%V_2O_5 Original	351	0.02	0.34	0.36	0.84	50.5
%V_2O_5 Final	423	0.02	0.34	0.35	0.84	54.1
%Al_2O_3 Original	352	5.9	11.4	12.3	24.0	30.8
%P_2O_5 Original	352	0	0.030	0.040	0.28	96.2
%S Original	336	0.02	0.20	0.24	0.90	51.5
Titan (Pad1=24, Pad2=71, Pad3=43)						
Length (m)	638	1	3	3	3	NA
%SAT Original	508	0	4.6	5.9	29.3	90.7
%SAT Final	575	0	4.5	5.9	29.3	90.7
%Fe_2O_3 Original	461	9.5	29.8	29.5	57.7	30.3
%Fe_2O_3 Final	575	8.4	29.0	28.8	57.7	30.9
%TiO_2 Original	461	0.8	5.3	5.5	14.6	44.4
%TiO_2 Final	575	0.8	5.2	5.3	14.7	46.4
%V_2O_5 Original	461	0.01	0.11	0.13	0.54	69.9
%V_2O_5 Final	575	0.01	0.10	0.13	0.58	72.1
%Al_2O_3 Original	461	7.5	10.8	11.5	23.0	21.4
%P_2O_5 Original	461	0	0.020	0.043	0.30	100.0
%S Original	455	0	0.28	0.29	1.48	51.7

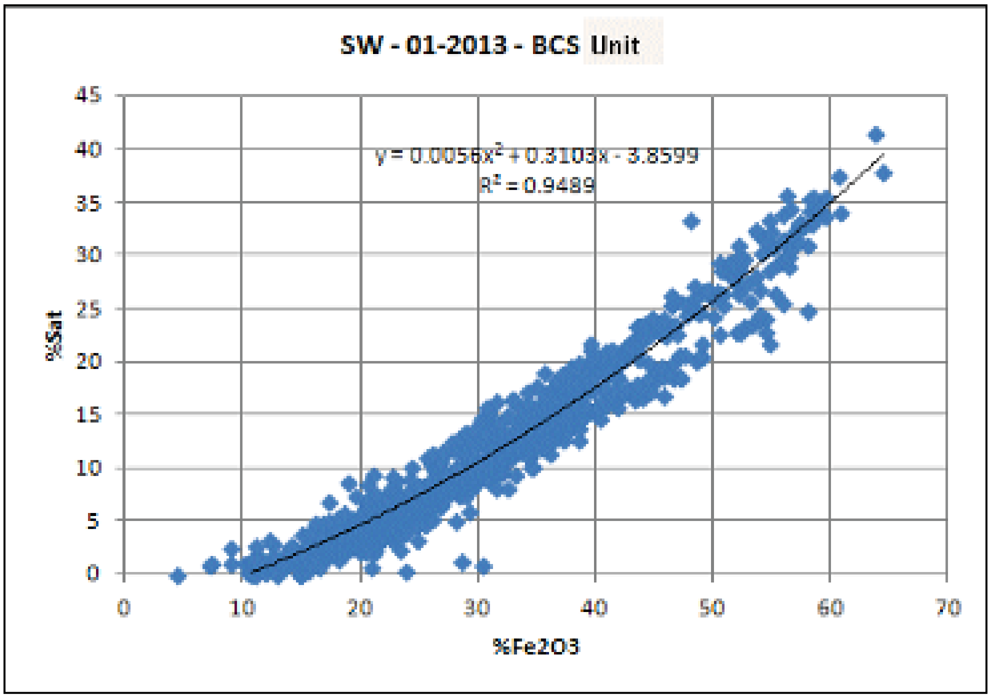
Excludes outlier of 6.41%S - See text for explanation of Pad1, Pad2 and Pad3.

Table 14-7: Statistics of hole assay interval data in the Armitage mineralized units

Variable	Number	Min.	Median	Average	Max.	%CV
BSC (Pad1=2, Pad2=10, Pad3=6)						
Length (m)	995	3	3	3	3.2	NA
%SAT Original	987	0.1	11.1	12.7	39.8	62.9
%SAT Final	995	0.1	10.9	12.7	39.8	63.2
%Fe_2O_3 Original	979	7.4	29.1	30.7	59.5	33.7
%Fe_2O_3 Final	995	7.4	28.9	30.7	59.5	33.7
%TiO_2 Original	979	0.3	4.6	5.0	10.9	46.7
%TiO_2 Final	995	0.3	4.6	5.0	10.9	46.5
%V_2O_5 Original	979	0.01	0.35	0.37	0.94	44.7
%V_2O_5 Final	995	0.01	0.35	0.37	0.94	44.7
%Al_2O_3 Original	979	9.3	16.4	16.5	26.9	18.5
%P_2O_5 Original	979	0.008	0.020	0.031	0.26	98.3
%S Original(1)	980	0.01	0.11	0.14	1.81	78.3
MC (Pad1=9, Pad2=4, Pad3=4)						
Length (m)	593	2	3	3	3.5	NA
%SAT Original	580	0.4	17.6	17.1	39.7	44.0
%SAT Final	593	0	17.4	16.9	39.7	44.6
%Fe_2O_3 Original	585	7.2	38.3	37.6	57.9	26.6
%Fe_2O_3 Final	593	7.2	38.1	37.5	57.9	26.8
%TiO_2 Original	585	0.7	7.5	7.3	13.2	33.3
%TiO_2 Final	593	0.7	7.5	7.3	13.2	33.5
%V_2O_5 Original	585	0.01	0.45	0.43	0.82	34.2
%V_2O_5 Final	593	0.01	0.44	0.43	0.82	34.6
%Al_2O_3 Original	585	8.4	13.3	13.8	24.1	18.2
%P_2O_5 Original	585	0.007	0.019	0.030	0.19	95.4
%S Original	585	0.04	0.22	0.26	1.57	63.4

Variable	Number	Min.	Median	Average	Max.	%CV
ULS (Pad1=12, Pad2=0, Pad3=2)						
Length (m)	831	2	3	3	4	NA
%SAT Original	814	0.2	12.7	13.2	35.1	54.0
%SAT Final	831	0.2	12.6	13.1	35.1	53.8
%Fe_2O_3 Original	824	2.3	35.5	35.3	58.7	26.0
%Fe_2O_3 Final	831	2.3	35.5	35.3	58.7	26.0
%TiO_2 Original	824	0.2	7.7	7.5	13.0	32.0
%TiO_2 Final	831	0.2	7.7	7.5	13.0	32.0
%V_2O_5 Original	824	0.01	0.31	0.33	0.75	44.0
%V_2O_5 Final	831	0.01	0.31	0.33	0.75	44.1
%Al_2O_3 Original	824	8.0	13.9	14.1	22.8	16.2
%P_2O_5 Original	824	0.007	0.020	0.020	0.15	84.8
%S Original	824	0.02	0.40	0.43	1.42	46.8
Titan (Pad1=0, Pad2=2, Pad3=0)						
Length (m)	400	1	3	3	3.7	NA
%SAT Original	392	0.3	3.5	4.4	15.9	62.9
%SAT Final	392	0.3	3.5	4.4	15.9	62.9
%Fe_2O_3 Original	390	5.4	25.8	26.3	39.3	18.4
%Fe_2O_3 Final	392	5.4	25.8	26.4	39.3	18.5
%TiO_2 Original	390	0.2	4.1	4.4	8.3	31.8
%TiO_2 Final	392	0.2	4.1	4.4	8.3	32.0
%V_2O_5 Original	390	0.01	0.09	0.10	0.33	39.3
%V_2O_5 Final	392	0.01	0.09	0.10	0.33	39.3
%Al_2O_3 Original	390	6.6	12.0	12.1	27.4	19.1
%P_2O_5 Original	390	0.010	0.022	0.031	0.33	78.8
%S Original	390	0.01	0.36	0.37	1.29	42.0

Excludes outlier of 6.41%S - See text for explanation of Pad1, Pad2 and Pad3.



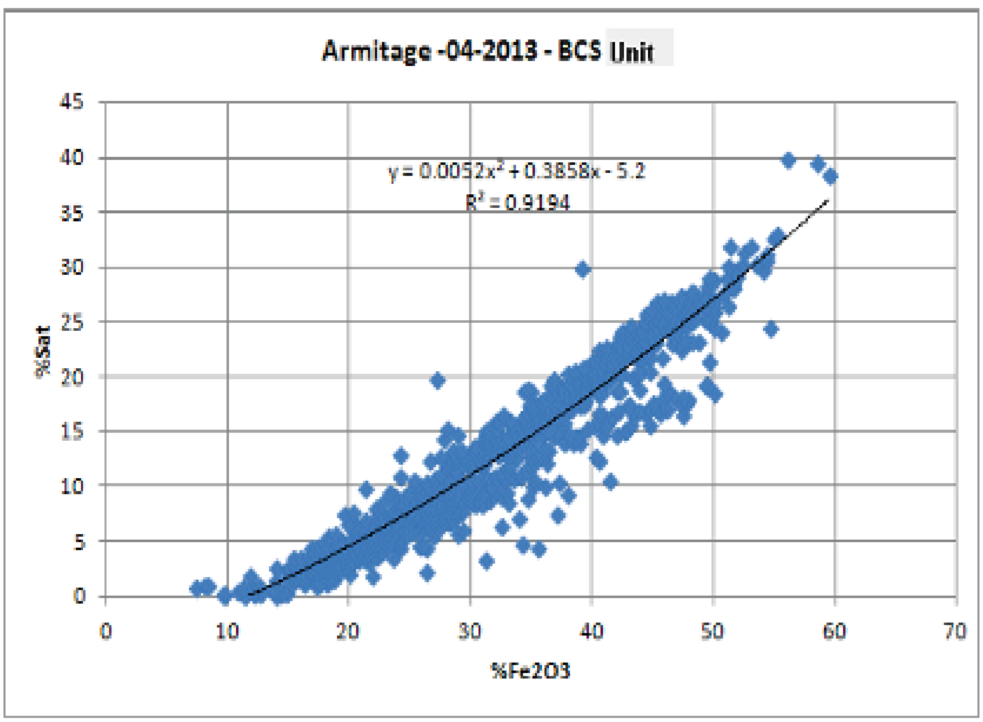


Figure 14-23: Correlation and regression of %Satmagan on %Fe_2O_3. BCS Unit of Southwest in upper image and Armitage in lower image

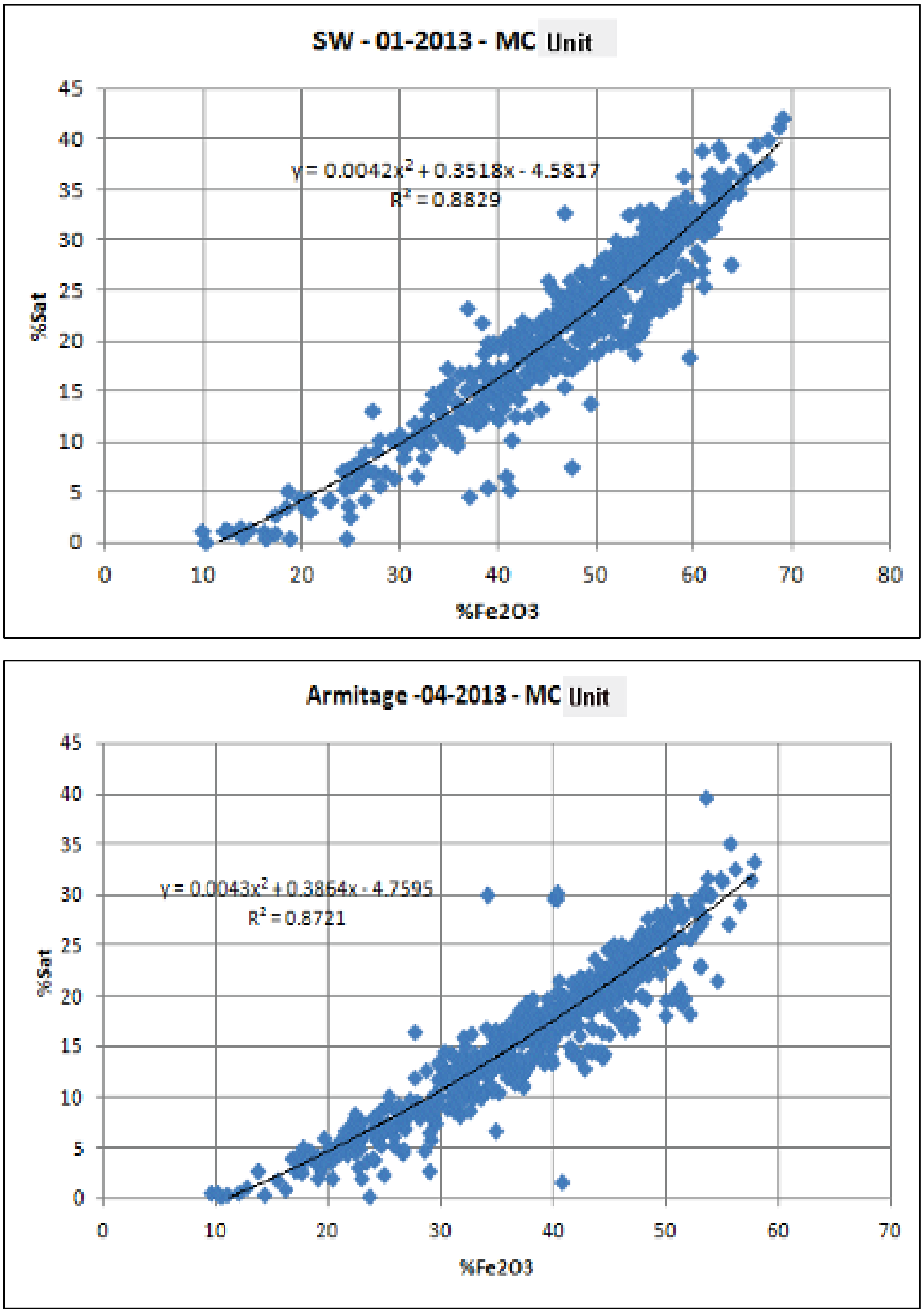
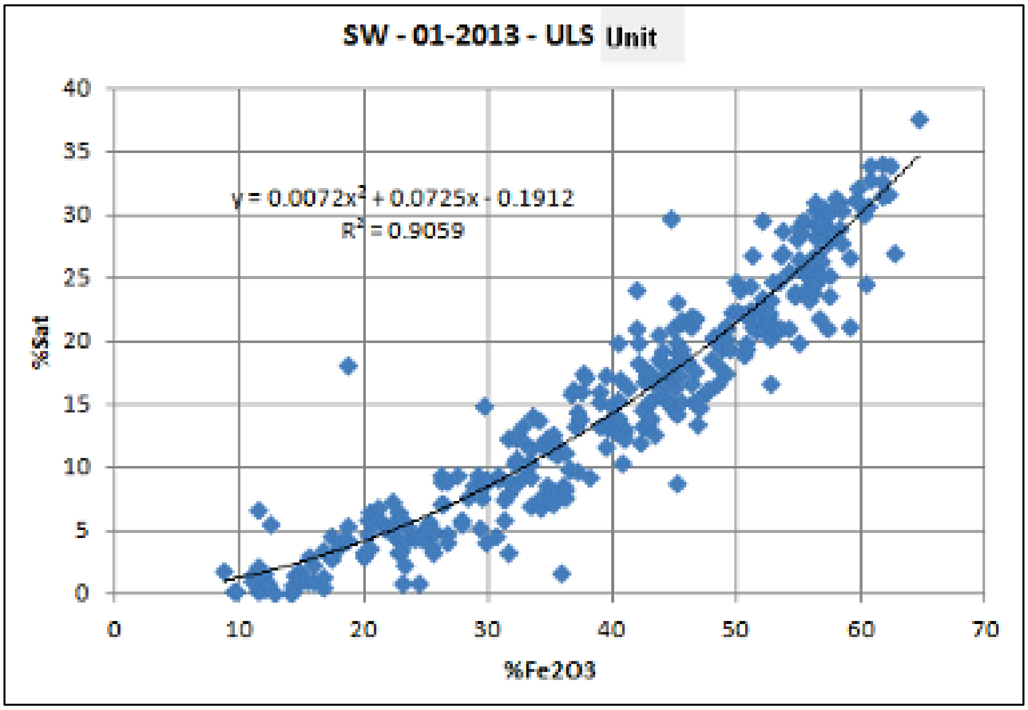


Figure 14-24: Correlation and regression of %Satmagan on %Fe_2O_3. MC unit of Southwest in upper image and Armitage in lower image



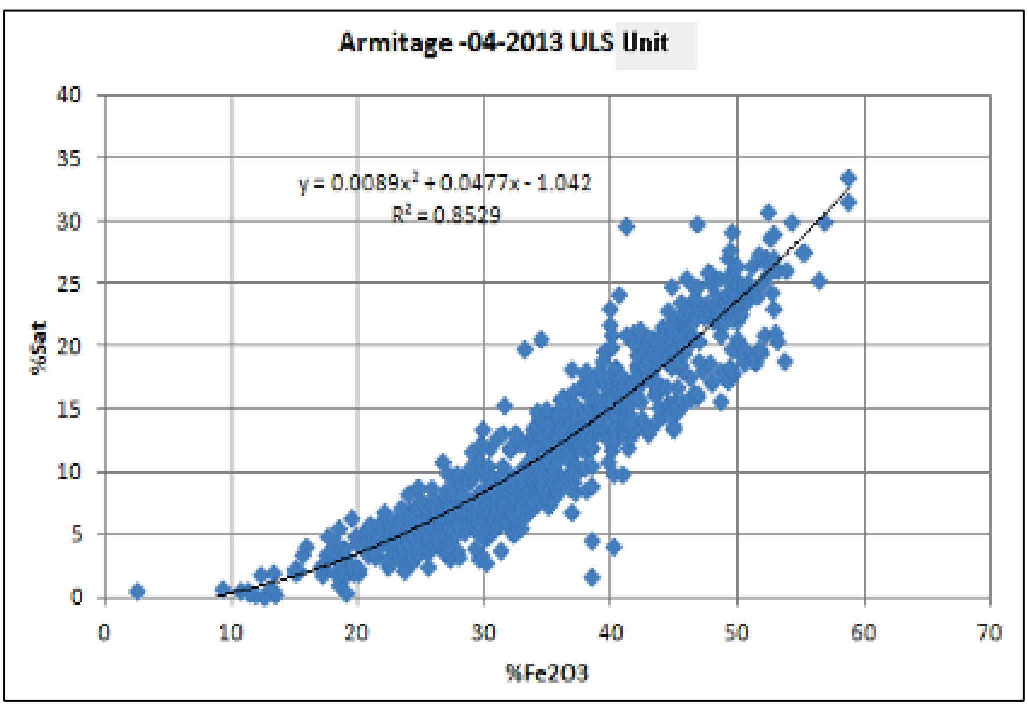
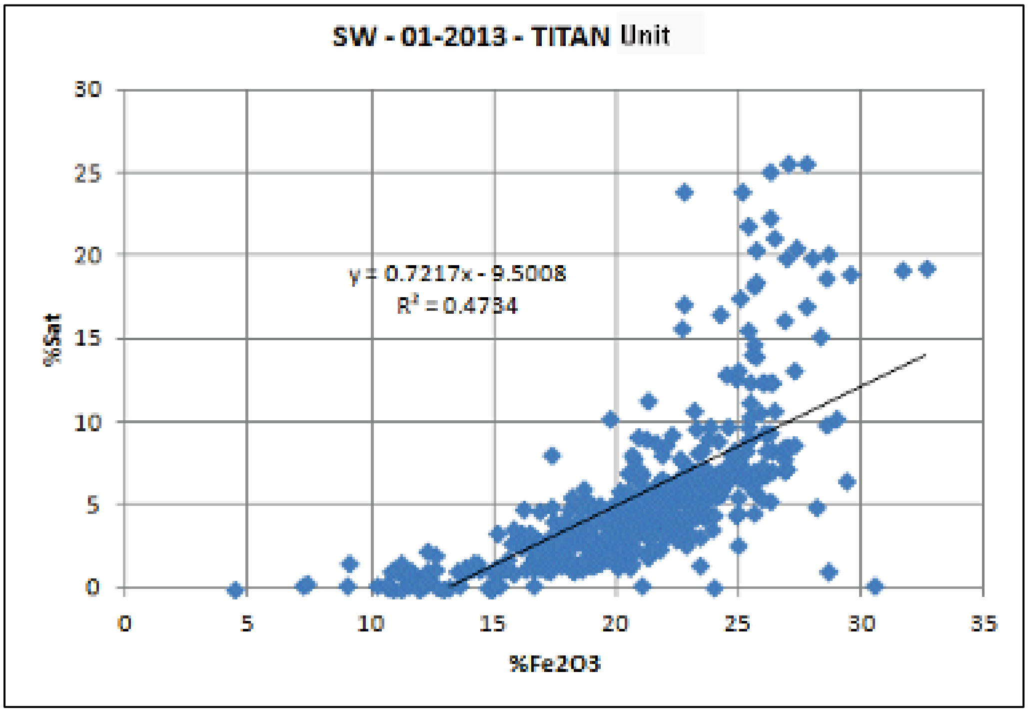


Figure 14-25: Correlation and regression of %Satmagan on %Fe_2O_3. ULS Unit of Southwest in upper image and Armitage in lower image



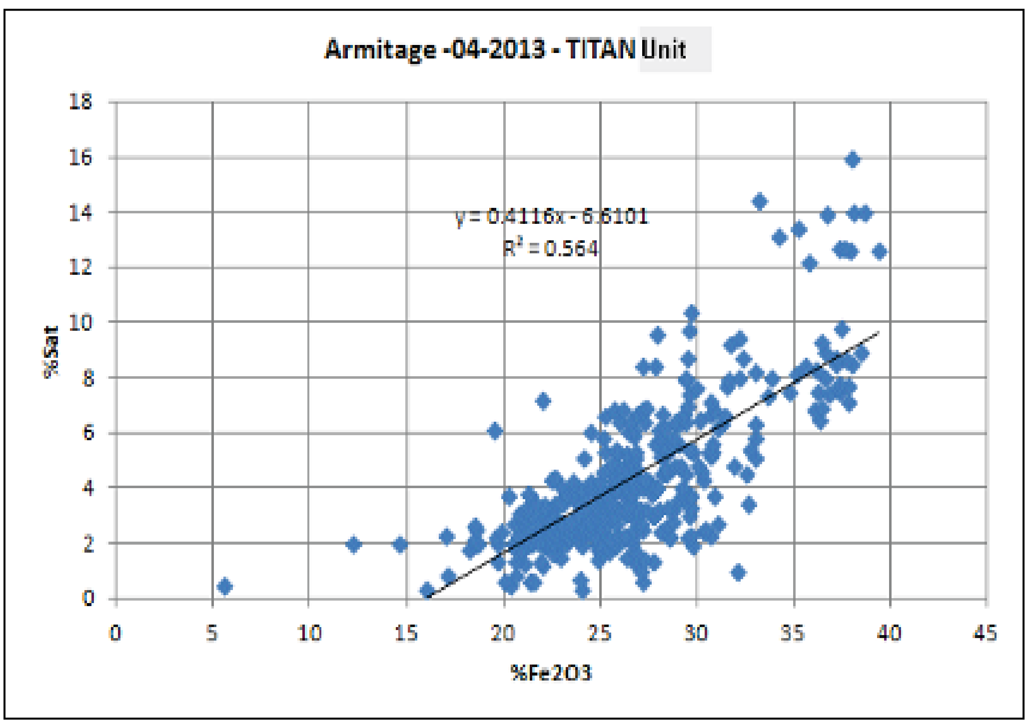
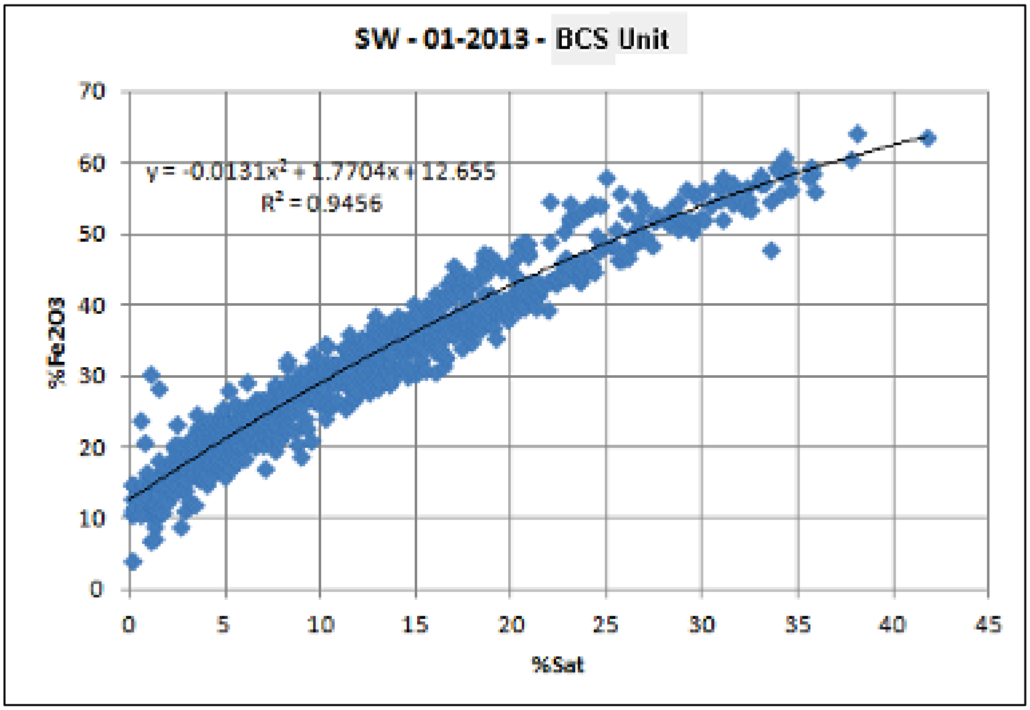


Figure 14-26: Correlation and regression of %Satmagan on %Fe_2O_3. Titan unit of Southwest in upper image and Armitage in lower image



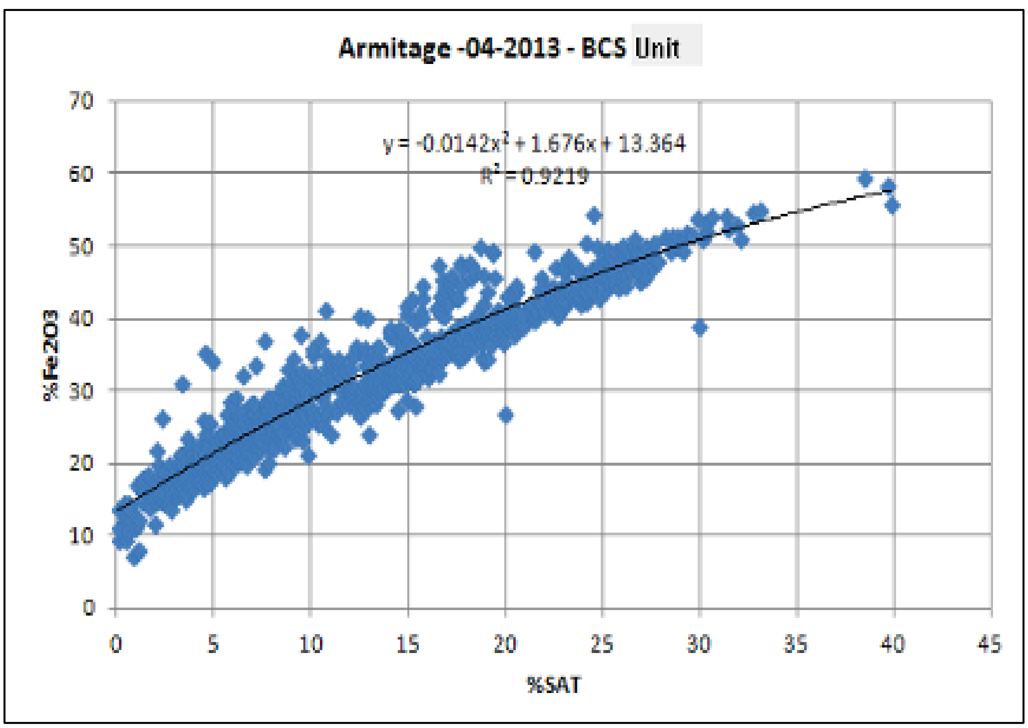
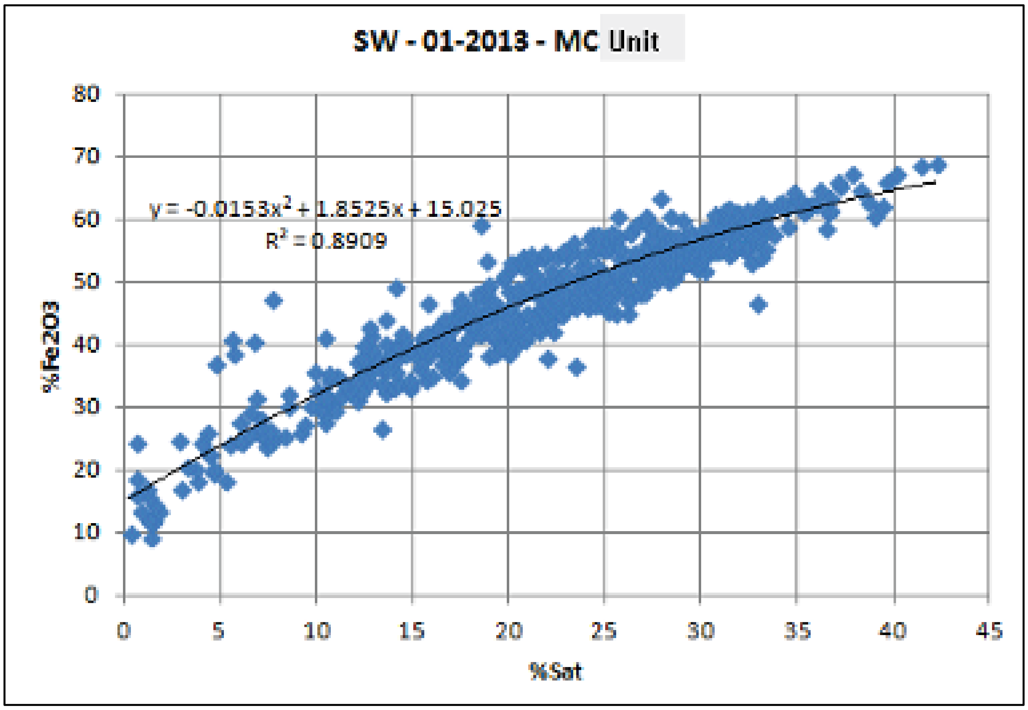


Figure 14-27: Correlation and regression of %Fe_2O_3 with %Satmagan. BCS Unit of Southwest in upper image and Armitage in lower image



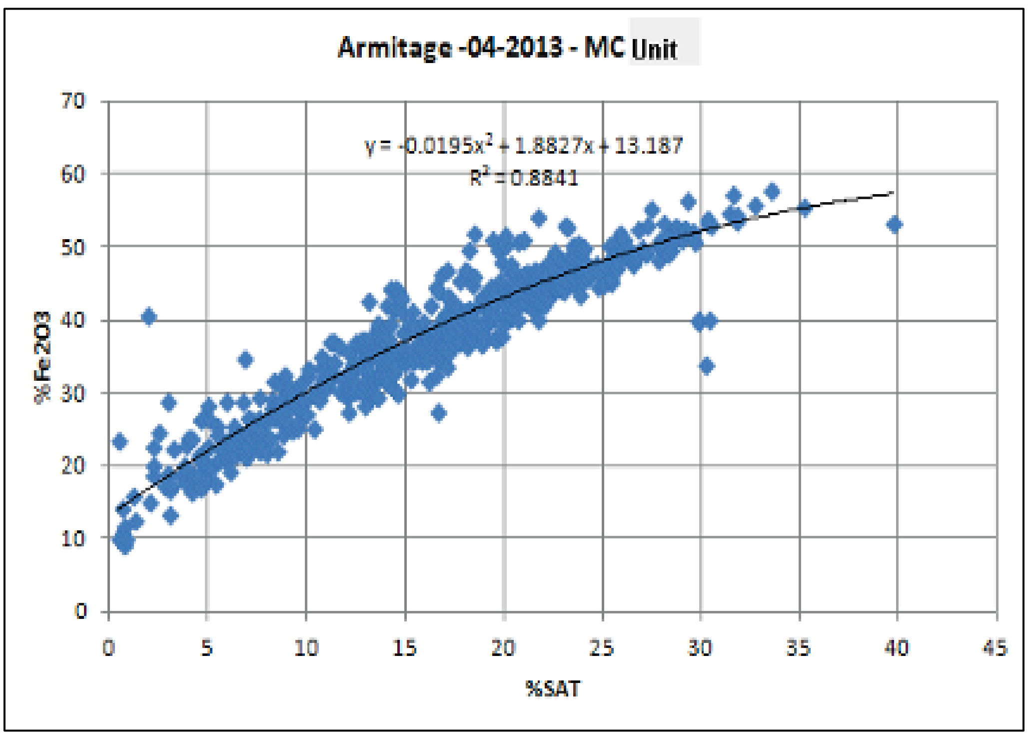
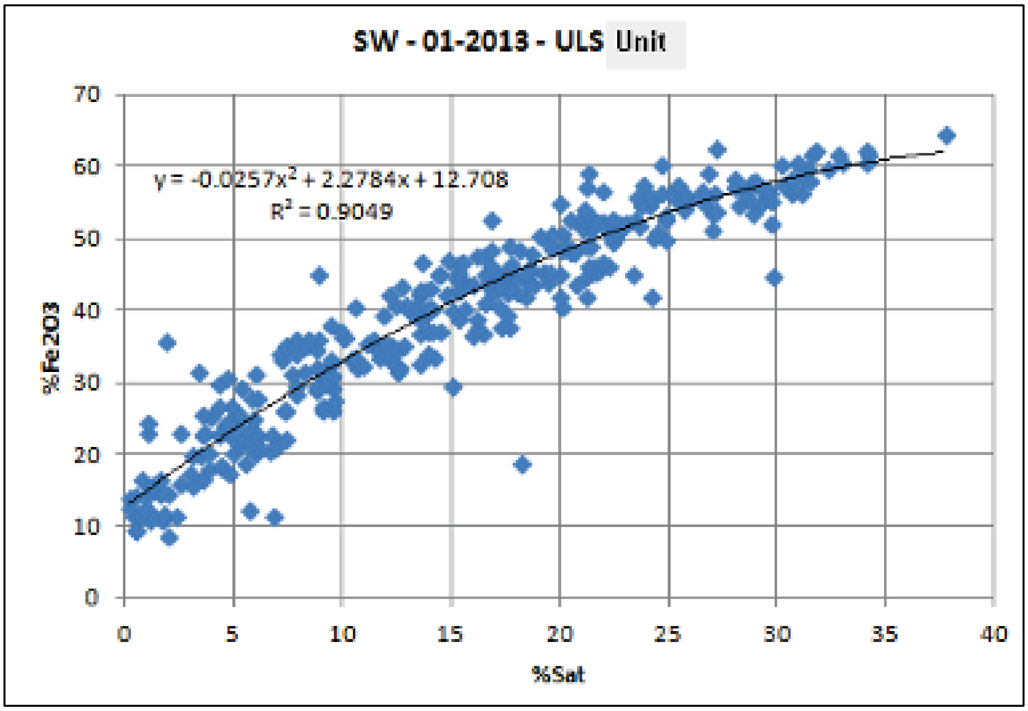


Figure 14-28: Correlation and regression of %Fe_2O_3 with %Satmagan. MC Unit of Southwest in upper image and Armitage in lower image



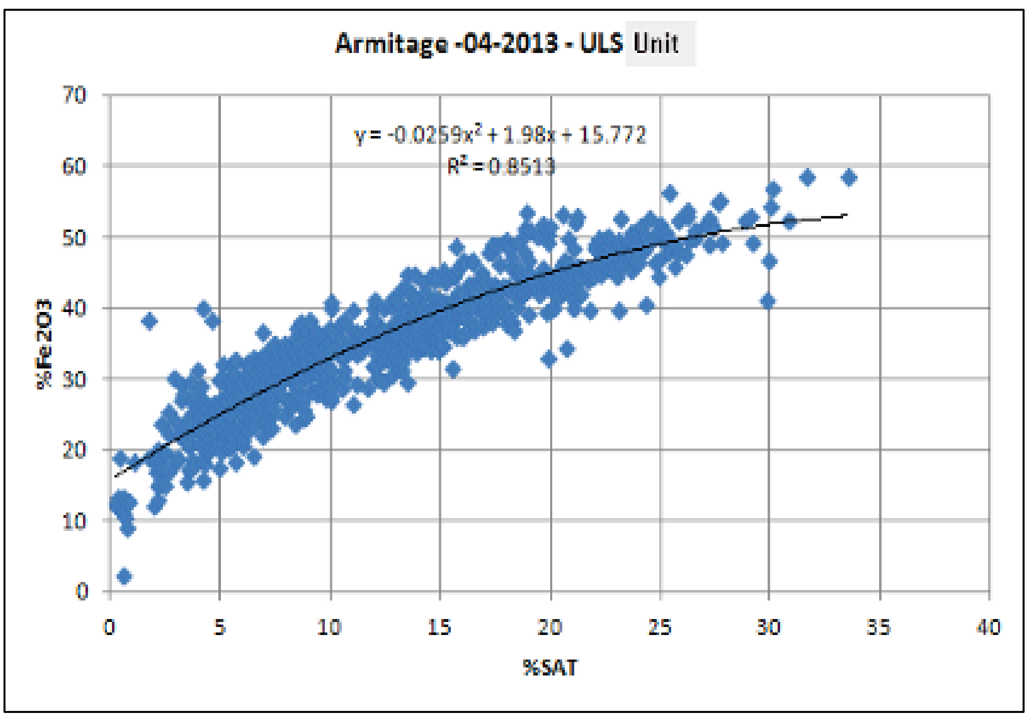
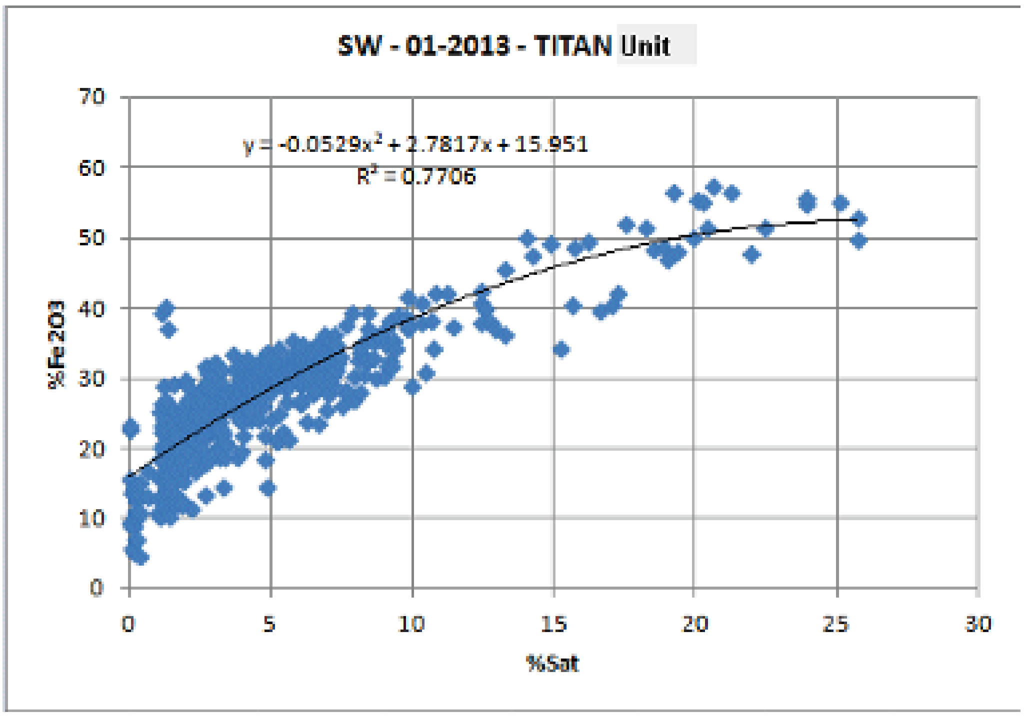


Figure 14-29: Correlation and regression of %Fe_2O_3 with %Satmagan. ULS Unit of Southwest in upper image and Armitage in lower image



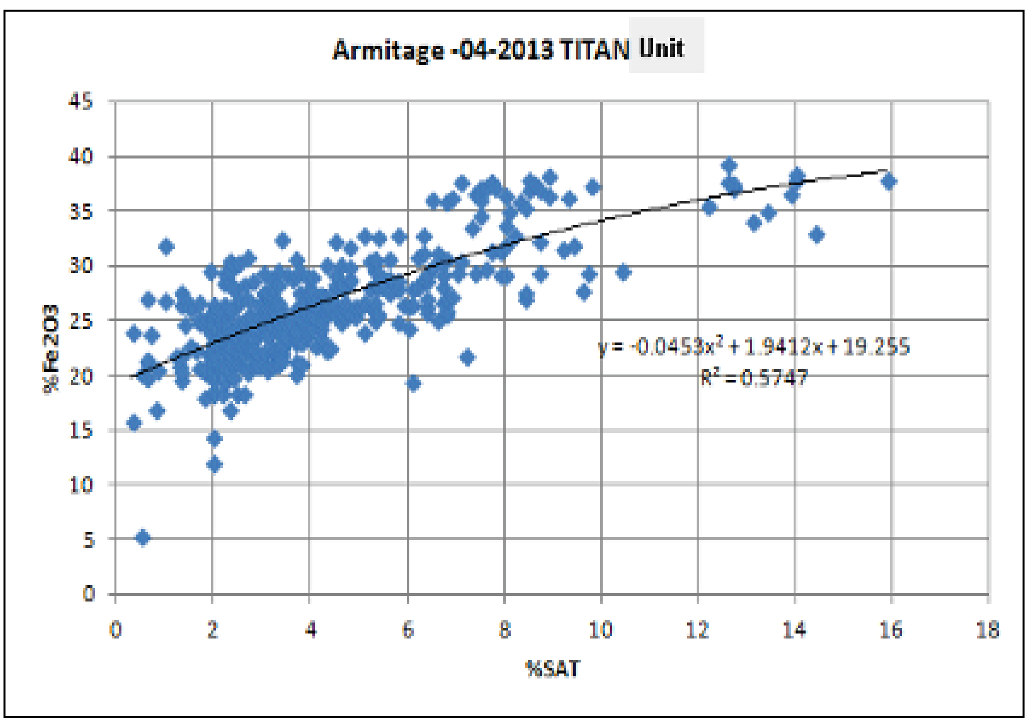
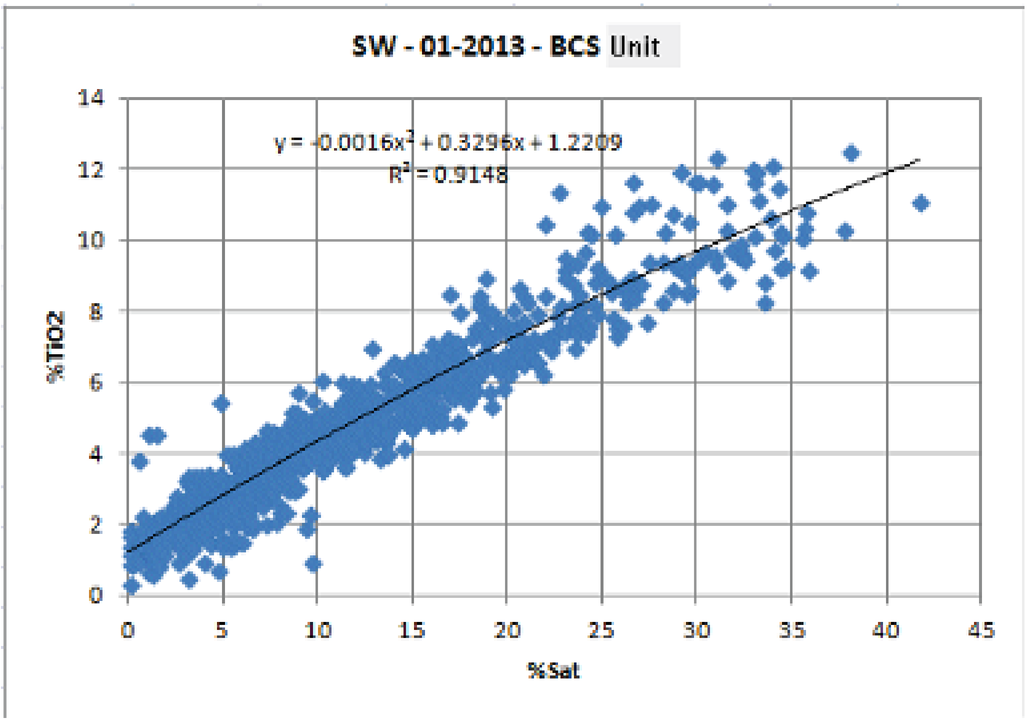


Figure 14-30: Correlation and regression of %Fe_2O_3 with %Satmagan. Titan Unit of Southwest in upper image and Armitage in lower image



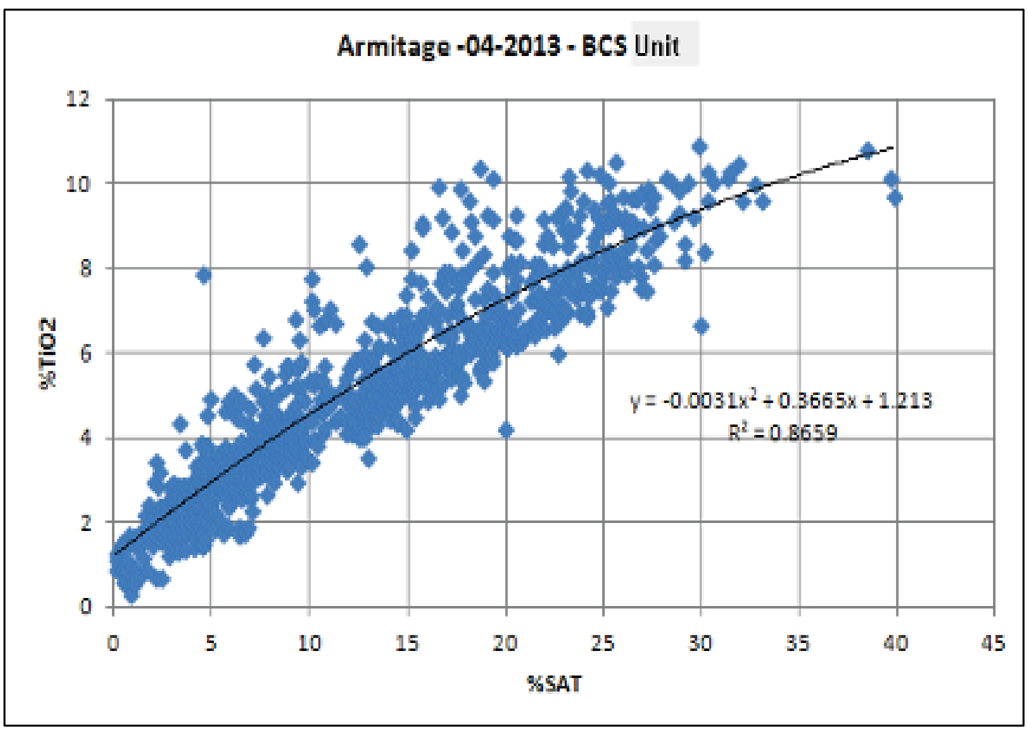


Figure 14-31: Correlation and regression of %TiO_2 with %Satmagan. BCS Unit of Southwest in upper image and Armitage in lower image

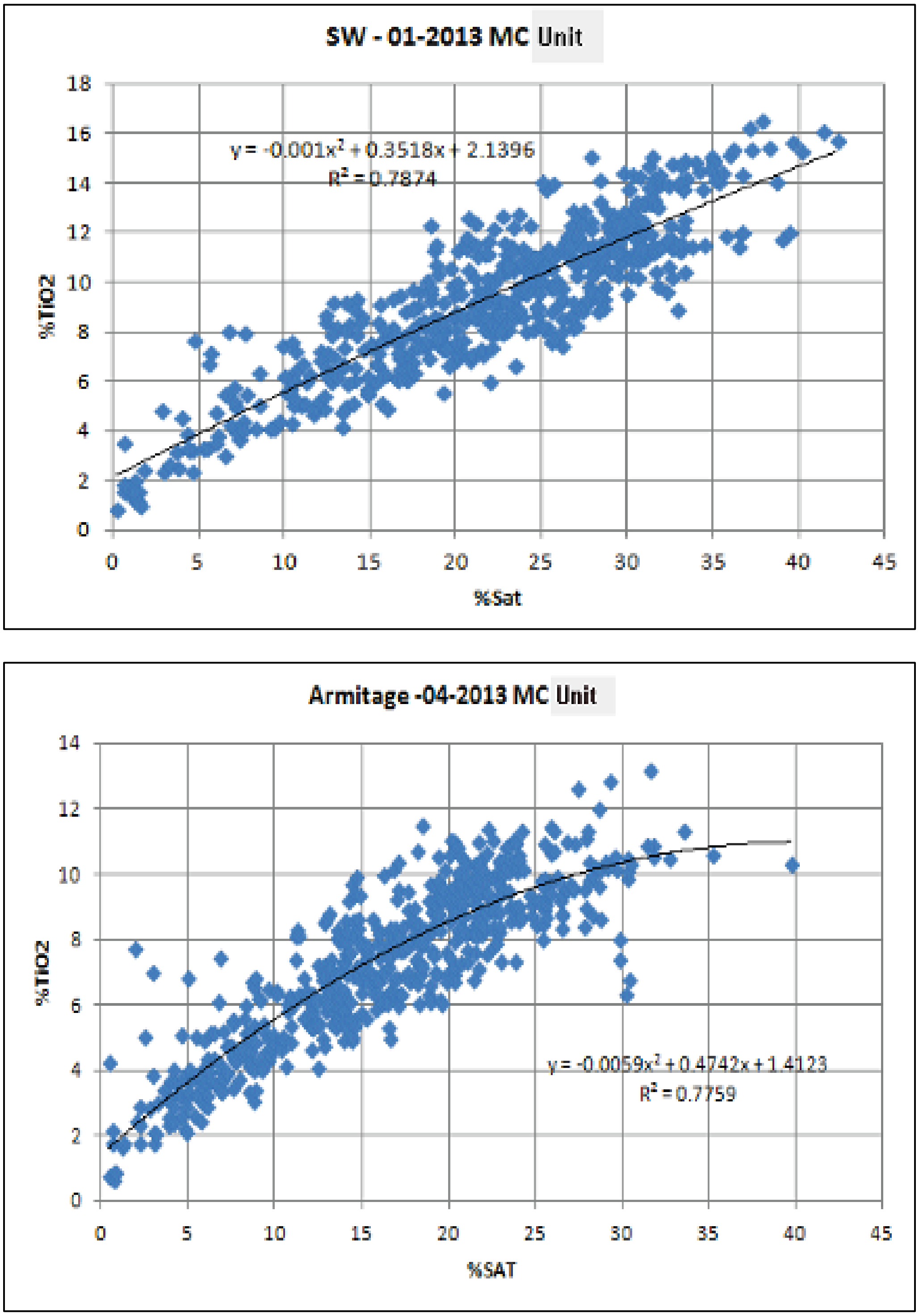
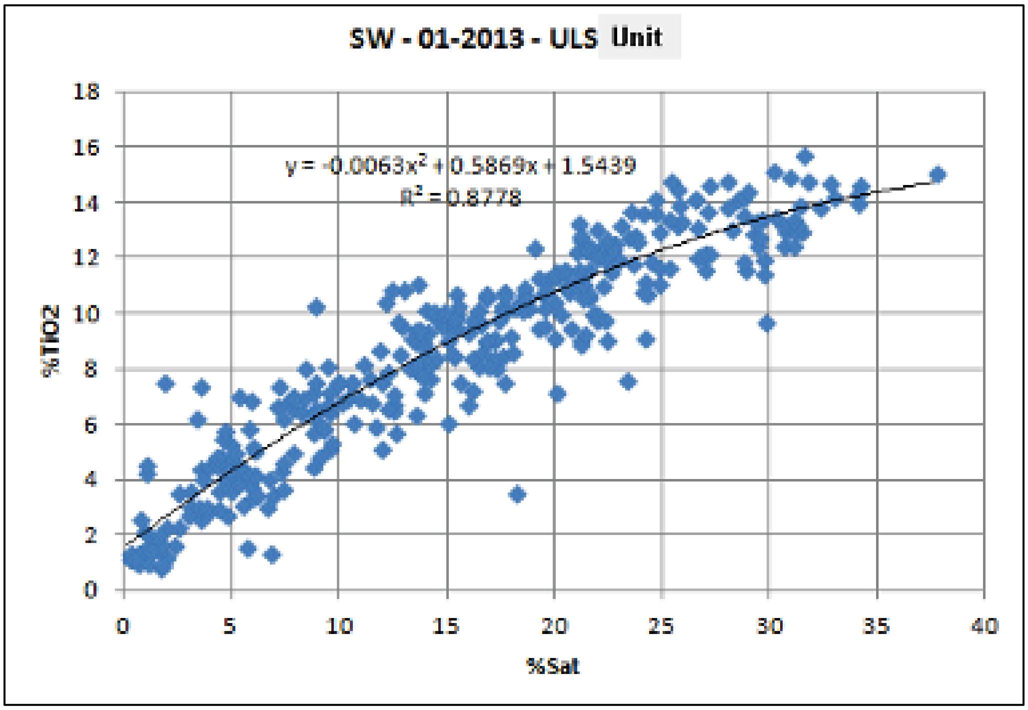


Figure 14-32: Correlation and regression of %TiO_2 with %Satmagan. MC Unit of Southwest in upper image and Armitage in lower image



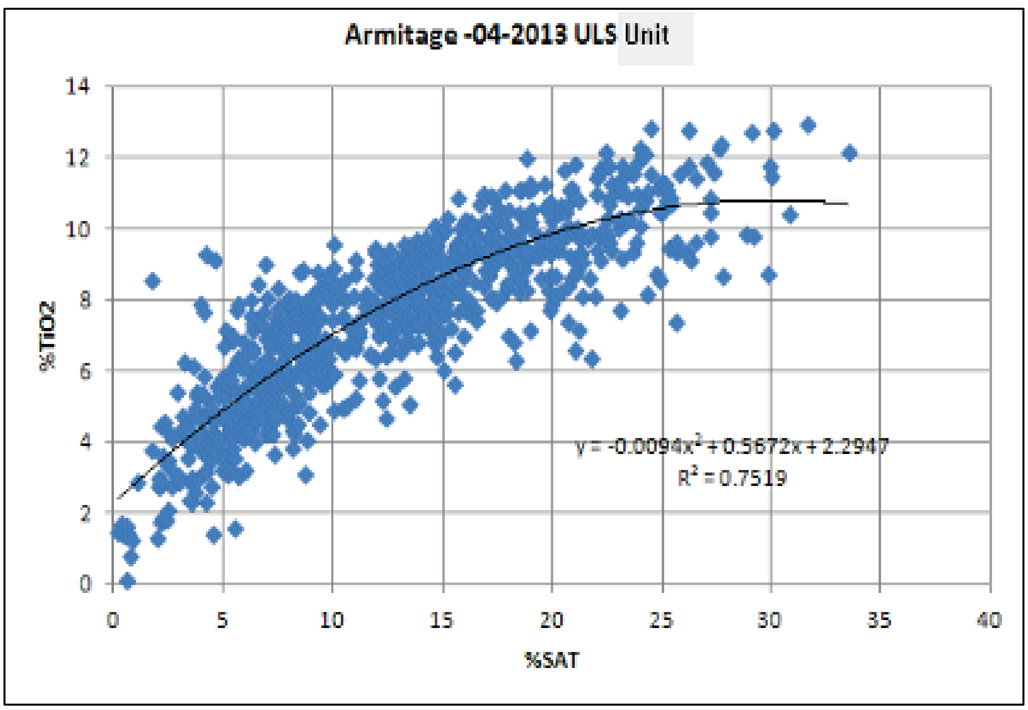
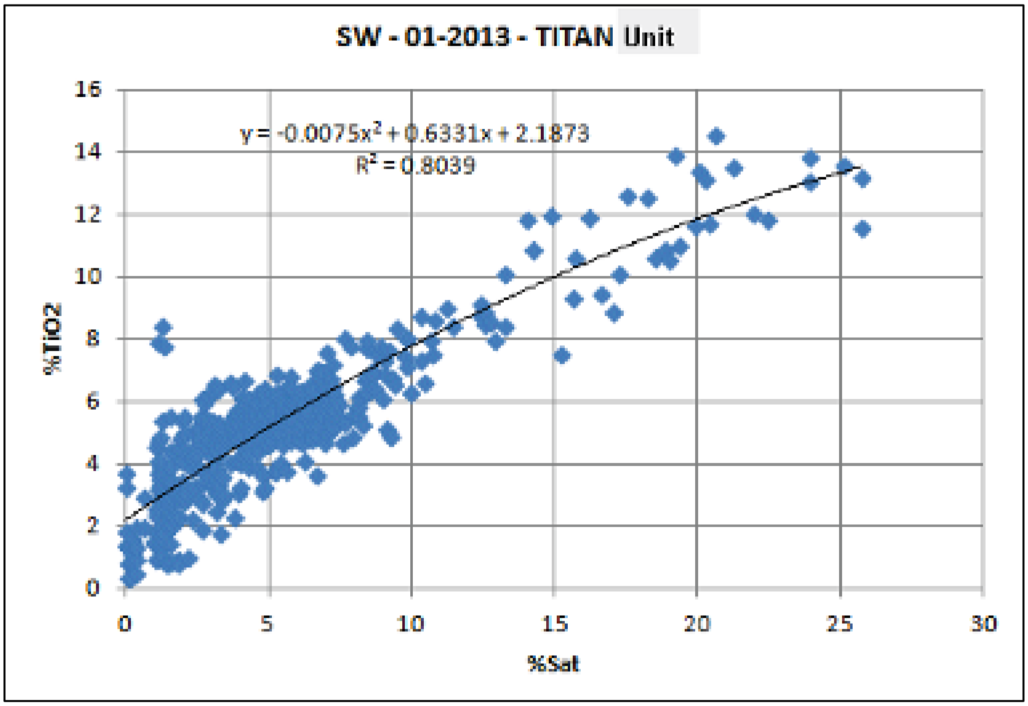


Figure 14-33: Correlation and regression of %TiO_2 with %Satmagan. ULS Unit of Southwest in upper image and Armitage in lower image



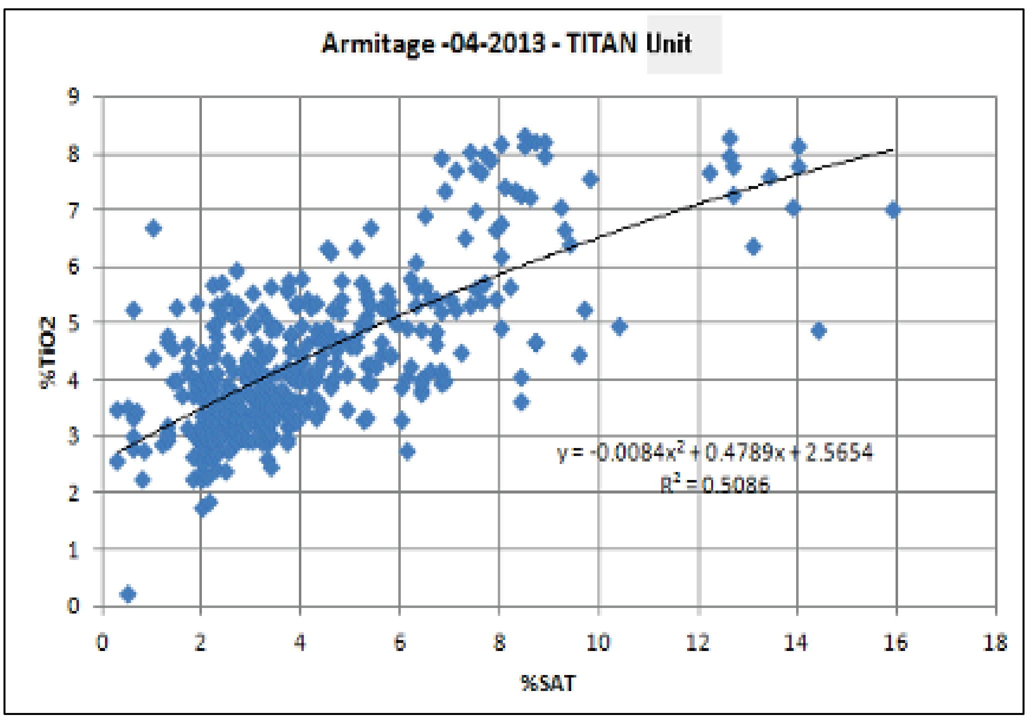
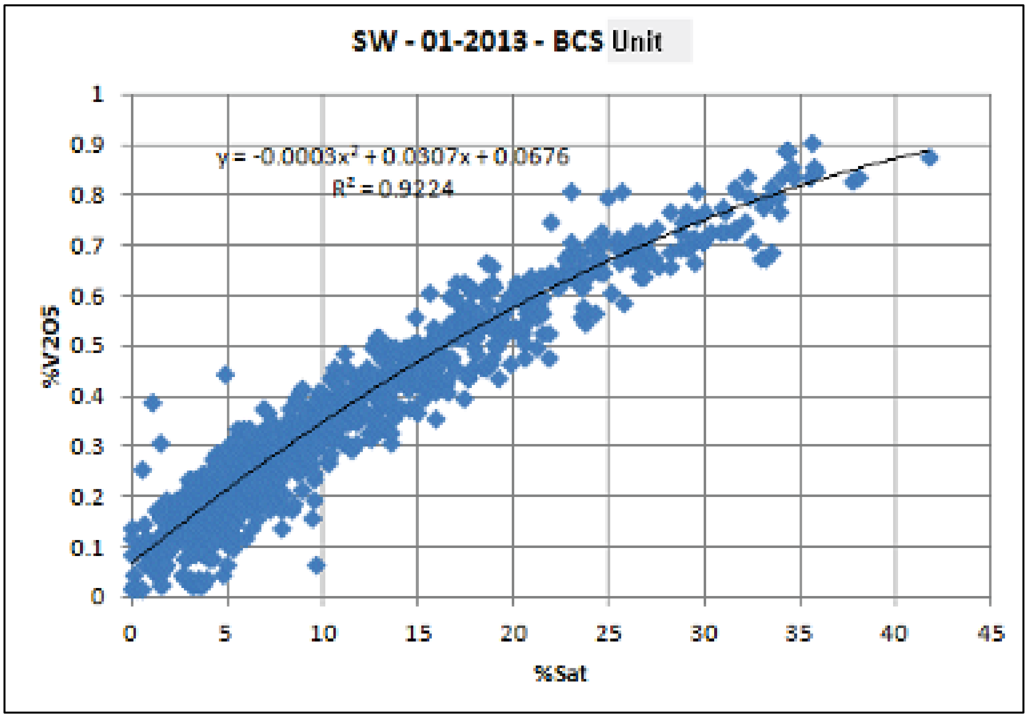


Figure 14-34: Correlation and regression of %TiO_2 with %Satmagan. Titan unit of Southwest in upper image and Armitage in lower image



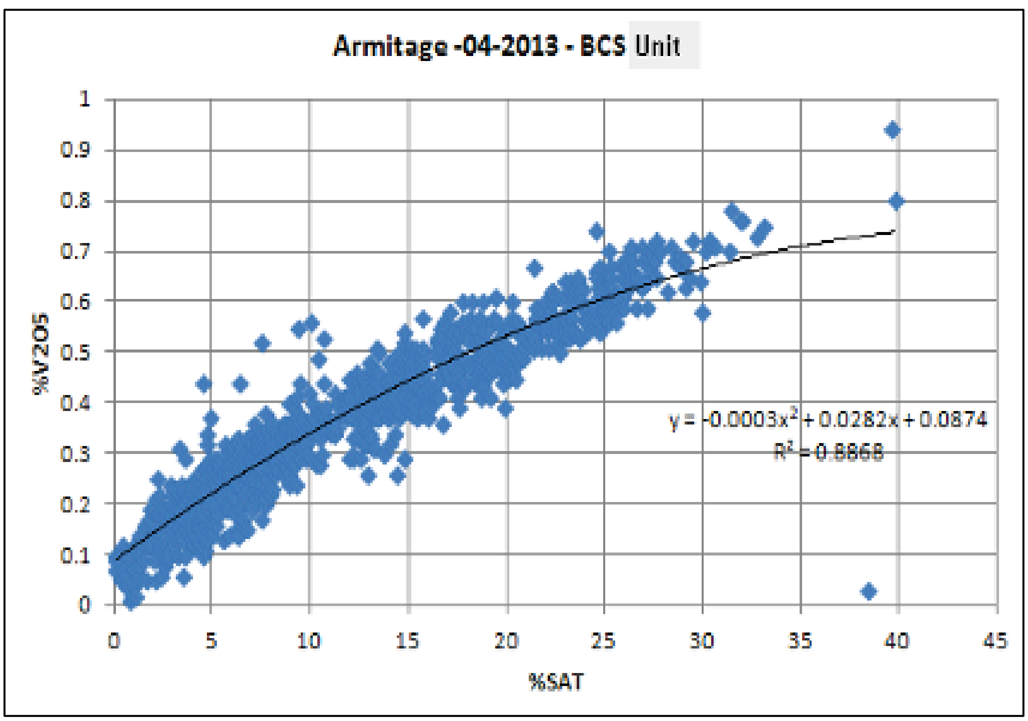


Figure 14-35: Correlation and regression of %V_2O_5 on %Satmagan. BCS Unit of Southwest in upper image and Armitage in lower image

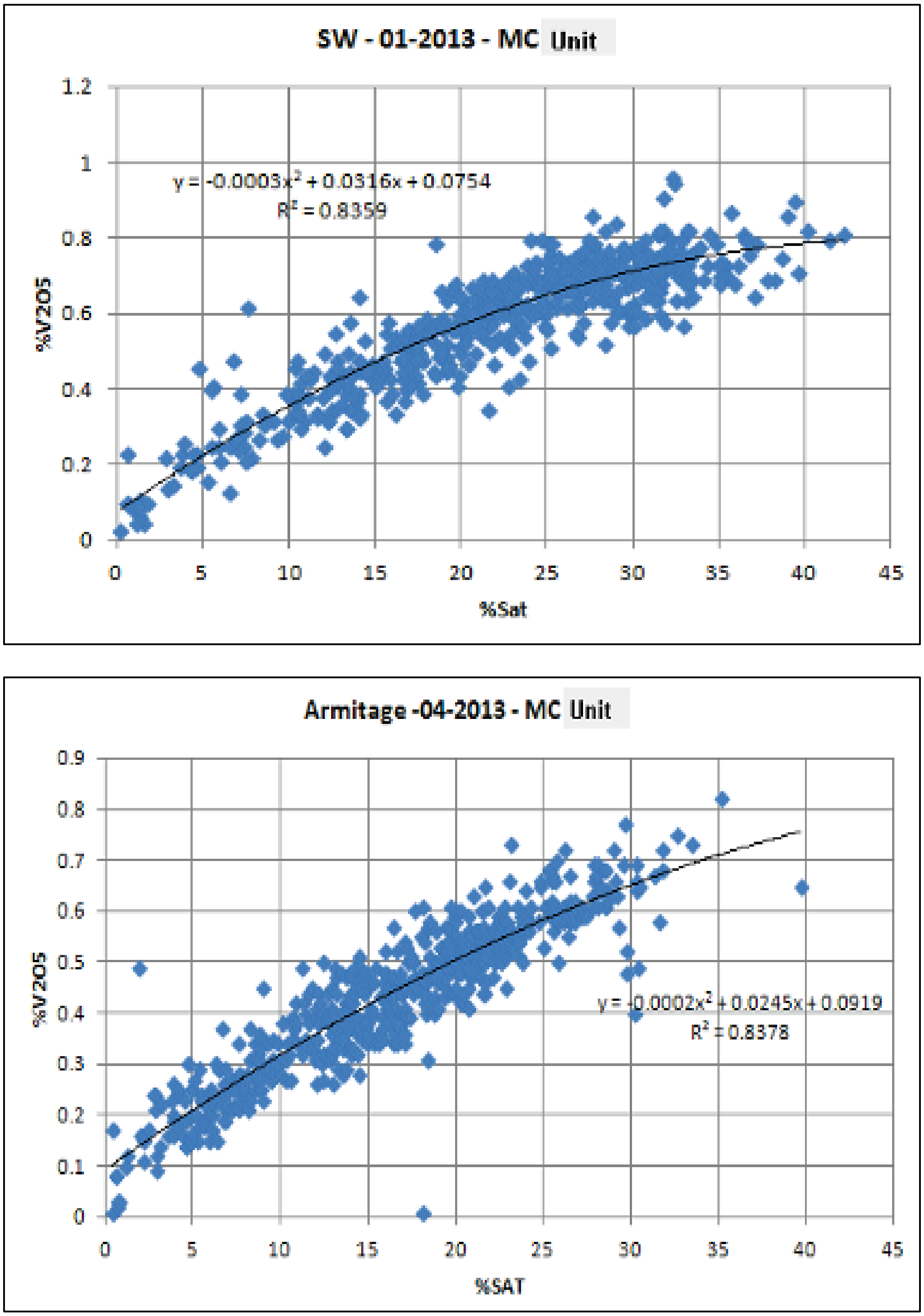
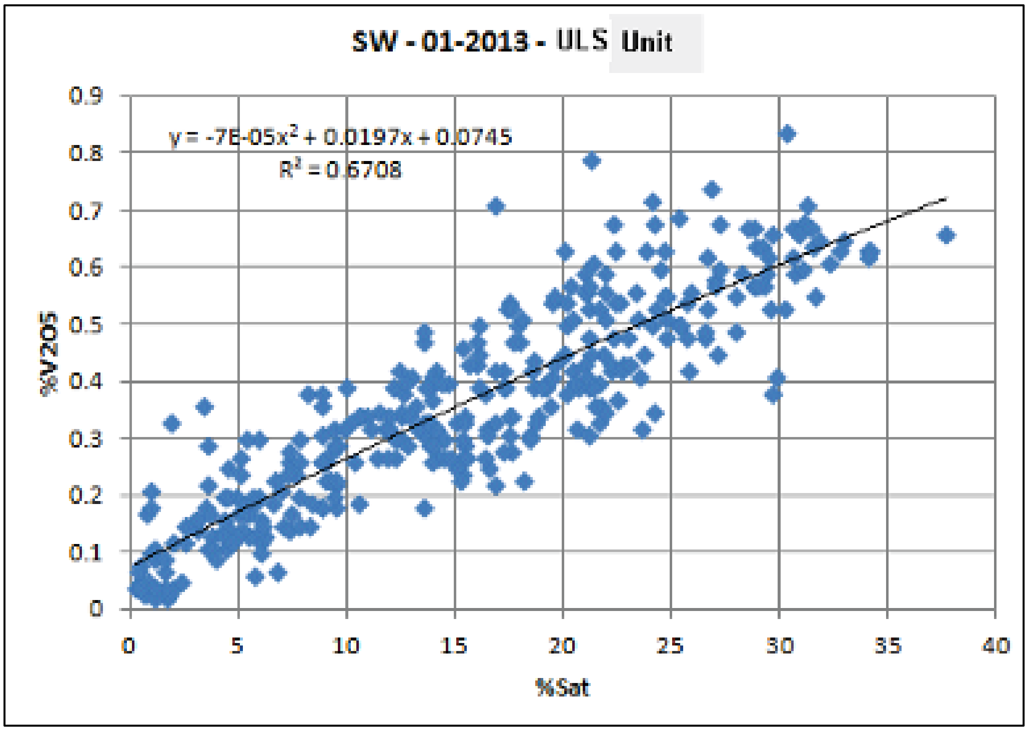


Figure 14-36: Correlation and regression of %V_2O_5 on %Satmagan. MC Unit of Southwest in upper image and Armitage in lower image



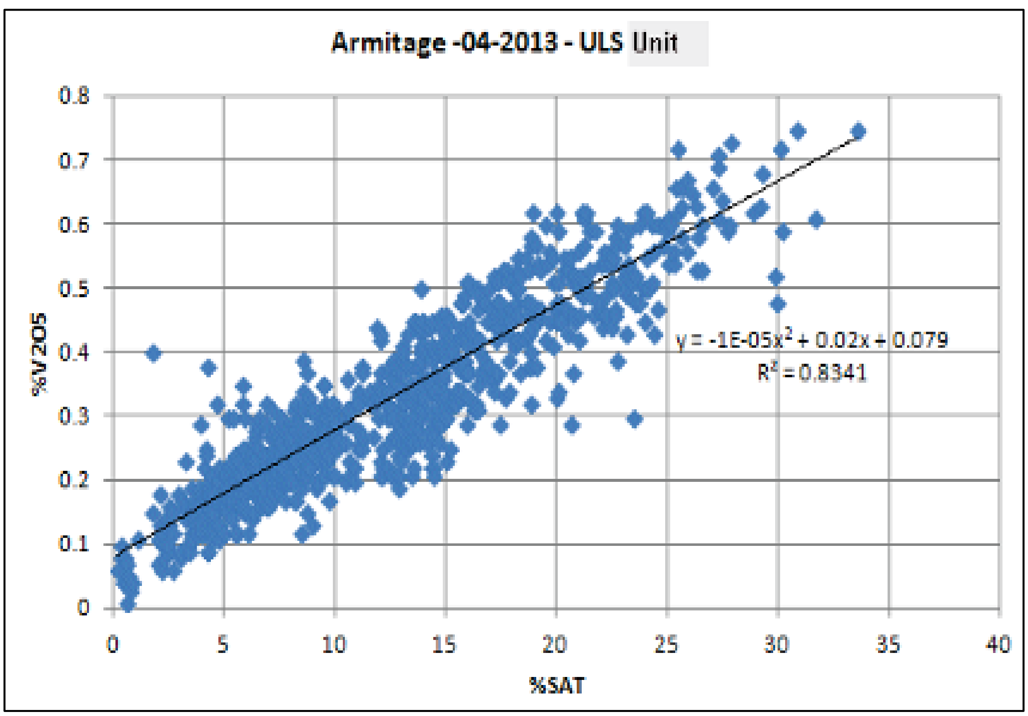


Figure 14-37: Correlation and regression of %V_2O_5 on %Satmagan. ULS Unit of Southwest in upper image and Armitage in lower image

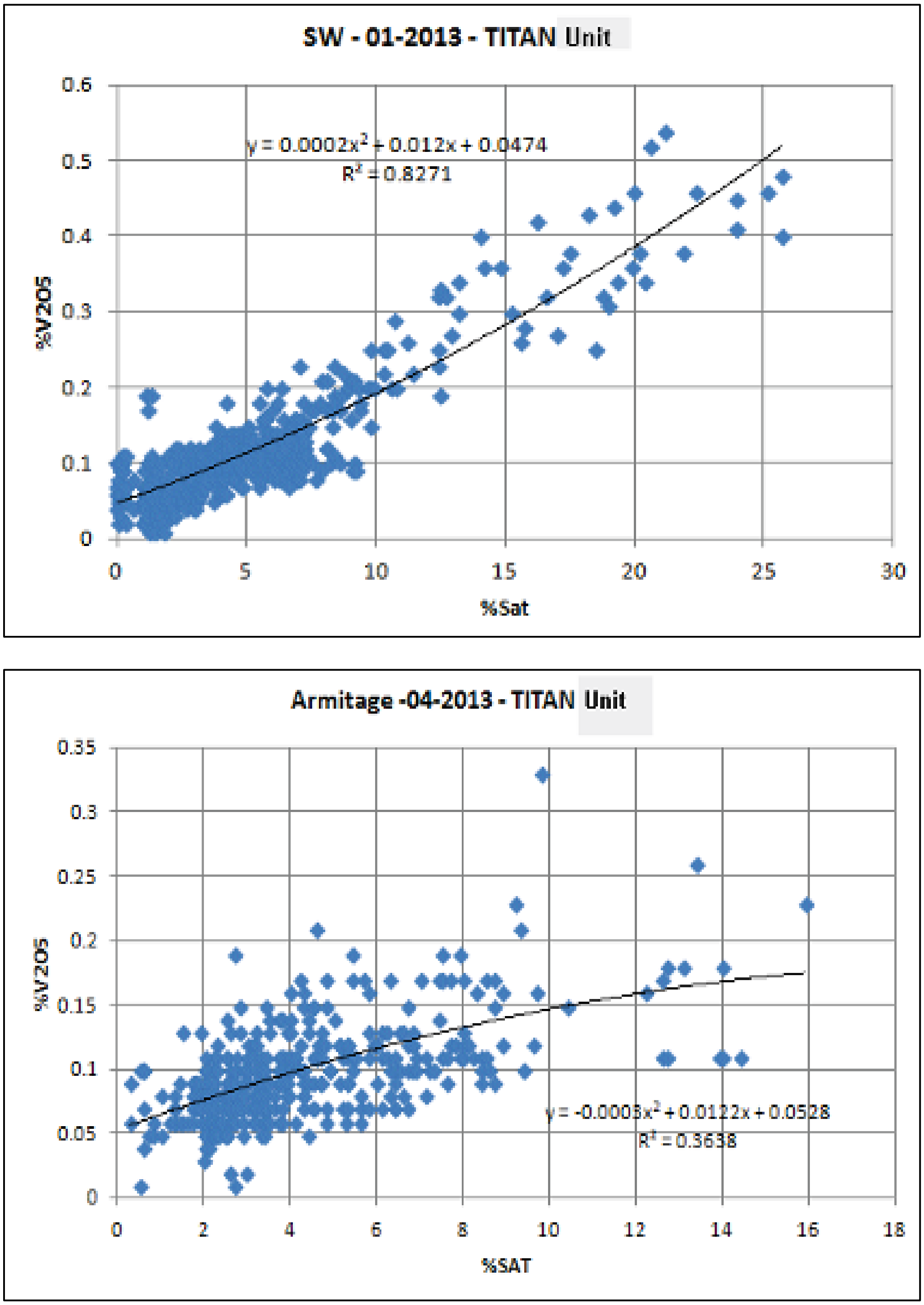


Figure 14-38: Correlation and regression of %V_2O_5 on %Satmagan. Titan unit of Southwest in upper image and Armitage in lower image

Statistics of final sample data used in block grade interpolation are compiled in Table 14-8. A comparison of values in the same unit, but for different deposits, shows the following similarities and differences:

- %Satmagan of BCS and Titan are fairly similar, with medium average grades around 12.5% in BCS and lower average grades of less than 6% in Titan;
- Averages in the MC (23% vs. 17%) and ULS (16% vs. 13%) are higher in Southwest, compared to Armitage;
- Variability of %Satmagan is high, particularly in BCS (coefficients of variation or CVs around 70%) and Titan (CVs above 60%);
- This high variability accounts for very low (zero values) and very high (from 30% to 40%) grades in all units, and reflects the layered nature of the deposits;
- Given the tight relationship of %Satmagan and %Fe_2O_3 (Figure 14-24 and Figure 14-25), there is a higher average %Fe_2O_3 in the Southwest MC and ULS, compared to the corresponding Armitage units. Compared to %Satmagan, the variability of %Fe_2O_3 is low (CVs from 18% to 40%);
- For %TiO_2, we continue to have higher average grades in the Southwest MC and ULS, as compared to the same two Armitage units, but differences tend to be smaller;
- Average %TiO_2 in the BCS and Titan of both deposits are rather similar (from 4.4% to 5.3%). Variability of %TiO_2 is moderate, with CVs from 32% to 53%;
- The average %V_2O_5 of the Southwest MC unit continues to be higher than the same Armitage MC unit (0.57% vs. 0.43%);
- Average %V_2O_5 of BCS and ULS are rather similar in both deposits (from 0.33% to 0.37%);
- Average %V_2O_5 of Titan is low in both deposits (0.10% and 0.13%);
- Variability of % V_2O_5 is moderate, with CVs from 33% to 55% (except Titan of Southwest, with 72%).

The spatial continuity of %Satmagan, %Fe_2O_3, %TiO_2 and %V_2O_5 in each of the four units of the two deposits have been investigated through the computation of experimental variograms. Variograms (correlograms) are computed in the principal dip, horizontal strike and cross dip+strike directions for each unit. That cross dip+strike direction is close to the average direction of drillholes.

In the Southwest deposit, the average dip direction dips 70° to N130, while the horizontal strike has a N40 azimuth. In the Armitage deposit, the average dip is 70° to N160, while the horizontal strike has a N70 azimuth. In both dip and strike, the lag distance is the nominal drillhole spacing of 100 m, while it is reduced to the 3 m sample length in the cross dip+strike direction. Angular tolerance is +/-20° in all directions. The experimental variograms are shown in Figure 14-39 to Figure 14-54.

Table 14-8: Statistics of sample data by unit and deposit

Variable	Number	Min.	Median	Average	Max.	%CV	Deposit	Unit
%Satmagan	1353	0	9.3	12.2	41.7	73.7	SW	BCS
%Satmagan	995	0.1	10.9	12.7	39.8	63.2	Armitage	BCS
%Satmagan	655	0	25.5	23.4	42.2	40.4	SW	MC
%Satmagan	593	0	17.4	16.9	39.7	44.6	Armitage	MC
%Satmagan	423	0	16.4	16.1	37.7	61.4	SW	ULS
%Satmagan	831	0.2	12.6	13.1	35.1	53.8	Armitage	ULS
%Satmagan	575	0	4.5	5.9	29.3	90.7	SW	Titan
%Satmagan	392	0.3	3.5	4.4	15.9	62.9	Armitage	Titan
%Fe_2O_3	1353	4.3	27.0	30.0	64.5	40.0	SW	BCS
%Fe_2O_3	995	7.4	28.9	30.7	59.5	33.7	Armitage	BCS
%Fe_2O_3	655	9.5	49.8	46.7	68.9	26.2	SW	MC
%Fe_2O_3	593	7.2	38.1	37.5	57.9	26.8	Armitage	MC
%Fe_2O_3	423	8.7	41.8	38.8	64.7	35.3	SW	ULS
%Fe_2O_3	831	2.3	35.5	35.3	58.7	26.0	Armitage	ULS
%Fe_2O_3	575	8.4	29.0	28.8	57.7	30.9	SW	Titan
%Fe_2O_3	392	5.4	25.8	26.4	39.3	18.5	Armitage	Titan
%TiO_2	1353	0.3	4.0	4.7	12.6	53.4	SW	BCS
%TiO_2	995	0.3	4.6	5.0	10.9	46.5	Armitage	BCS
%TiO_2	655	0.8	9.5	9.2	16.5	34.6	SW	MC
%TiO_2	593	0.7	7.5	7.3	13.2	33.5	Armitage	MC
%TiO_2	423	0.8	9.0	8.3	15.8	43.9	SW	ULS
%TiO_2	831	0.2	7.7	7.5	13.0	32.0	Armitage	ULS
%TiO_2	575	0.8	5.2	5.3	14.7	46.4	SW	Titan
%TiO_2	392	0.2	4.1	4.4	8.3	32.0	Armitage	Titan

Variable	Number	Min.	Median	Average	Max.	%CV	Deposit	Unit
$\%V_2O_5$	1353	0.02	0.32	0.36	0.91	54.7	SW	BCS
$\%V_2O_5$	995	0.01	0.35	0.37	0.94	44.7	Armitage	BCS
$\%V_2O_5$	655	0.03	0.62	0.57	0.96	32.7	SW	MC
$\%V_2O_5$	593	0.01	0.44	0.43	0.82	34.6	Armitage	MC
$\%V_2O_5$	423	0.02	0.34	0.35	0.84	54.1	SW	ULS
$\%V_2O_5$	831	0.01	0.31	0.33	0.75	44.1	Armitage	ULS
$\%V_2O_5$	575	0.01	0.10	0.13	0.58	72.1	SW	Titan
$\%V_2O_5$	392	0.01	0.09	0.10	0.33	39.3	Armitage	Titan
$\%Al_2O_3$	1067	7.1	16.6	16.4	26.8	20.8	SW	BCS
$\%Al_2O_3$	979	9.3	16.4	16.5	26.9	18.5	Armitage	BCS
$\%Al_2O_3$	556	5.6	11.0	11.6	23.6	27.0	SW	MC
$\%Al_2O_3$	585	8.4	13.3	13.8	24.1	18.2	Armitage	MC
$\%Al_2O_3$	352	5.9	11.4	12.3	24.0	30.8	SW	ULS
$\%Al_2O_3$	824	8.0	13.9	14.1	22.8	16.2	Armitage	ULS
$\%Al_2O_3$	461	7.5	10.8	11.5	23.0	21.4	SW	Titan
$\%Al_2O_3$	390	6.6	12.0	12.1	27.4	19.1	Armitage	Titan
$\%P_2O_5$	1067	0	0.020	0.039	0.36	113.1	SW	BCS
$\%P_2O_5$	979	0.008	0.020	0.031	0.26	98.3	Armitage	BCS
$\%P_2O_5$	556	0	0.020	0.031	0.24	107.5	SW	MC
$\%P_2O_5$	585	0.007	0.019	0.030	0.19	95.4	Armitage	MC
$\%P_2O_5$	352	0	0.030	0.040	0.28	96.2	SW	ULS
$\%P_2O_5$	824	0.007	0.020	0.020	0.15	84.8	Armitage	ULS
$\%P_2O_5$	461	0	0.020	0.043	0.30	100.0	SW	Titan
$\%P_2O_5$	390	0.010	0.022	0.031	0.33	78.8	Armitage	Titan
%S(1)	955	0.01	0.09	0.11	1.43	85.6	SW	BCS
%S	980	0.01	0.11	0.14	1.81	78.3	Armitage	BCS
%S	522	0.02	0.14	0.16	1.87	80.6	SW	MC
%S	585	0.04	0.22	0.26	1.57	63.4	Armitage	MC
%S	336	0.02	0.20	0.24	0.90	51.5	SW	ULS
%S	824	0.02	0.40	0.43	1.42	46.8	Armitage	ULS
%S	455	0	0.28	0.29	1.48	51.7	SW	Titan
%S	390	0.01	0.36	0.37	1.29	42.0	Armitage	Titan

(1) Excludes outlier of 6.41%S

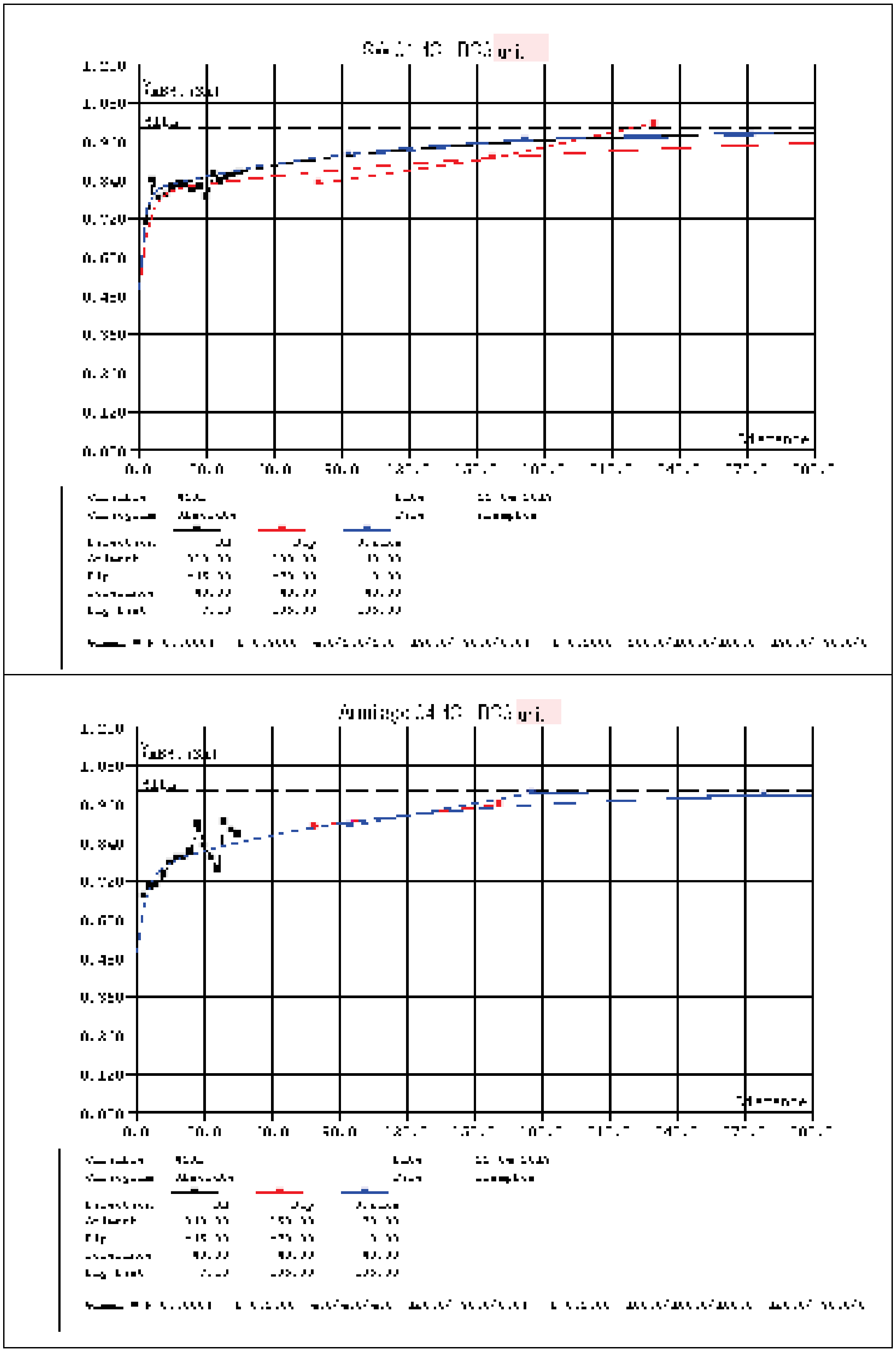


Figure 14-39: Variograms of %Satmagan grade in the BCS unit. Southwest is shown in the upper graph and Armitage in the lower graph

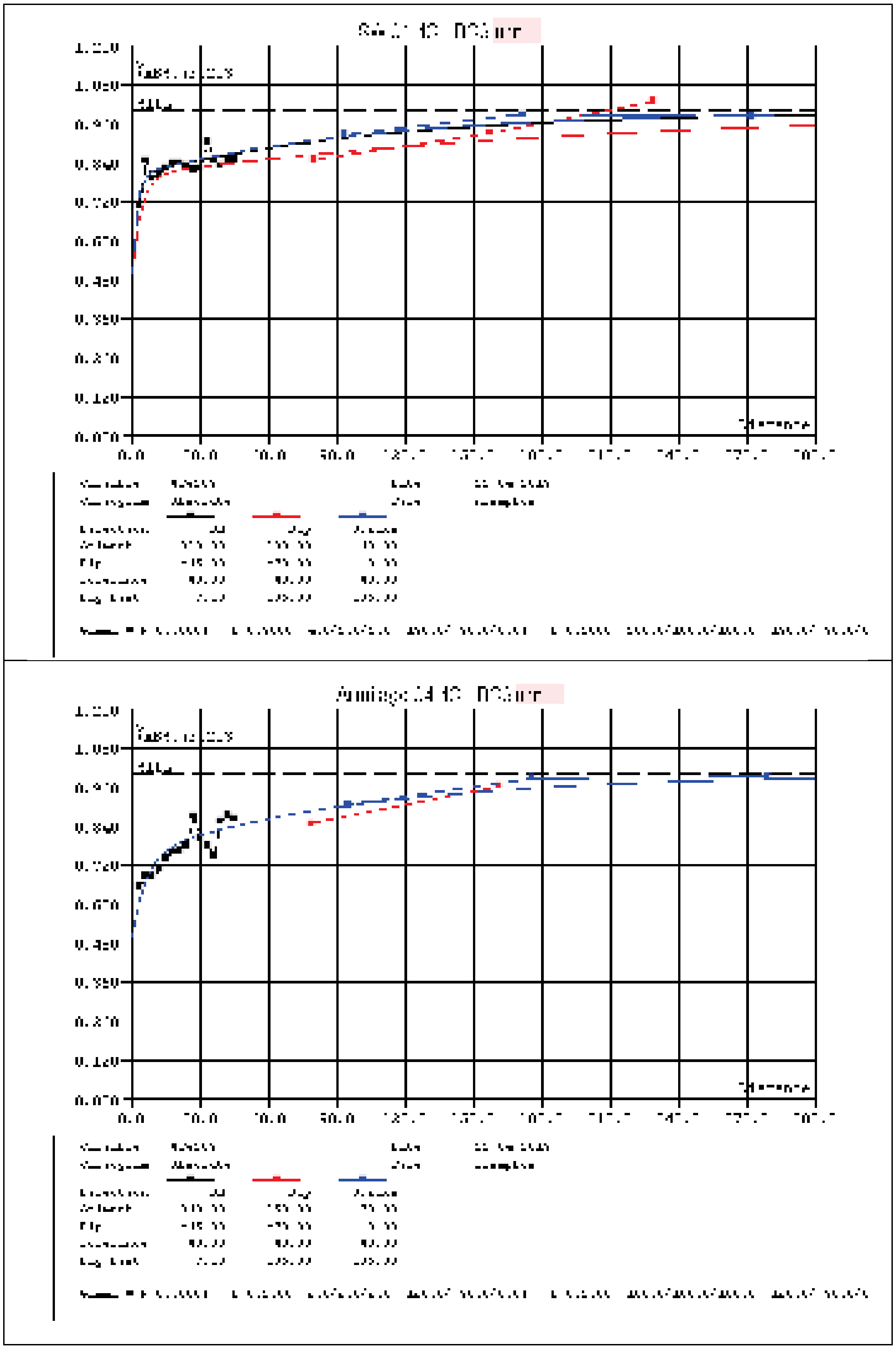


Figure 14-40: Variograms of %Fe_2O_3 Grade in the BCS unit. Southwest is shown in the upper graph and Armitage in the lower graph

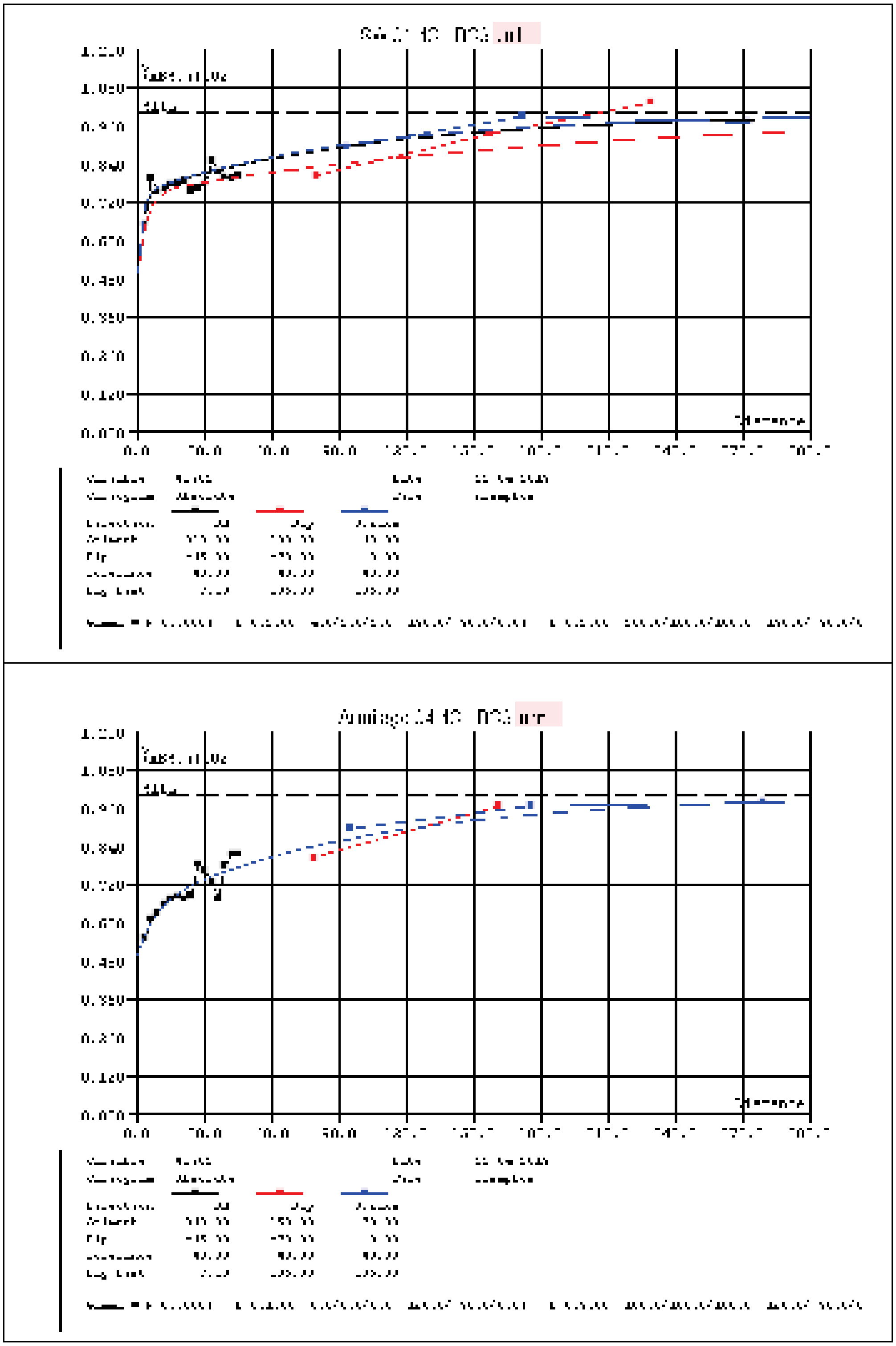


Figure 14-41: Variograms of %TiO_2 grade in the BCS unit. Southwest is shown in the upper graph and Armitage in the lower graph

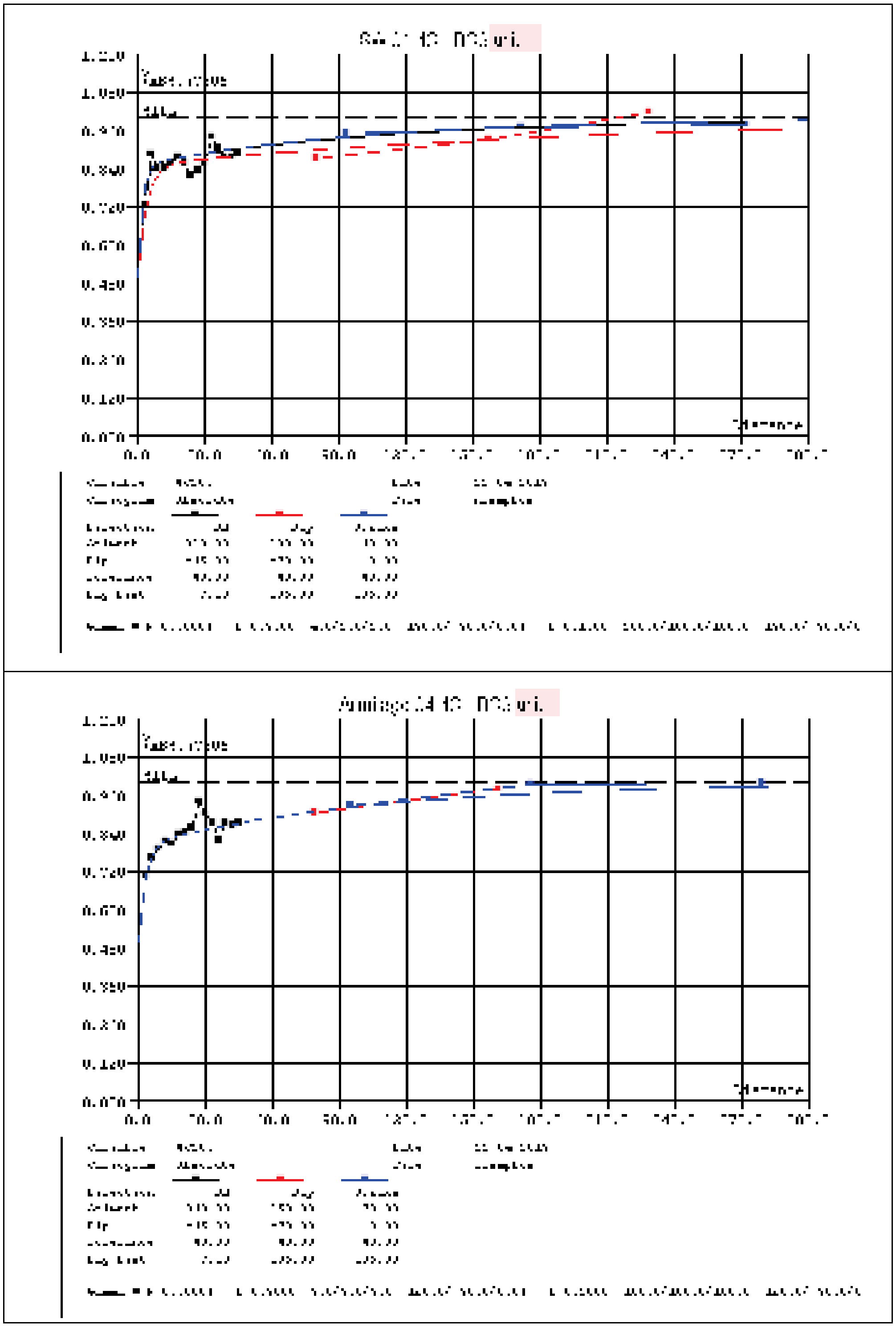


Figure 14-42: Variograms of $\%V_2O_5$ Grade in the BCS Unit. Southwest is shown in the upper graph and Armitage in the lower graph

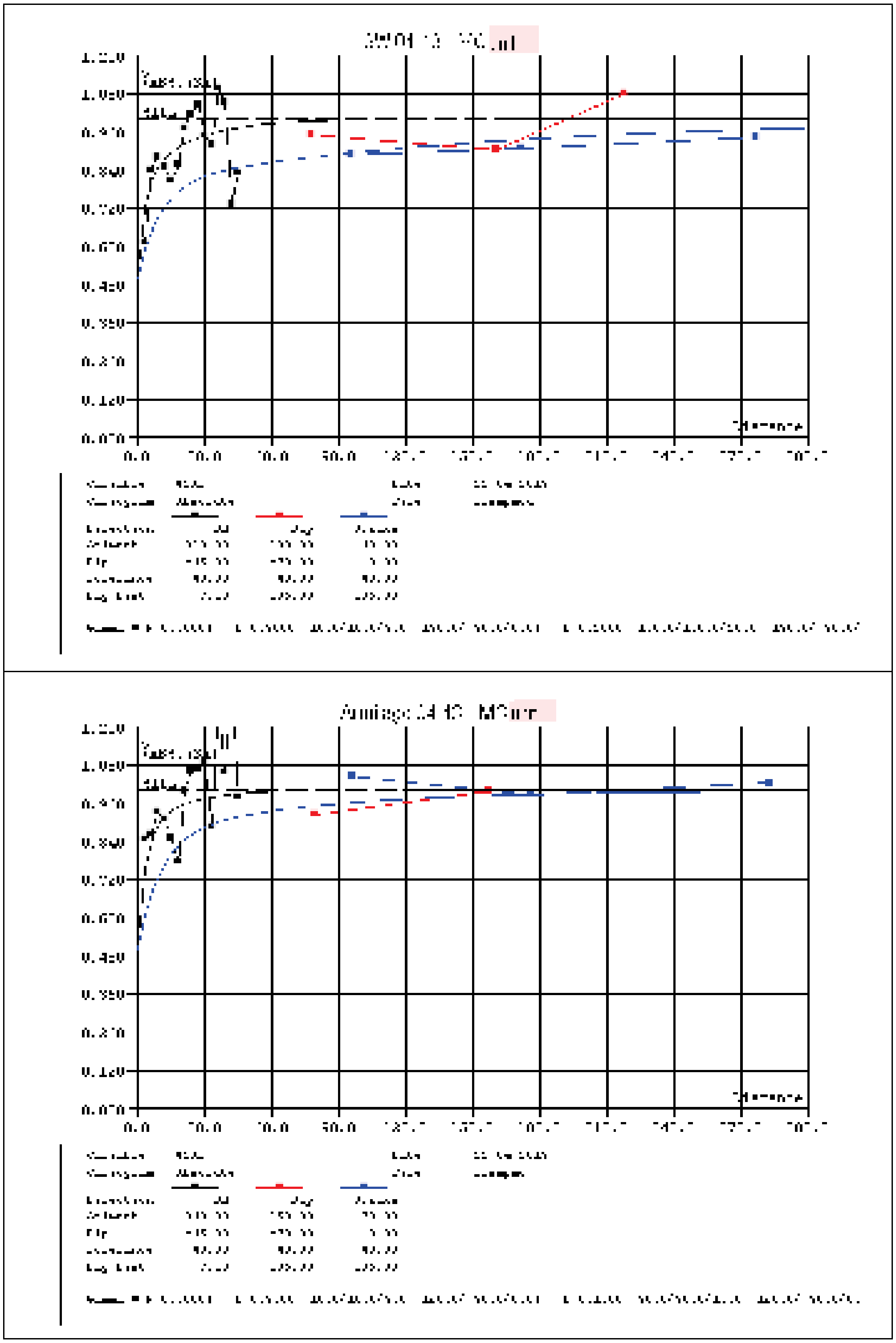


Figure 14-43: Variograms of %Satmagan Grade in the MC Unit. Southwest is shown in the upper graph and Armitage in the lower graph

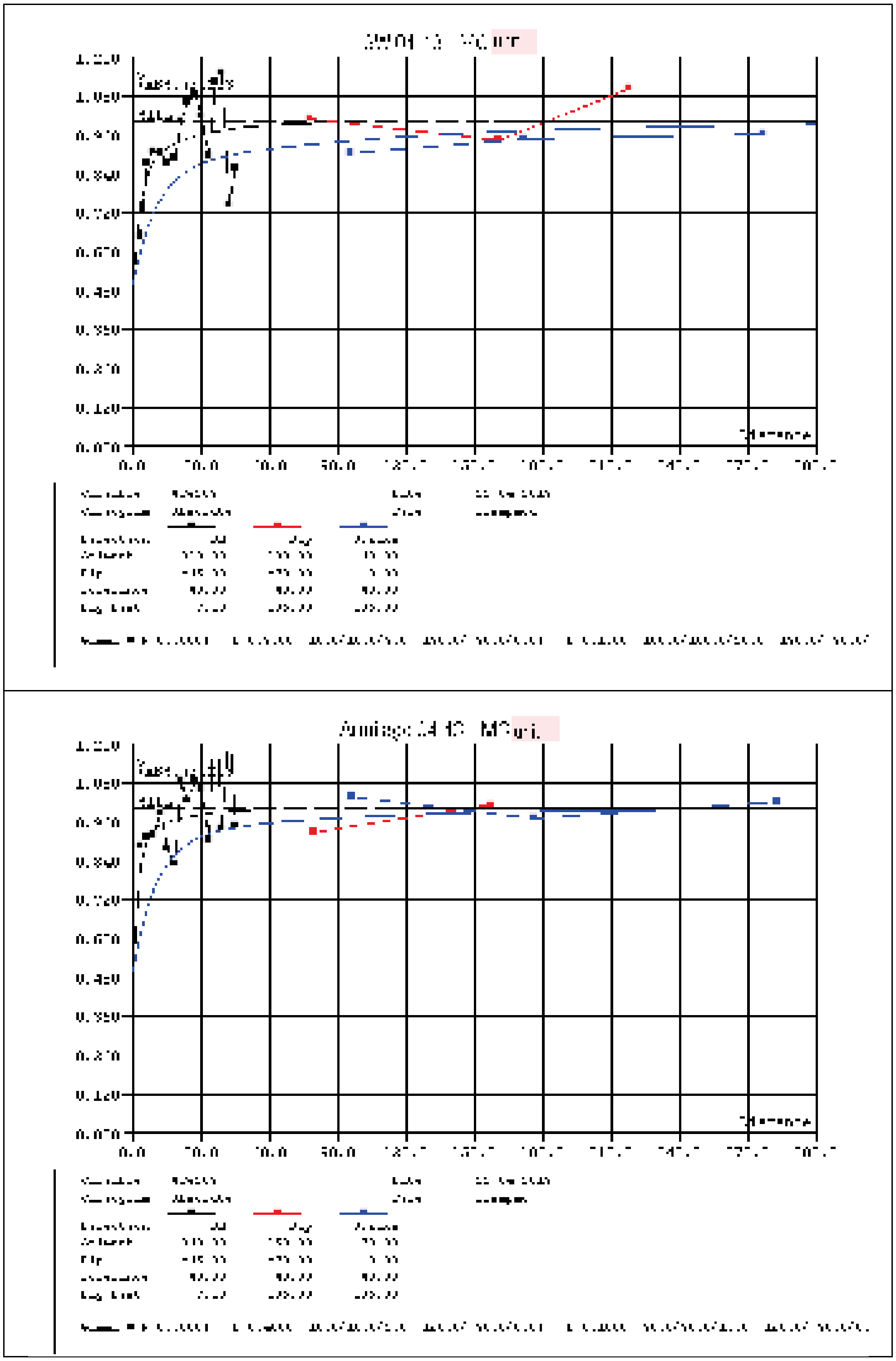


Figure 14-44: Variograms of %Fe_2O_3 Grade in the MC Unit. Southwest is shown in the upper graph and Armitage in the lower graph

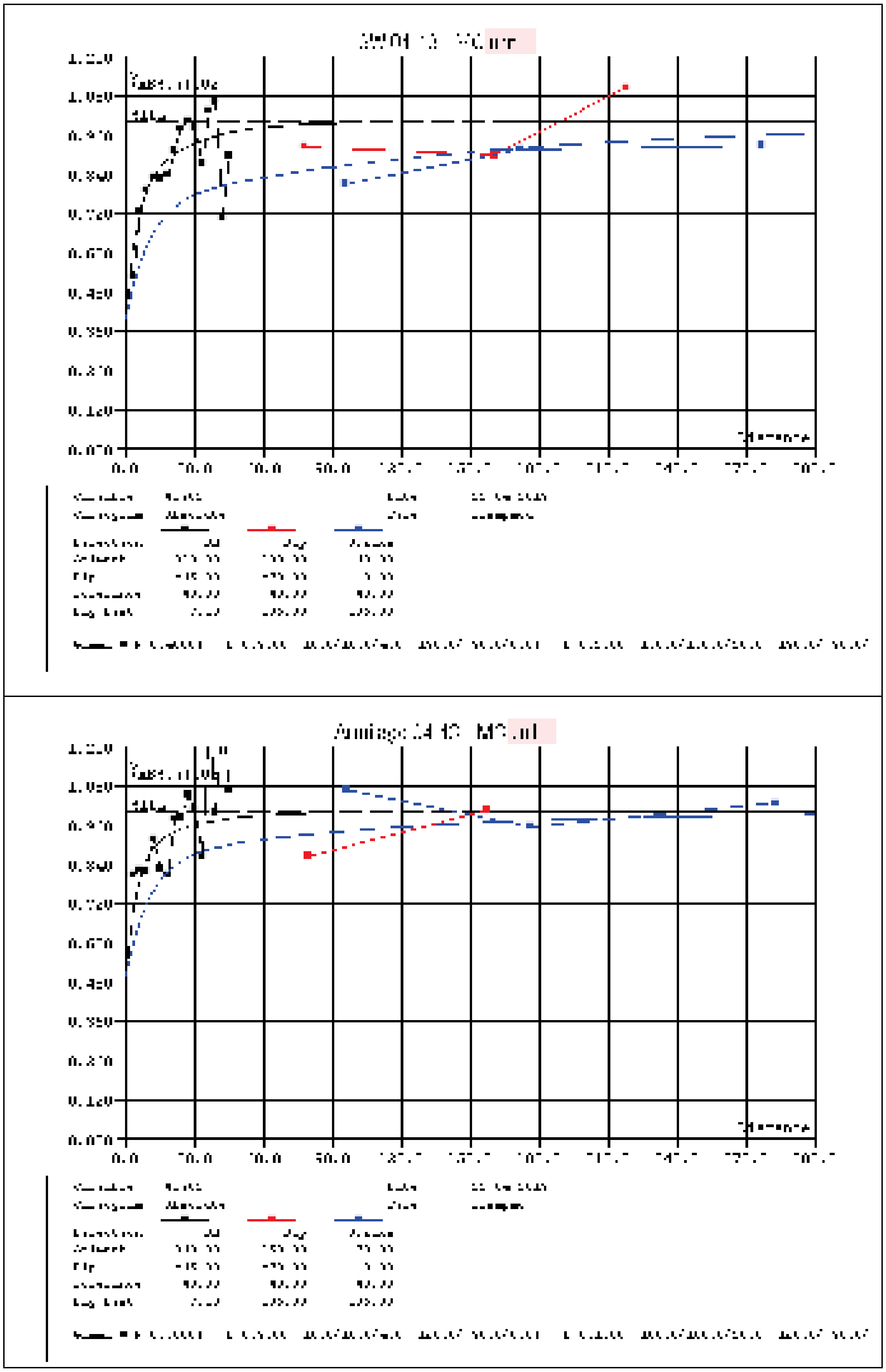


Figure 14-45: Variograms of %TiO_2 Grade in the MC Unit. Southwest is shown in the upper graph and Armitage in the lower graph

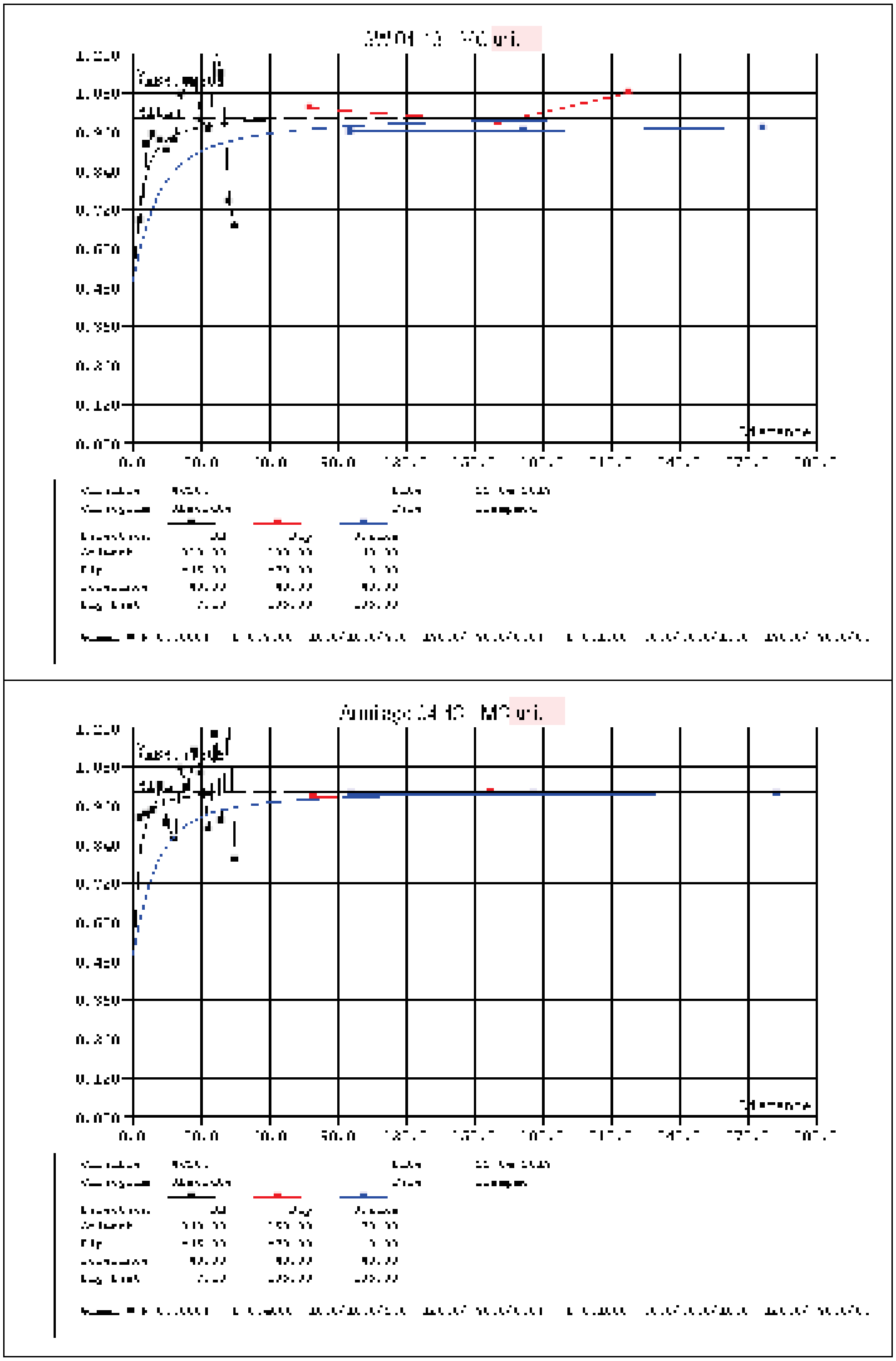


Figure 14-46: Variograms of %V_2O_5 Grade in the MC Unit. Southwest is shown in the upper graph and Armitage in the lower graph

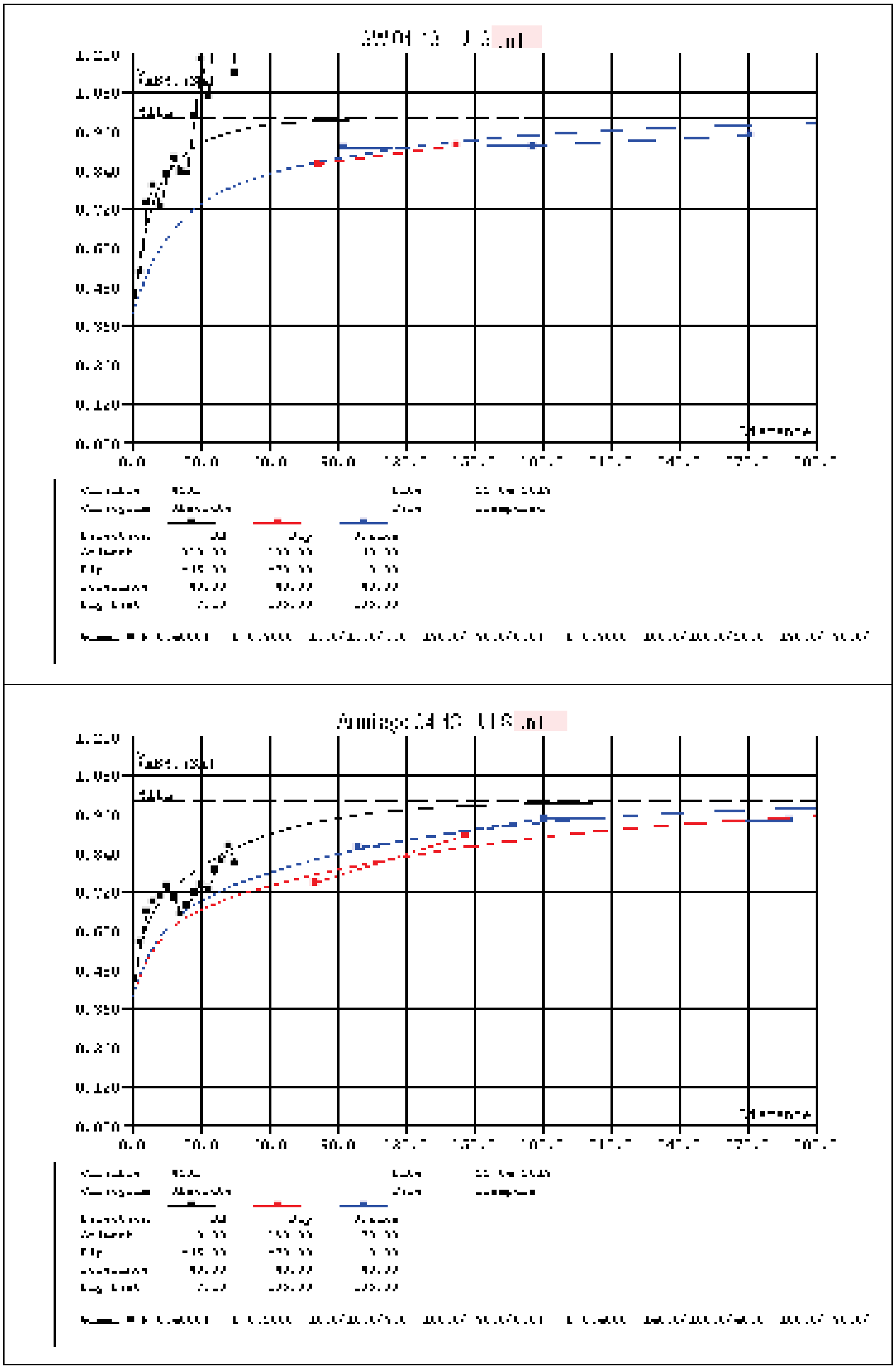


Figure 14-47: Variograms of %Satmagan Grade in the ULS Unit. Southwest is shown in the upper graph and Armitage in the lower graph

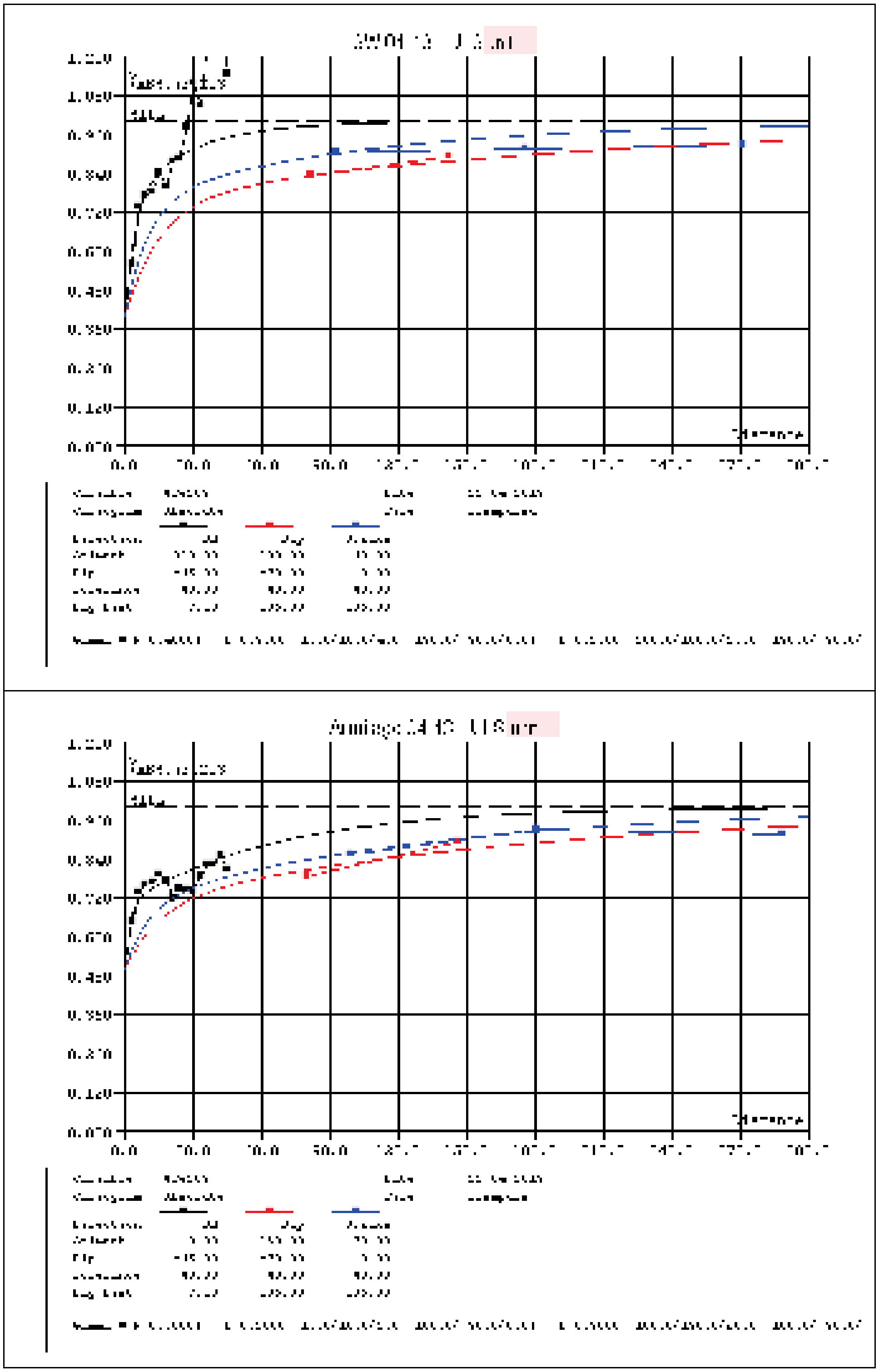


Figure 14-48: Variograms of %Fe_2O_3 Grade in the ULS Unit. Southwest is shown in the upper graph and Armitage in the lower graph

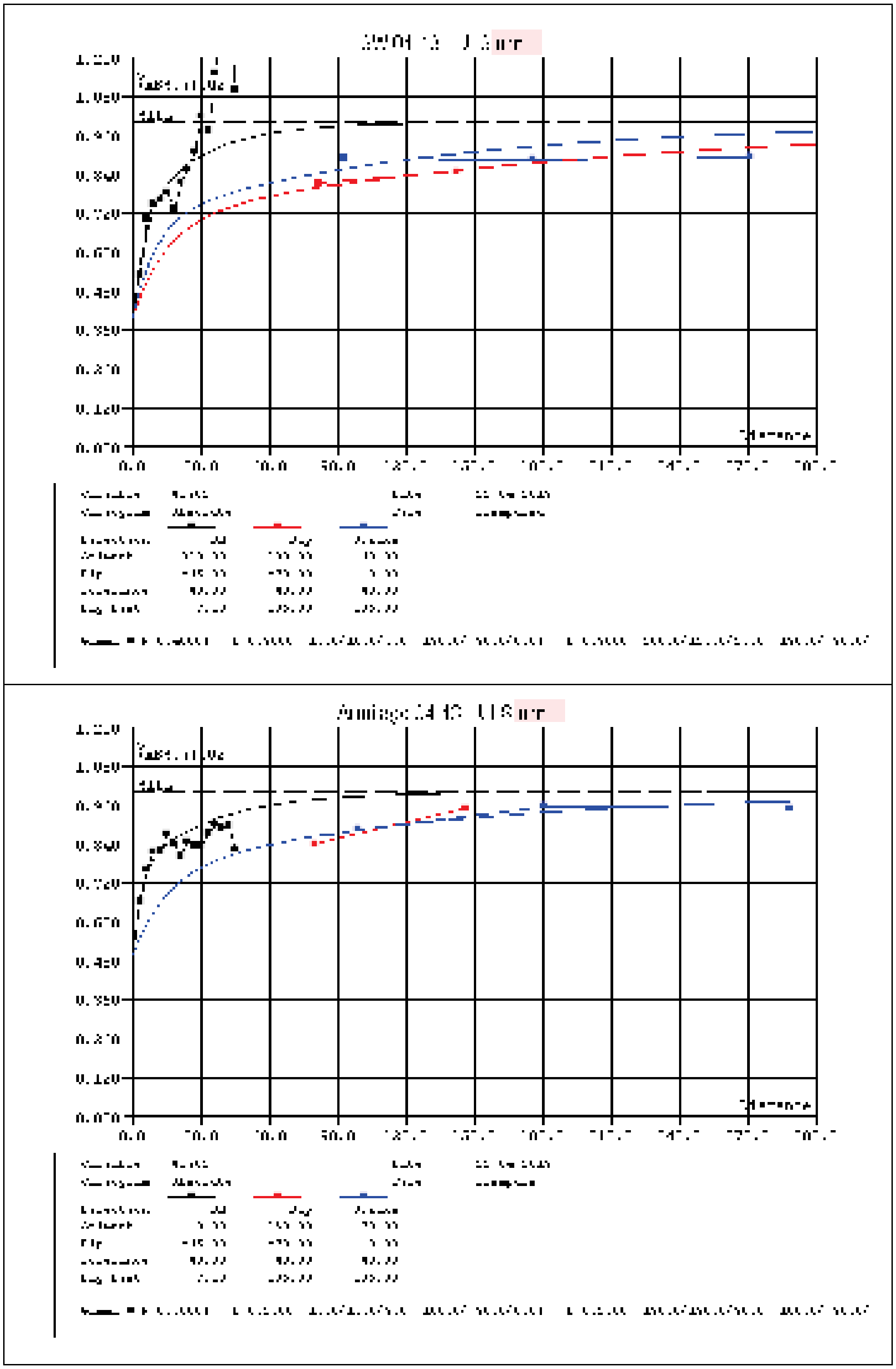


Figure 14-49: Variograms of %TiO_2 Grade in the ULS Unit. Southwest is shown in the upper graph and Armitage in the lower graph

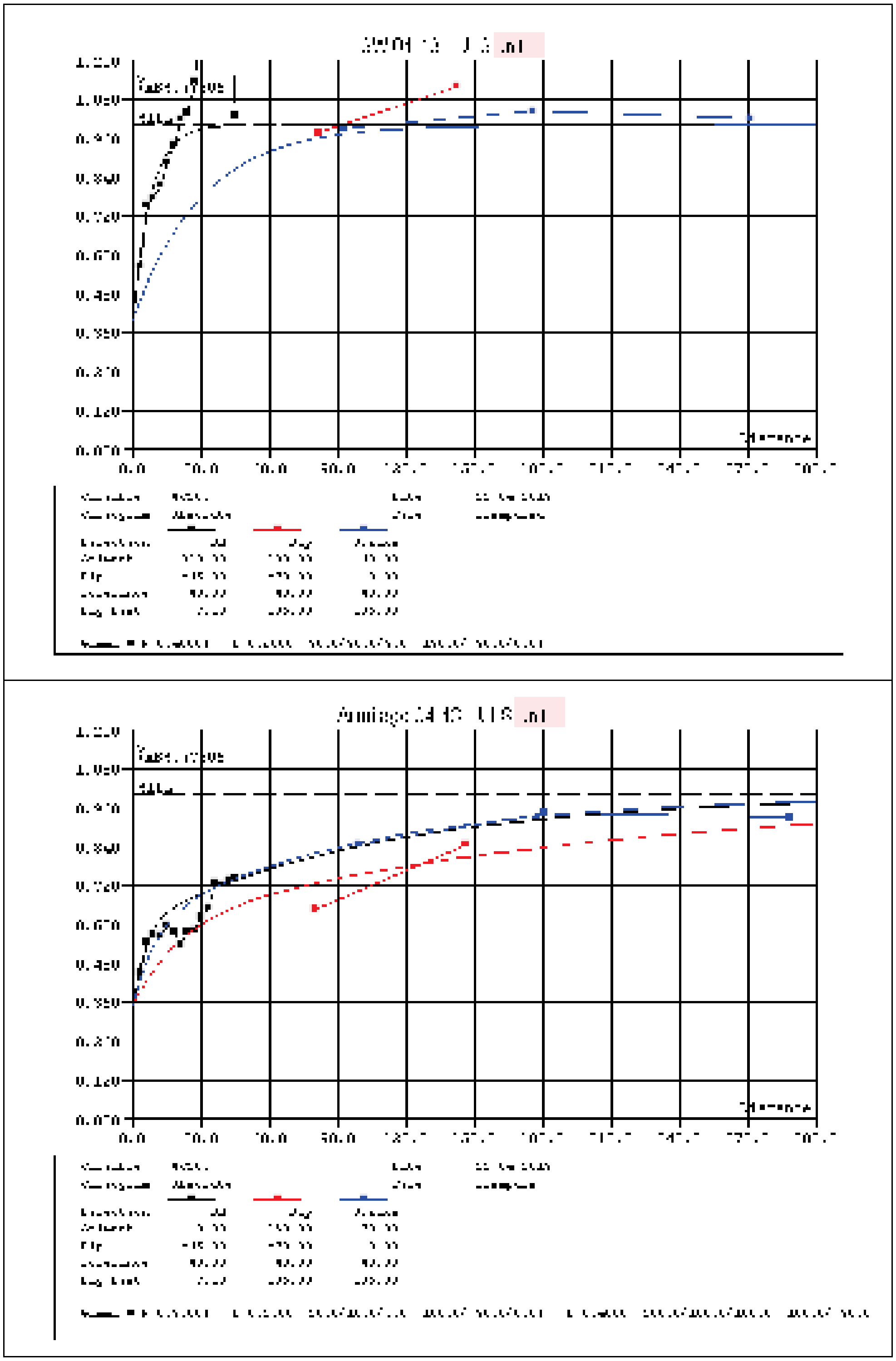


Figure 14-50: Variograms of %V_2O_5 Grade in the ULS Unit. Southwest is shown in the upper graph and Armitage in the lower graph

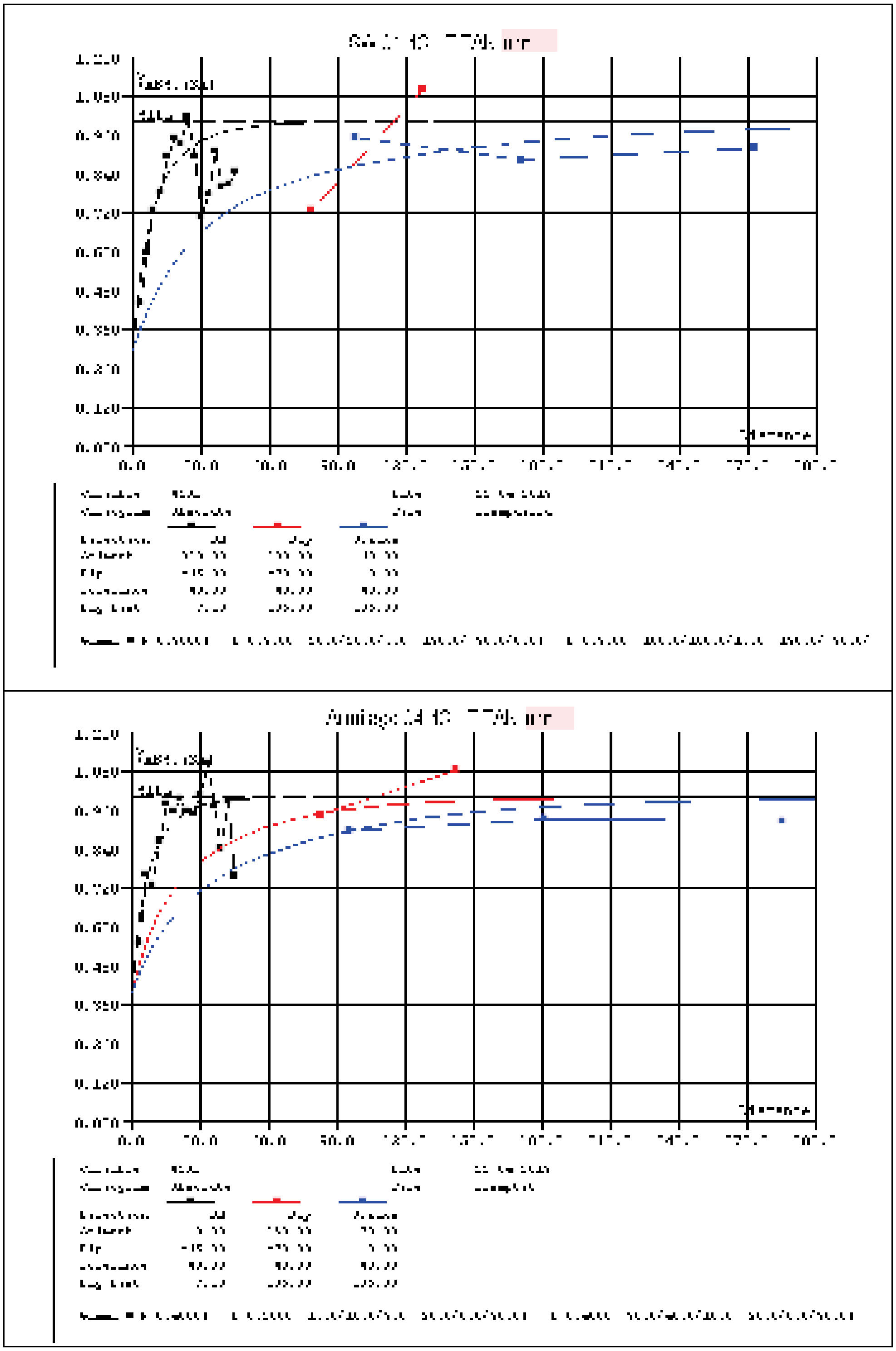


Figure 14-51: Variograms of %Satmagan Grade in the Titan Unit. Southwest is shown in the upper graph and Armitage in the lower graph

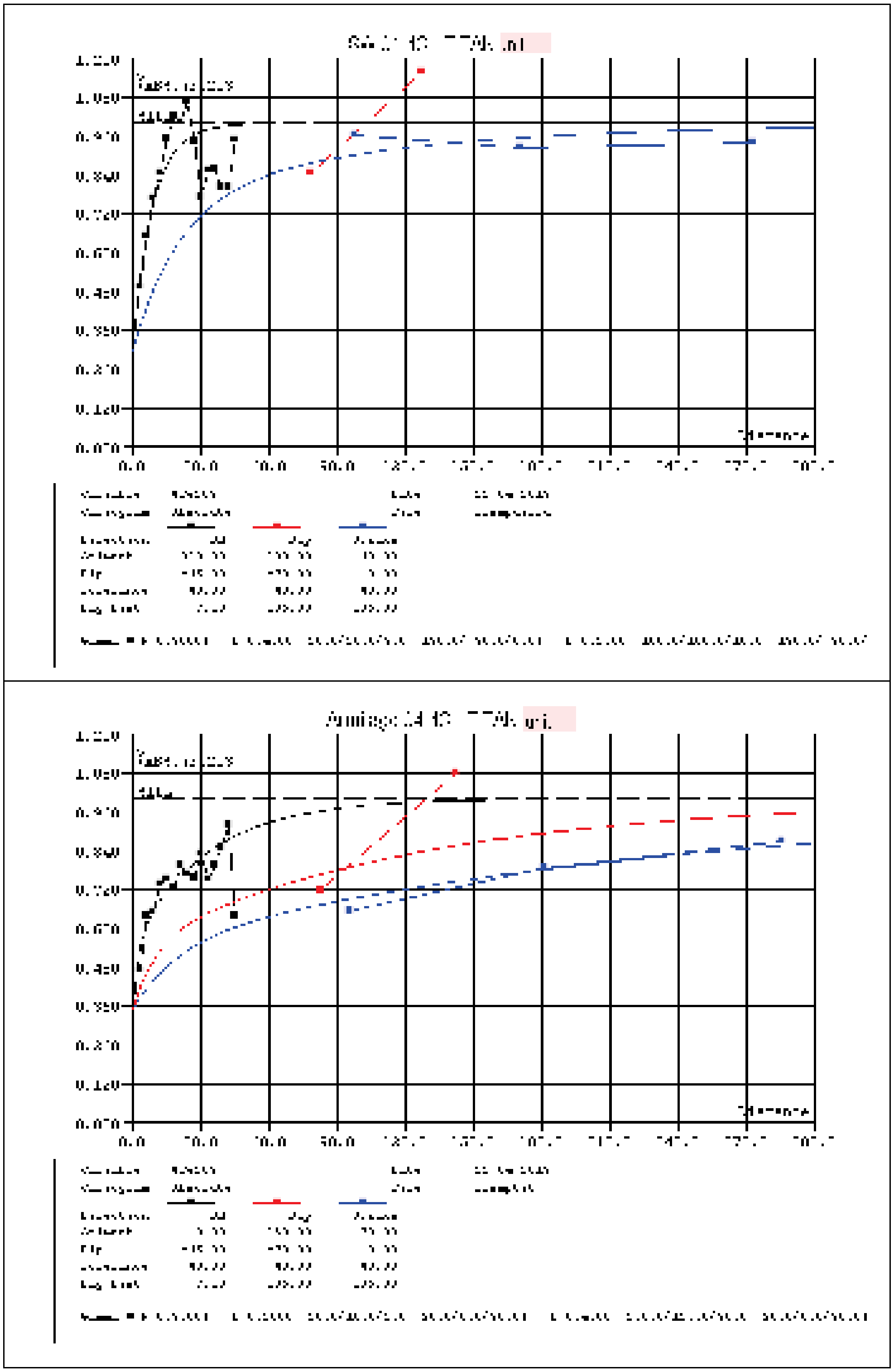


Figure 14-52: Variograms of %Fe_2O_3 Grade in the Titan Unit. Southwest is shown in the upper graph and Armitage in the lower graph

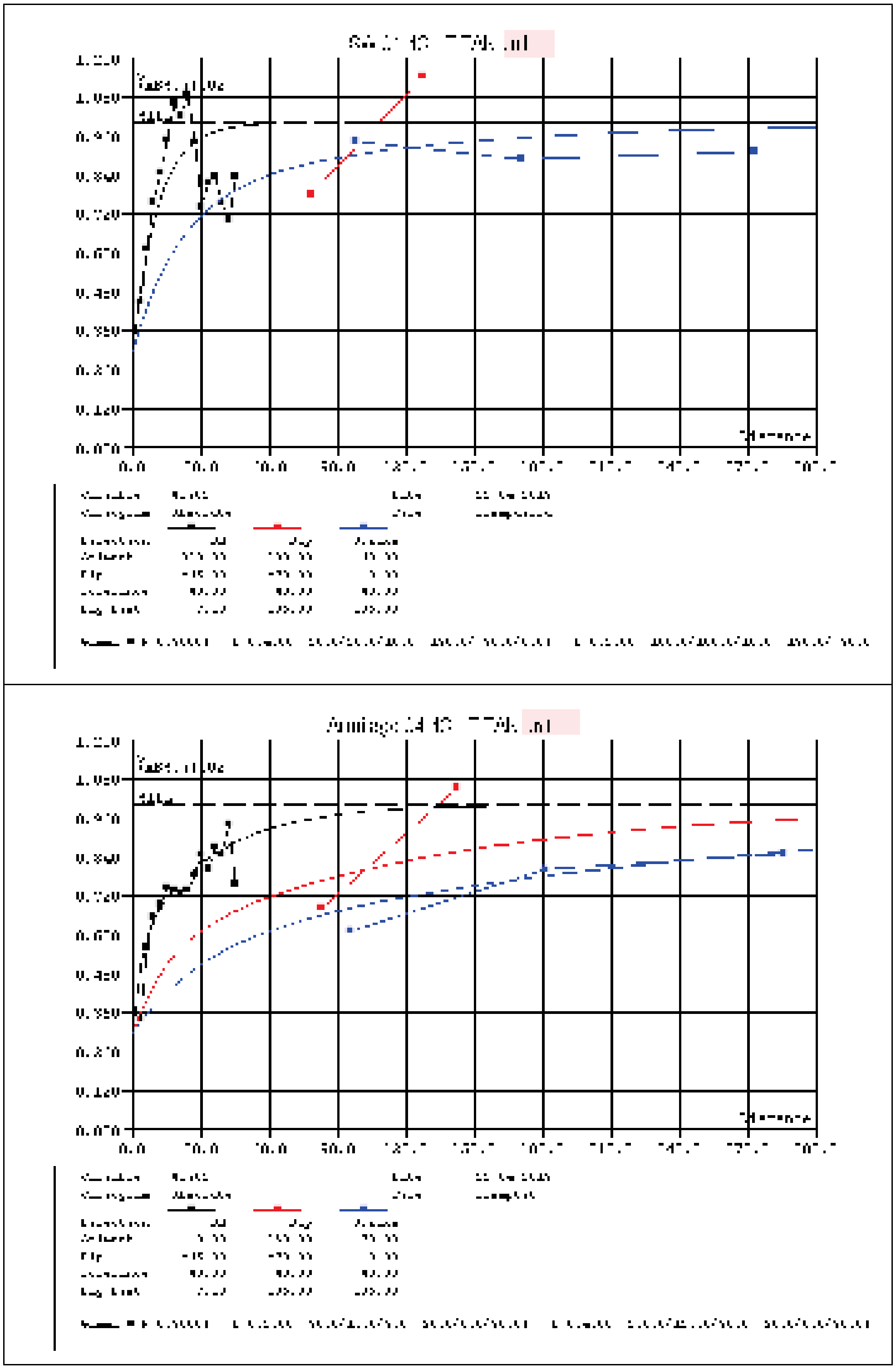


Figure 14-53: Variograms of %TiO_2 Grade in the Titan Unit. Southwest is shown in the upper graph and Armitage in the lower graph

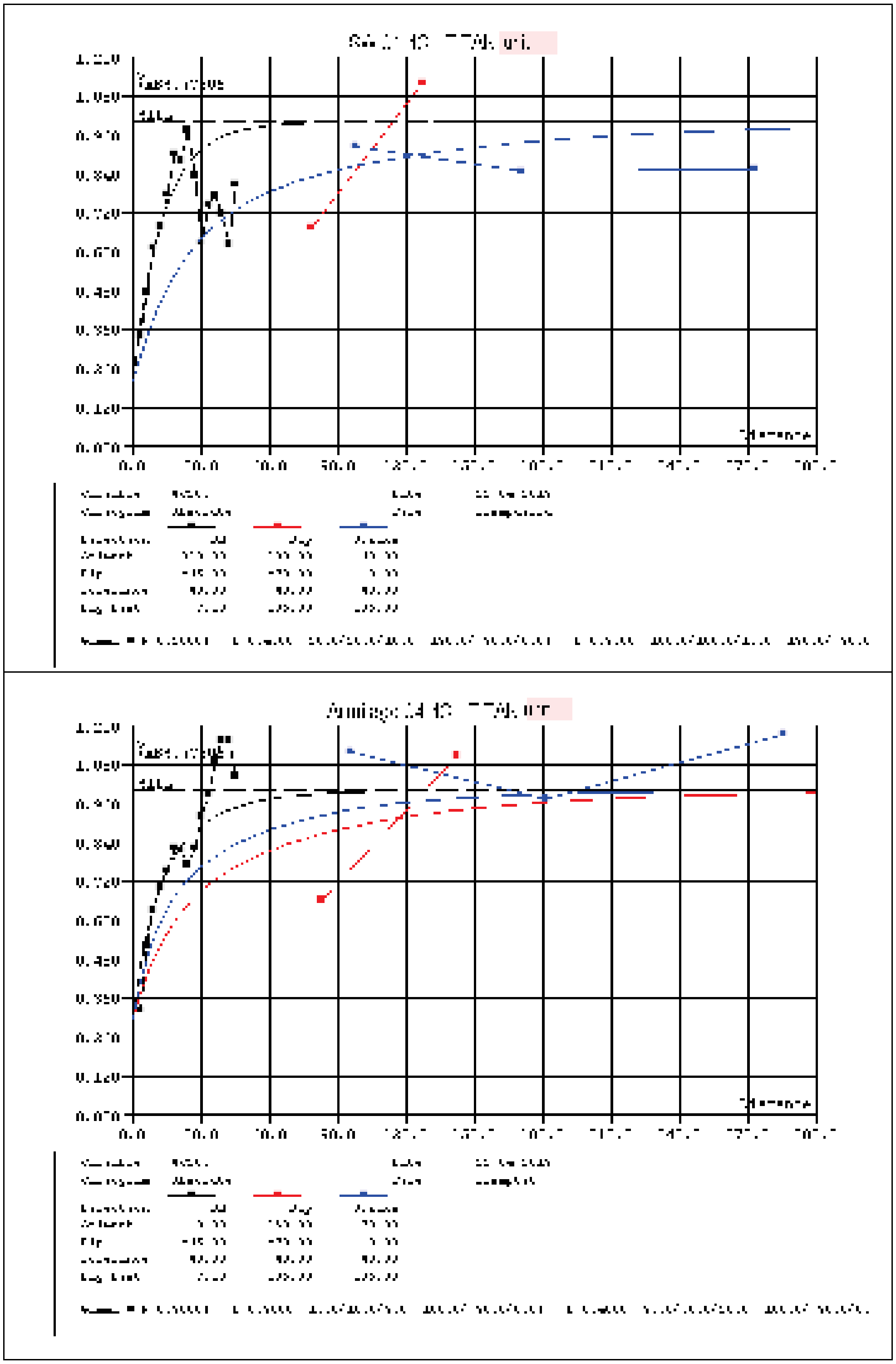


Figure 14-54: Variograms of %V_2O_5 Grade in the Titan Unit. Southwest is shown in the upper graph and Armitage in the lower graph

The variograms within each unit are fairly similar, reflecting the tight interrelationship between the four principal variables. They are all characterized by a significant nugget effect (from 30% to 50%) and a first short-range structure (between 6 m and 90 m), plus a second long-range structure (between 30 m and 750 m). The long range is about the same in the dip and strike directions, but definitely shorter across dip+strike. This finding is consistent with the deposit type and controls on ore deposition (see Chapter 8). Variogram model parameters are listed in Table 14-9.

Table 14-9: Parameters of variogram models

Dep.	Unit	Var	C0	C1	A1 (m)			C2	A2 (m)		
					Dip	Strike	X		Dip	Strike	X
Southwest	BCS	%Sat	0.5	0.3	12	6	6	0.2	450	200	200
Southwest	BCS	%Fe_2O_3	0.5	0.3	12	6	6	0.2	600	300	300
Southwest	BCS	%TiO_2	0.5	0.25	12	6	6	0.25	600	300	300
Southwest	BCS	%V_2O_5	0.5	0.35	12	6	6	0.15	600	300	300
Southwest	MC	%Sat	0.5	0.3	30	30	9	0.2	120	120	30
Southwest	MC	%Fe_2O_3	0.5	0.35	30	30	9	0.15	300	300	60
Southwest	MC	%TiO_2	0.4	0.35	30	30	12	0.25	450	450	60
Southwest	MC	%V_2O_5	0.5	0.35	30	30	9	0.15	150	150	45
Southwest	ULS	%Sat	0.4	0.3	45	45	15	0.30	200	200	45
Southwest	ULS	%Fe_2O_3	0.4	0.35	45	30	12	0.25	600	300	75
Southwest	ULS	%TiO_2	0.4	0.3	45	30	15	0.3	600	375	75
Southwest	ULS	%V_2O_5	0.4	0.3	90	90	21	0.3	90	90	21
Southwest	Titan	%Sat	0.3	0.35	60	60	30	0.35	300	300	45
Southwest	Titan	%Fe_2O_3	0.3	0.45	60	60	21	0.25	300	300	30
Southwest	Titan	%TiO_2	0.3	0.45	60	60	30	0.25	300	300	30
Southwest	Titan	%V_2O_5	0.2	0.45	60	60	30	0.35	300	300	45
Armitage	BCS	%Sat	0.5	0.25	12	12	12	0.25	300	300	300
Armitage	BCS	%Fe_2O_3	0.5	0.25	18	18	18	0.25	300	300	300
Armitage	BCS	%TiO_2	0.5	0.15	24	24	24	0.35	300	300	300
Armitage	BCS	%V_2O_5	0.5	0.3	9	9	9	0.2	300	300	300

Dep.	Unit	Var	C0	C1	A1 (m)			C2	A2 (m)		
					Dip	Strike	X		Dip	Strike	X
Armitage	MC	%Sat	0.5	0.35	30	30	9	0.15	210	210	45
Armitage	MC	%Fe_2O_3	0.5	0.4	30	30	6	0.1	210	210	45
Armitage	MC	%TiO_2	0.5	0.35	30	30	12	0.15	300	300	60
Armitage	MC	%V_2O_5	0.5	0.4	30	30	6	0.1	150	150	30
Armitage	ULS	%Sat	0.4	0.2	30	30	9	0.4	520	360	150
Armitage	ULS	%Fe_2O_3	0.5	0.2	45	30	6	0.3	540	390	180
Armitage	ULS	%TiO_2	0.5	0.25	45	45	9	0.25	390	390	90
Armitage	ULS	%V_2O_5	0.35	0.25	60	30	15	0.4	600	300	300
Armitage	Titan	%Sat	0.4	0.2	30	45	9	0.4	120	210	30
Armitage	Titan	%Fe_2O_3	0.35	0.2	30	60	6	0.45	375	750	90
Armitage	Titan	%TiO_2	0.30	0.25	45	90	9	0.45	375	750	90
Armitage	Titan	%V_2O_5	0.30	0.30	45	30	21	0.4	225	150	60

C0 is the nugget effect. C1 is the magnitude of a first exponential structure with range A1 (three times the distance parameter of the exponential function). C2 is the magnitude of a first exponential structure with range A2. The sum of C0+C1+C2 is always scaled to 1 (corresponding to a correlation coefficient of zero).

14.5. Mineralized Block Grade Interpolation

Interpolations of grade information into the blocks of both deposits were treated in a similar fashion. Initially, an ordinary kriging interpolation (OK) was completed, but the kriged block estimates were found to be overly diluted with respect to the assay data, as a result of the significant nugget effect in the variograms (see Section 14.6) below for a more detailed discussion). The average %Satmagan (magnetite), %Fe_2O_3, %TiO_2 and %V_2O_5 of each 10 x 5 x 7 m block in the mineralized solids was then interpolated by inverse squared distance squared (ID2) from reported values of neighbor 3 m samples in the same mineralized unit.

Despite some minor differences in variogram models, the search parameters are about the same for all variables in all units and domains. This is because the search strategy is mostly a function of the nominal 100 x 100 x 3 m sample grid with the 100 x 100 m mesh aligned with the average dip of the mineralized zone in that domain. In any given unit and domain, the first run uses a

125 x 125 x 50 m ellipsoid with the 125 m circle in the average plane of the domain fraction of the unit. Orientations of those domain fractions for both deposits are listed in Table 14-10.

Table 14-10: Orientation of the principal plane of search ellipsoids

Deposit	Unit	Domain	Azimuth (°)	Dip (°)
Southwest	All	1	N130	45
Southwest	All	2	N130	70
Southwest	All	3	N130	55
Southwest	All	4	N130	72
Southwest	All	5	N130	72
Southwest	All	6	N130	83
Southwest	All	7	N130	70
Armitage	All	S1=1	N160	80
Armitage	All	S2=2	N160	60
Armitage	All	S3=3+4+6+7+9+10+12	N160	70
Armitage	All	S4=5+8	N170	70
Armitage	All	S5=11+13	N150	85

A dip of 45° to an azimuth of N130 corresponds to a N40 (or N220) horizontal strike.

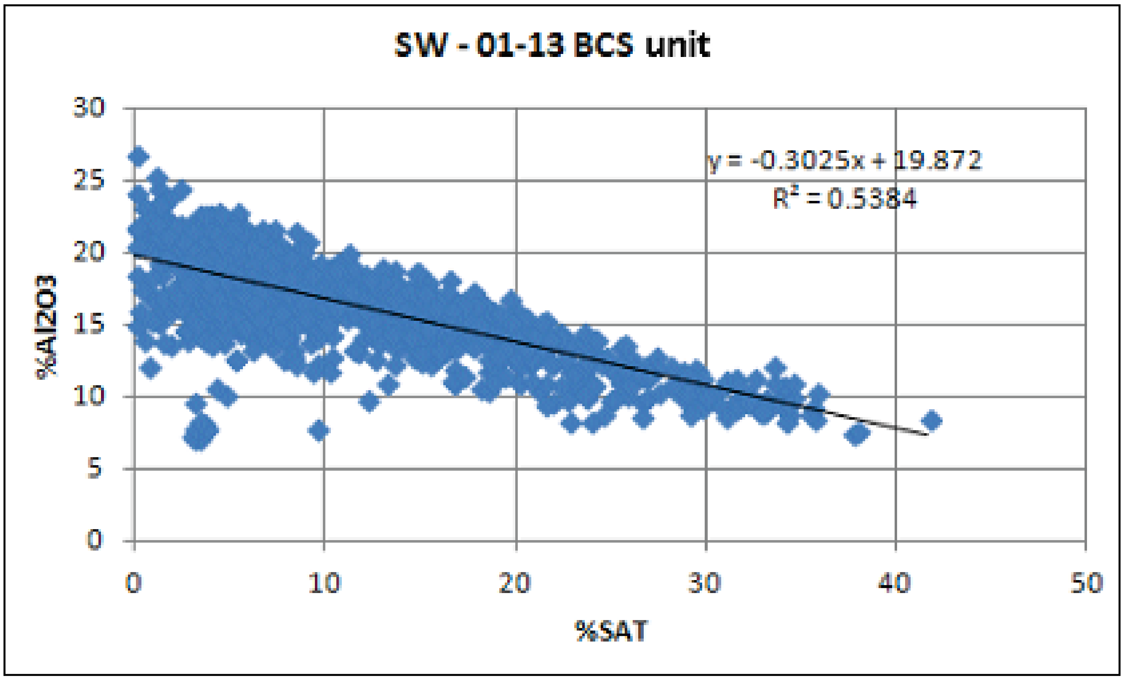
For the block, a minimum of five samples in a minimum of three holes (maximum of samples in the same hole is set to 2) is required for interpolation in that run. The maximum number of samples kept in the ellipsoid is limited to the ten closest to the centroid.

Blocks lacking enough information may be interpolated in a second run with a 250 x 250 x 100 m ellipsoid and with the same restrictions for the minimum number of samples and holes, but the maximum number of contributing samples is raised to 15.

Any remaining not estimated blocks, if any, are interpolated in a final run using a 500 x 500 x 200 m ellipsoid, where the maximum number of samples is raised to 20. Blocks are discretized according to a 4 (RX) x 2 (RY) x 3 (Z) grid.

The additional variables $\%P_2O_5$ and %S are also interpolated through a similar inverse squared distance (ID2) procedure.

Finally, $\%Al_2O_3$ is estimated through its residual in linear regression models over %Satmagan derived from all sample data in each zone and each deposit (Figure 14-55). This is the same procedure as the one used in previous 2010 and 2011 models.



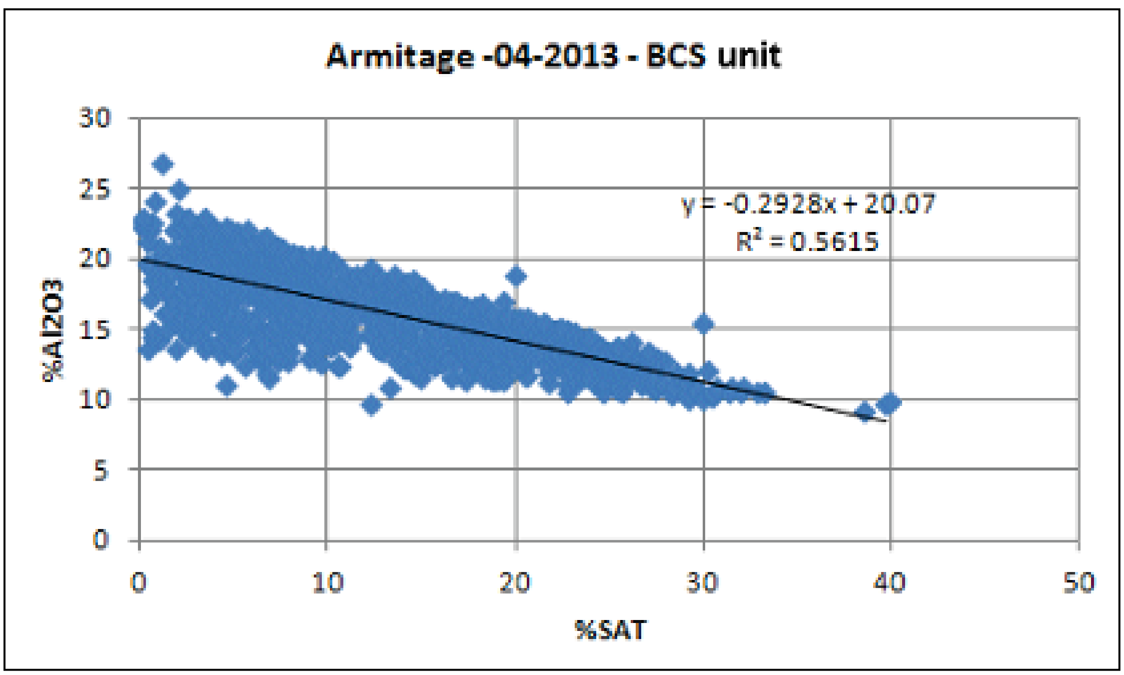
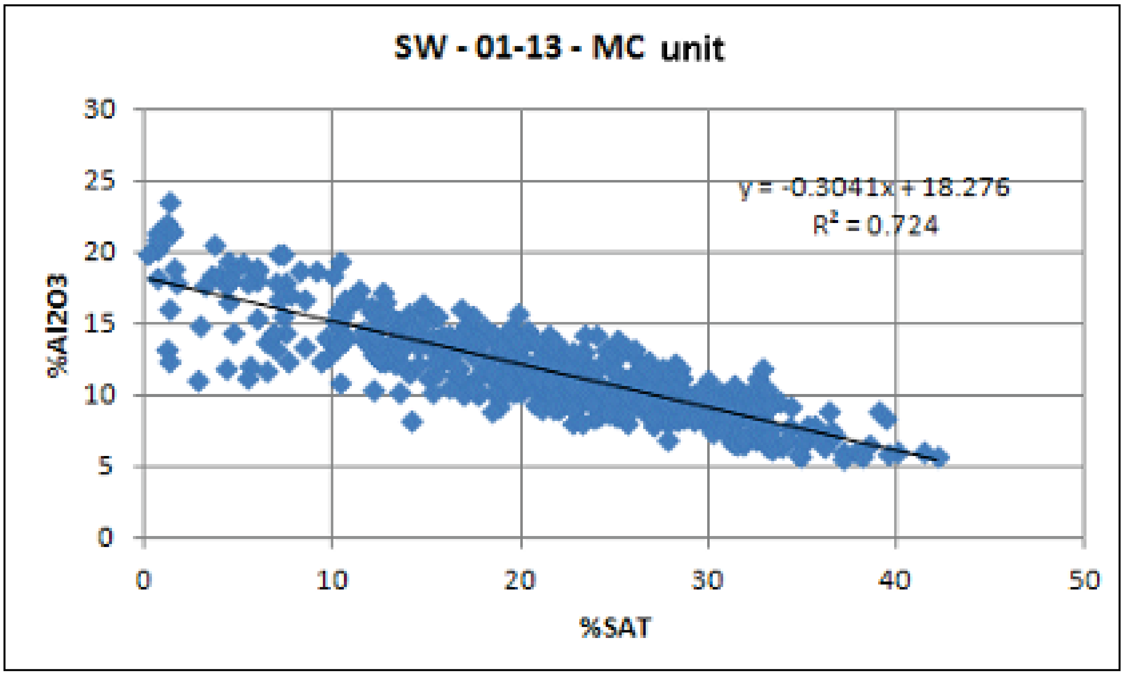


Figure 14-55: Correlation and regression of %Al2O3 over %Satmagan for BCS



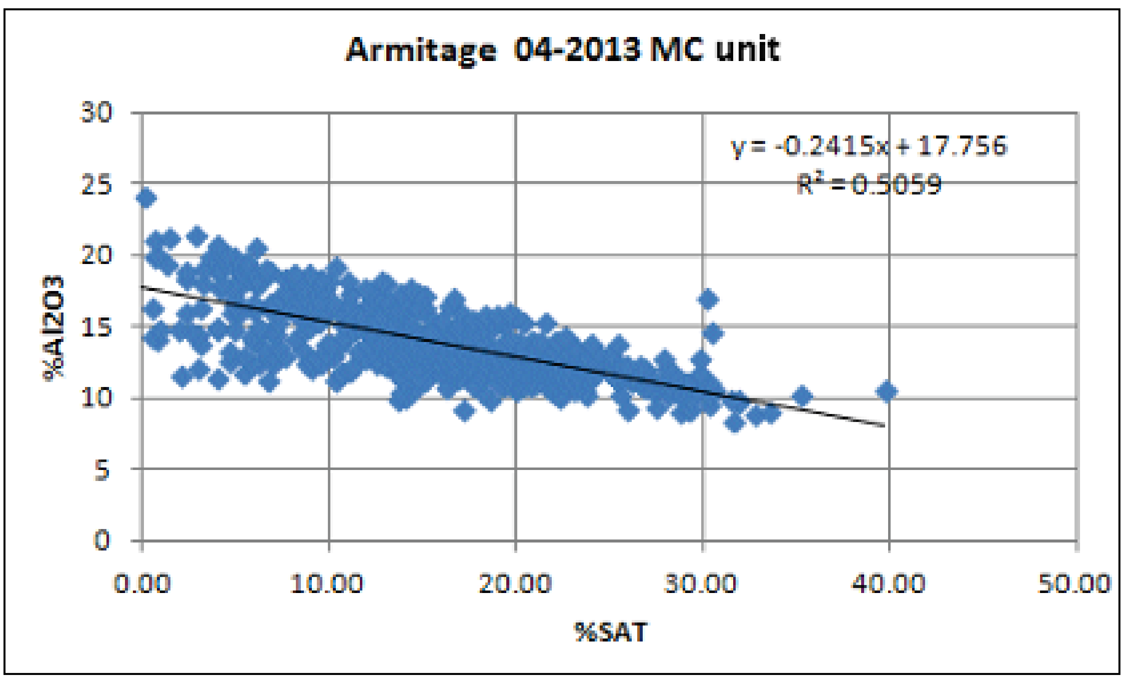
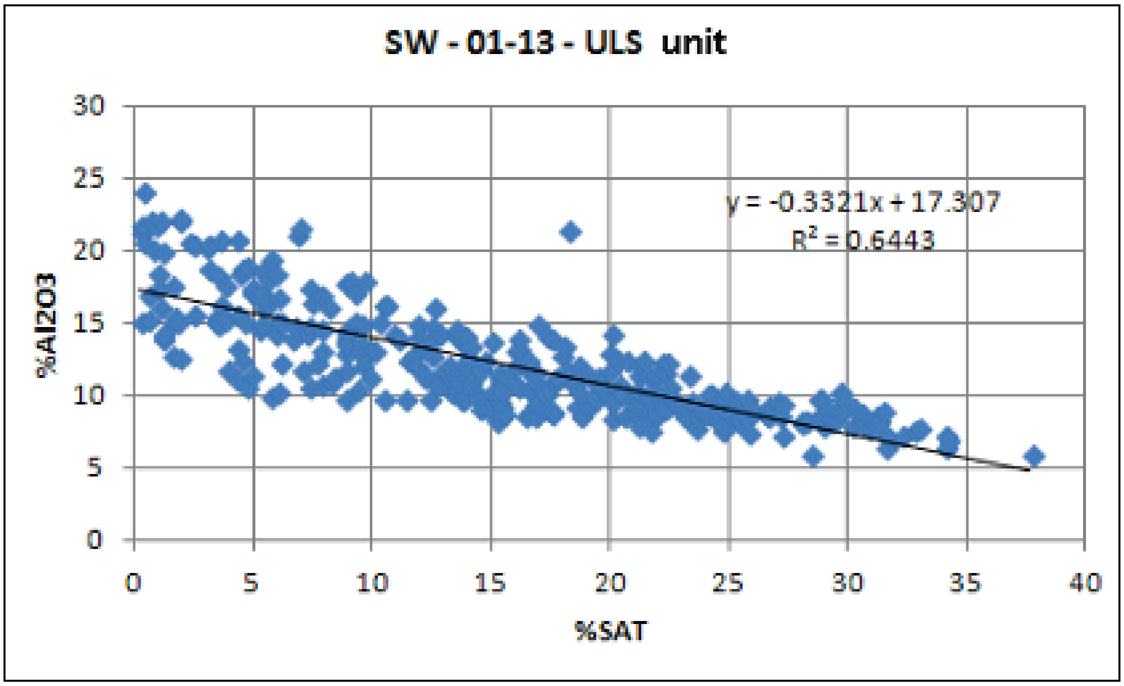


Figure 14-56: Correlation and regression of %Al2O3 over %Satmagan for MC



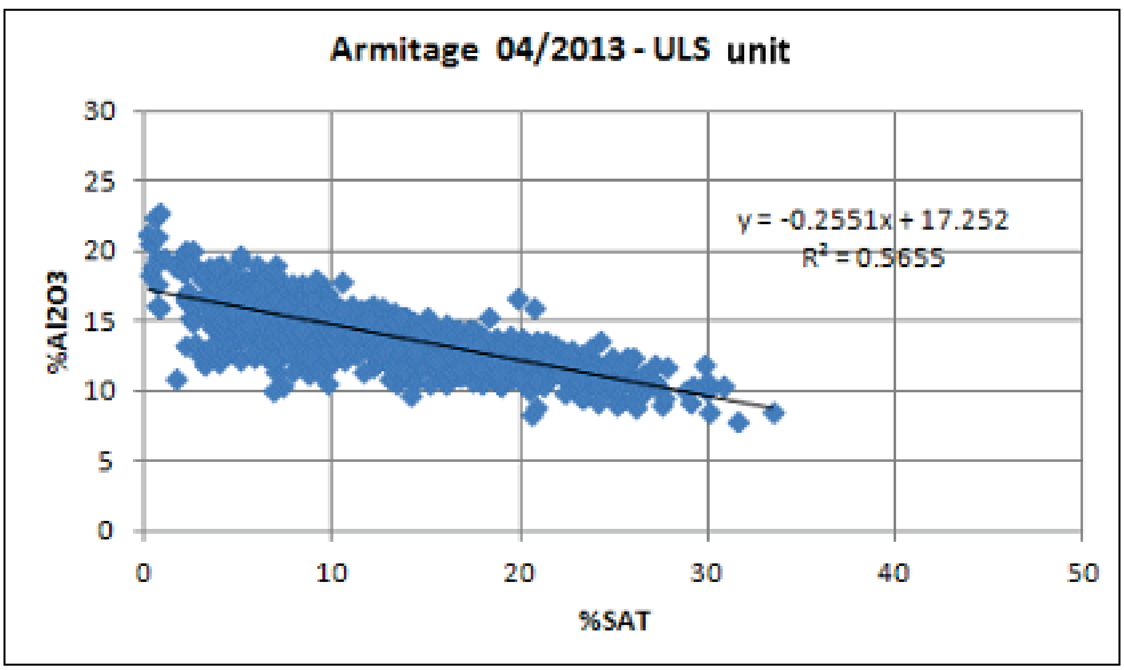
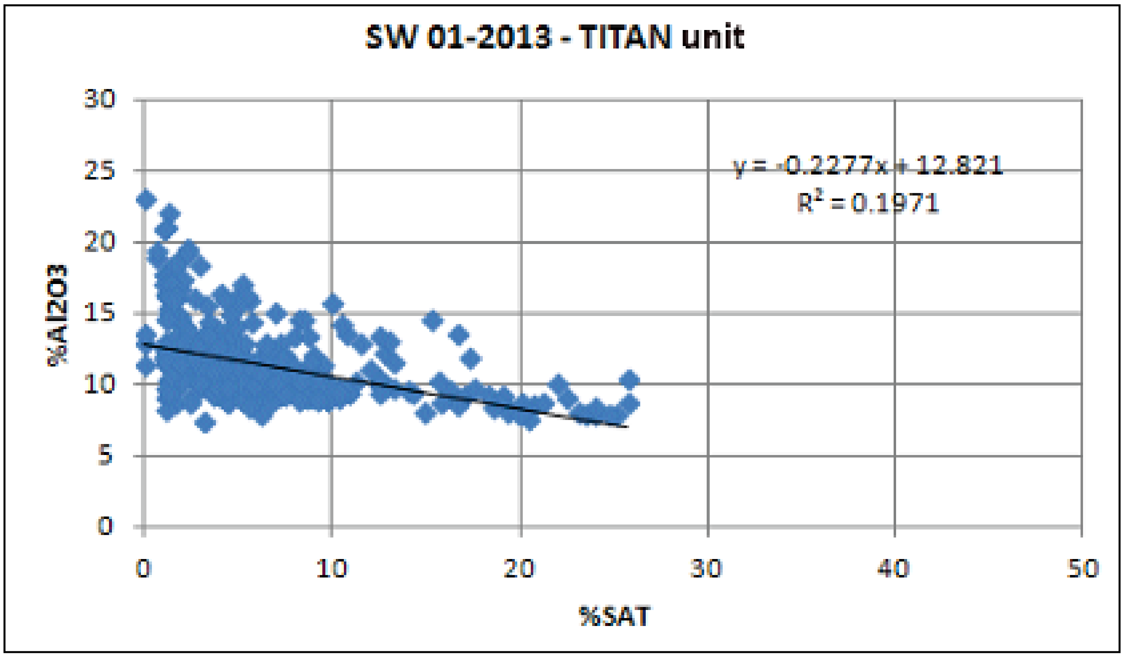


Figure 14-57: Correlation and regression of %Al2O3 over %Satmagan for ULS



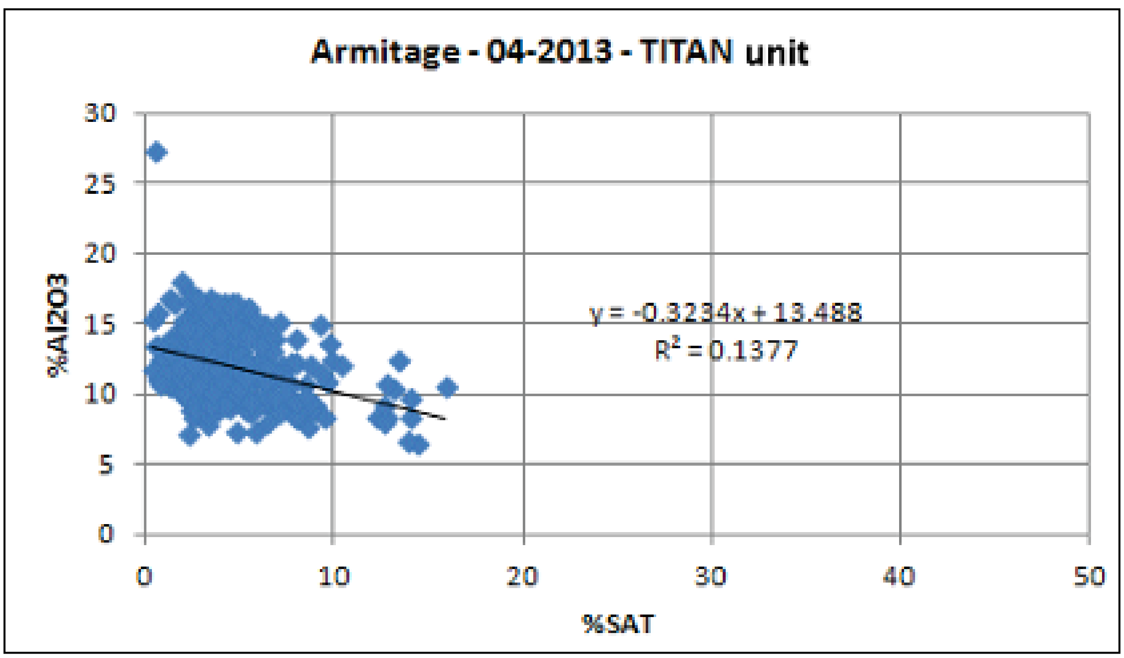


Figure 14-58: Correlation and regression of %Al2O3 over %Satmagan for Titan

Search conditions for the ID2 interpolation of additional variables are the same as those of the principal variables, with the exception being that an average orientation of search ellipsoids is used in all domains. The average values used for Southwest and Armitage are a dip of 70º to N130 and a dip of 70º to N160, respectively. This simplification does not alter the overall correlation relations between the principal and additional variables in the blocks, since there is no correlation between them in the first place.

Statistics for the block estimates of principal variables in each unit of the Southwest deposit are listed in Table 14-11 and Table 14-12 , and in Table 14-13 and Table 14-14 for the Armitage deposit. They are compared to the statistics of sample data in the same units in each deposit.

Table 14-11: Statistics of interpolated block principal values and samples in the Southwest Deposit

Deposit	Unit	Domain	Variable	Type	Number	Min.	Max.	Average	%CV
Southwest	BCS	All	%SAT	Block	141,493	0.1	35.8	11.9	39.6
				Sample	1,353	0.0	41.7	12.2	73.7
Southwest	BCS	All	%Fe_2O_3	Block	141,493	9.4	61.4	29.6	21.8
				Sample	1,353	4.3	64.5	30.0	40.0
Southwest	BCS	All	%TiO_2	Block	141,493	0.8	12.2	4.6	30.2
				Sample	1,353	0.3	12.6	4.7	53.4
Southwest	BCS	All	%V_2O_5	Block	141,493	0.03	0.85	0.36	28.9
				Sample	1,353	0.02	0.91	0.36	54.7
Southwest	MC	All	%SAT	Block	69,844	1.5	41.1	23.1	18.9
				Sample	655	0.0	42.2	23.4	40.4
Southwest	MC	All	%Fe_2O_3	Block	69,844	12.3	68.1	46.5	13.0
				Sample	655	9.5	68.9	46.7	26.2
Southwest	MC	All	%TiO_2	Block	69,844	1.4	15.8	9.1	19.4
				Sample	655	0.8	16.5	9.2	34.6
Southwest	MC	All	%V_2O_5	Block	69,844	0.08	0.91	0.57	15.8
				Sample	655	0.03	0.96	0.57	32.7
Southwest	ULS	All	%SAT	Block	45,432	1.0	36.0	16.1	33.9
				Sample	423	0.0	37.7	16.1	61.4
Southwest	ULS	All	%Fe_2O_3	Block	45,432	9.6	62.7	39.0	20.7
				Sample	423	8.7	64.7	38.8	35.3
Southwest	ULS	All	%TiO_2	Block	45,432	1.1	15.4	8.4	26.7
				Sample	423	0.8	15.8	8.3	43.9
Southwest	ULS	All	%V_2O_5	Block	45,432	0.04	0.76	0.36	29.4
				Sample	423	0.02	0.84	0.35	54.1

Deposit	Unit	Domain	Variable	Type	Number	Min.	Max.	Average	%CV
Southwest	Titan	All	%SAT	Block	69,036	0	27.0	5.8	66.8
				Sample	575	0.0	29.0	5.9	90.7
Southwest	Titan	All	$\%Fe_2O_3$	Block	69,036	9.6	57.1	29.1	19.5
				Sample	575	8.4	57.7	28.8	30.9
Southwest	Titan	All	$\%TiO_2$	Block	69,036	1.0	14.1	5.4	30.5
				Sample	575	0.8	14.7	5.3	46.4
Southwest	Titan	All	$\%V_2O_5$	Block	69,036	0.01	0.50	0.13	51.6
				Sample	575	0.01	0.58	0.13	72.2

Table 14-12: Statistics of interpolated block secondary values and samples in the Southwest Deposit

Deposit	Unit	Domain	Variable	Type	Number	Min.	Max.	Average	%CV
Southwest	BCS	All	$\%Al_2O_3$	Block	141,493	9.0	19.8	16.6	8.5
				Sample	1,067	7.1	26.8	16.4	20.8
Southwest	BCS	All	$\%P_2O_5$	Block	141,493	0.01	0.25	0.038	58.9
				Sample	1,067	0	0.36	0.039	113.1
Southwest	BCS	All	%S	Block	141,493	0.01	0.8	0.11	43.9
				Sample	955	0.01	1.43	0.11	85.6
Southwest	MC	All	$\%Al_2O_3$	Block	69,844	5.8	17.9	11.7	12.4
				Sample	556	5.6	23.6	11.6	27.0
Southwest	MC	All	$\%P_2O_5$	Block	69,844	0	0.22	0.030	50.8
				Sample	556	0	0.24	0.031	107.5
Southwest	MC	All	%S	Block	69,844	0.04	1.21	0.17	39.9
				Sample	522	0.02	1.87	0.16	80.6
Southwest	ULS	All	$\%Al_2O_3$	Block	45,432	5.5	17.0	12.3	14.9
				Sample	352	5.9	24.0	12.3	30.8
Southwest	ULS	All	$\%P_2O_5$	Block	45,432	0.01	0.22	0.036	48.2
				Sample	352	0	0.28	0.040	96.2
Southwest	ULS	All	%S	Block	45,432	0.04	0.75	0.25	33.6
				Sample	336	0.02	0.90	0.24	51.5
Southwest	Titan	All	$\%Al_2O_3$	Block	69,036	7.3	12.8	11.6	6.4
				Sample	461	7.5	23.0	11.5	21.4
Southwest	Titan	All	$\%P_2O_5$	Block	69,036	0.01	0.26	0.043	50.0
				Sample	461	0	0.30	0.043	100
Southwest	Titan	All	%S	Block	69,036	0	1.30	0.29	24.3
				Sample	455	0	1.48	0.29	51.7

Table 14-13: Statistics of interpolated block principal values and samples in the Armitage Deposit

Deposit	Unit	Domain	Variable	Type	Number	Min.	Max.	Average	%CV
Armitage	BCS	All	%SAT	Block	126,257	0.9	31.2	12.4	36.0
				Sample	995	0.1	39.8	12.7	63.2
Armitage	BCS	All	$\%Fe_2O_3$	Block	126,257	9.9	51.5	30.3	19.1
				Sample	995	7.4	59.5	30.7	33.7
Armitage	BCS	All	$\%TiO_2$	Block	126,257	0.5	10.3	4.9	27.4
				Sample	995	0.3	10.9	5	46.5
Armitage	BCS	All	$\%V_2O_5$	Block	126,257	0.03	0.77	0.37	23.0
				Sample	995	0.01	0.94	0.37	44.7
Armitage	MC	All	%SAT	Block	72,834	2.6	33.9	16.6	21.1
				Sample	593	0	39.7	17.1	44.0
Armitage	MC	All	$\%Fe_2O_3$	Block	72,834	14.3	55.9	37.1	13.1
				Sample	593	7.2	57.9	37.5	26.8
Armitage	MC	All	$\%TiO_2$	Block	72,834	2.1	11.9	7.2	17.3
				Sample	593	0.7	13.2	7.3	33.5
Armitage	MC	All	$\%V_2O_5$	Block	72,834	0.10	0.79	0.43	15.2
				Sample	593	0.01	0.82	0.43	34.6
Armitage	ULS	All	%SAT	Block	111,763	0.5	30.8	12.5	32.6
				Sample	831	0.2	35.1	13.1	53.8
Armitage	ULS	All	$\%Fe_2O_3$	Block	111,763	12.5	55.4	34.6	14.7
				Sample	831	2.3	58.7	35.3	26.0
Armitage	ULS	All	$\%TiO_2$	Block	111,763	1.5	12.3	7.3	18.6
				Sample	831	0.2	13	7.5	32
Armitage	ULS	All	$\%V_2O_5$	Block	111,763	0.06	0.71	0.32	27.7
				Sample	831	0.01	0.75	0.33	44.1
Armitage	Titan	All	%SAT	Block	65,315	0.7	15.1	4.4	34.8
				Sample	392	0.3	15.9	4.4	62.9
Armitage	Titan	All	$\%Fe_2O_3$	Block	65,315	12.3	38.6	26.4	10.9
				Sample	392	5.4	39.3	26.4	18.5
Armitage	Titan	All	$\%TiO_2$	Block	65,315	1.7	8.2	4.4	19.8
				Sample	392	0.2	8.3	4.4	32
Armitage	Titan	All	$\%V_2O_5$	Block	65,315	0.01	0.28	0.10	24.8
				Sample	392	0.01	0.33	0.10	39.3

Table 14-14: Statistics of interpolated block secondary values and samples in the Armitage Deposit

Deposit	Unit	Domain	Variable	Type	Number	Min.	Max.	Average	%CV
Armitage	BCS	All	$\%Al_2O_3$	Block	126,257	9.1	23.1	16.8	11.5
				Sample	979	9.3	26.9	16.5	18.5
Armitage	BCS	All	%P2O5	Block	126,257	0.01	0.23	0.029	49.9
				Sample	979	0.01	0.26	0.031	98.3
Armitage	BCS	All	%S	Block	126,257	0.02	1.2	0.13	48.4
				Sample	980	0.01	1.80	0.14	78.3
Armitage	MC	All	$\%Al_2O_3$	Block	72,834	9.0	20.0	13.9	9.1
				Sample	585	8.4	24.1	13.8	18.2
Armitage	MC	All	$\%P_2O_5$	Block	72,834	0.01	0.15	0.032	47.5
				Sample	585	0.01	0.19	0.030	95.4
Armitage	MC	All	%S	Block	72,834	0.06	1.43	0.27	43.5
				Sample	585	0.04	1.57	0.26	63.4
Armitage	ULS	All	$\%Al_2O_3$	Block	111,763	8.5	21.1	14.2	9.6
				Sample	824	8.0	22.8	14.1	16.2
Armitage	ULS	All	$\%P_2O_5$	Block	111,763	0.01	0.15	0.023	46.4
				Sample	824	0.01	0.15	0.020	84.8
Armitage	ULS	All	%S	Block	111,763	0.05	1.10	0.41	30.6
				Sample	824	0.02	1.42	0.43	46.8
Armitage	Titan	All	$\%Al_2O_3$	Block	65,315	7.7	23.7	12.2	13.0
				Sample	390	6.6	27.4	12.1	19.1
Armitage	Titan	All	$\%P_2O_5$	Block	65,315	0.01	0.19	0.033	49.5
				Sample	390	0.01	0.33	0.031	78.8
Armitage	Titan	All	%S	Block	65,315	0.06	1.10	0.37	23.1
				Sample	390	0.01	1.29	0.37	42.0

As a general rule, most of the blocks are interpolated in the first run, with none or only few blocks in the third run, with the largest search ellipsoid. The average interpolated block grades tend to keep close to the average sample grade in the same unit. As expected, there is much less variability for block estimates (range between maximum and minimum or coefficient of variation, i.e., %CV in tables), compared to sample data variability, thus reflecting the difference in support size (on an order of magnitude basis ~12.5 kg for samples vs. ~1,250,000 kg for blocks).

Typical sections and bench plans with interpolated %Satmagan of mineralized blocks are shown in Figure 14-59 to Figure 14-63 for the Southwest deposit. Similar examples of mineralized blocks in the Armitage are shown in Figure 14-64 to Figure 14-68.

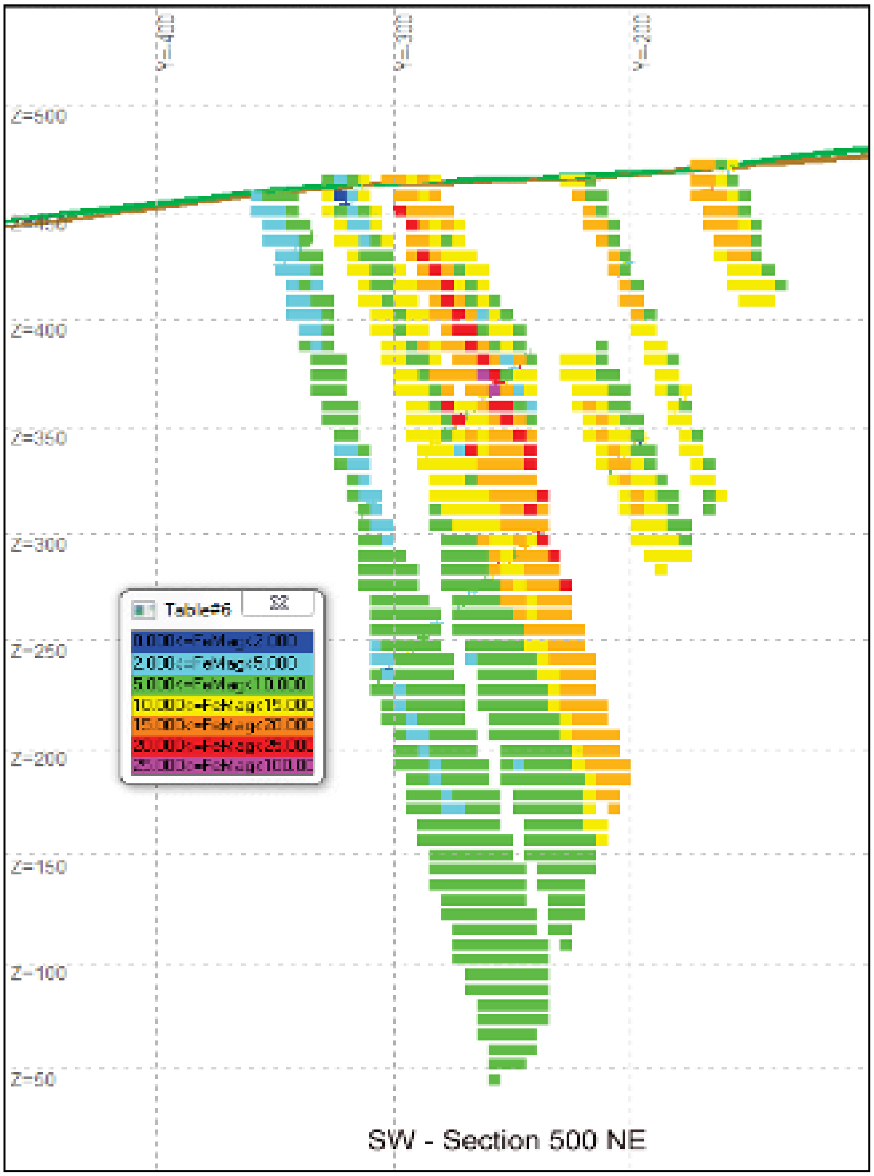


Figure 14-59: Southwest drill panel 500 cut through interpolated %Satmagan mineralized blocks. Section is viewed looking northeast

Mineralized drill samples in the same section corridors are shown with a color coded "+" for comparison.

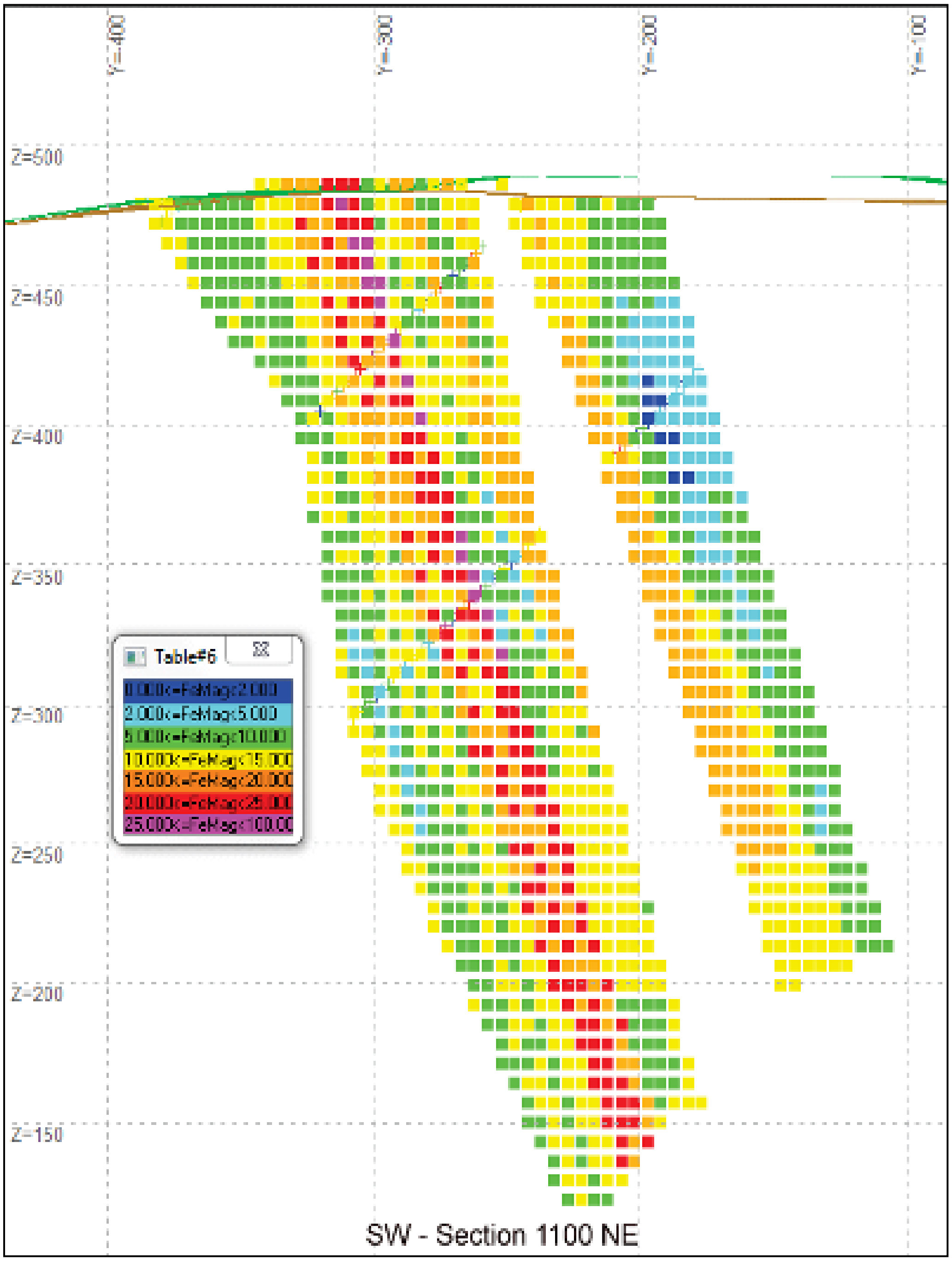


Figure 14-60: Southwest drill panel 1100 cut through interpolated %Satmagan mineralized blocks. Section is viewed looking northeast

Mineralized drill samples in the same section corridors are shown with a color coded "+" for comparison.

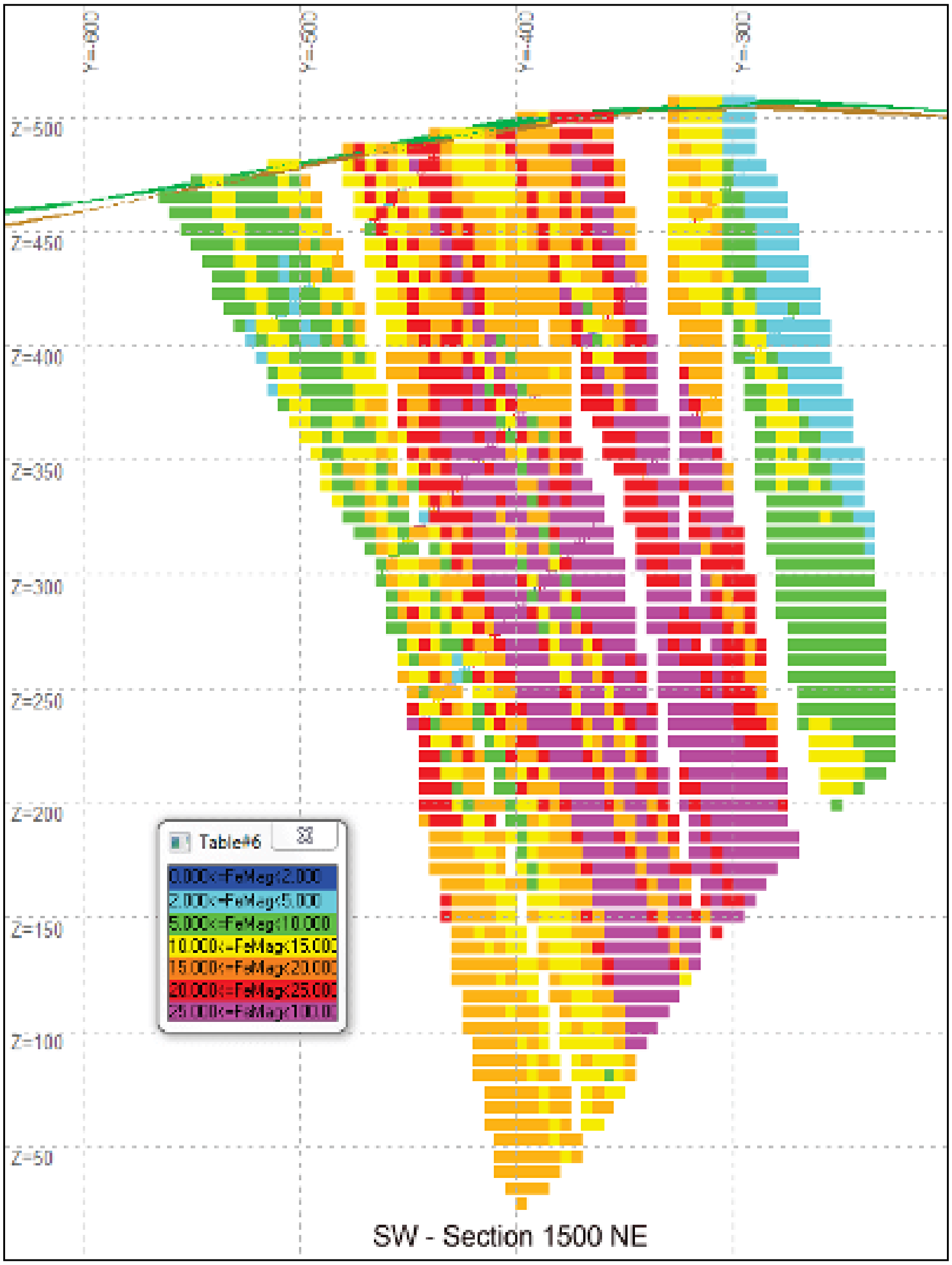


Figure 14-61: Southwest drill panel 1500 cut through interpolated %Satmagan mineralized blocks. Section is viewed looking northeast

Mineralized drill samples in the same section corridors are shown with a color coded "+" for comparison.

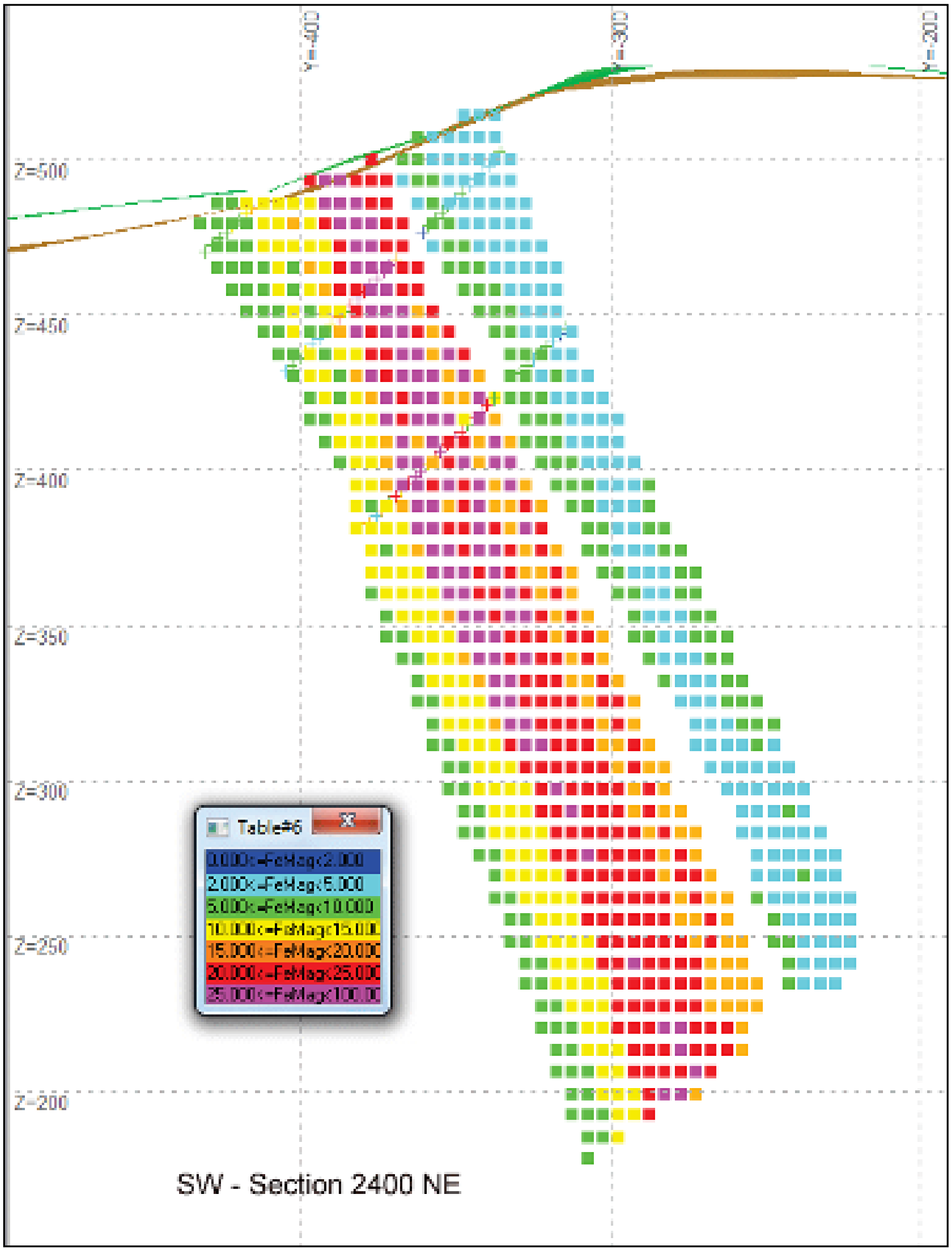


Figure 14-62: Southwest drill panel 2400 cut through interpolated %Satmagan mineralized blocks. Section is viewed looking northeast

Mineralized drill samples in the same section corridors are shown with a color coded "+" for comparison.

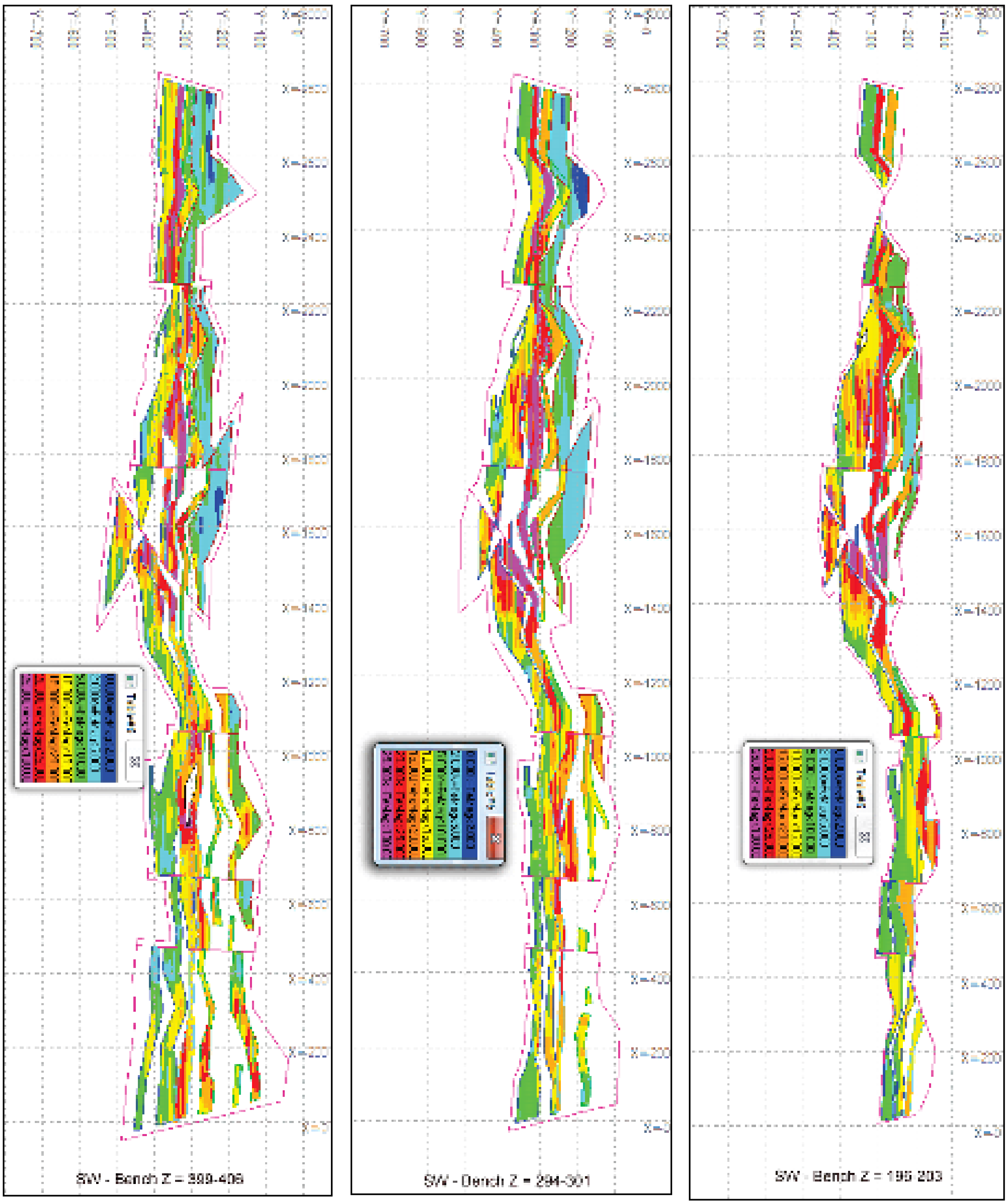


Figure 14-63: Southwest bench plans with interpolated %Satmagan in mineralized blocks

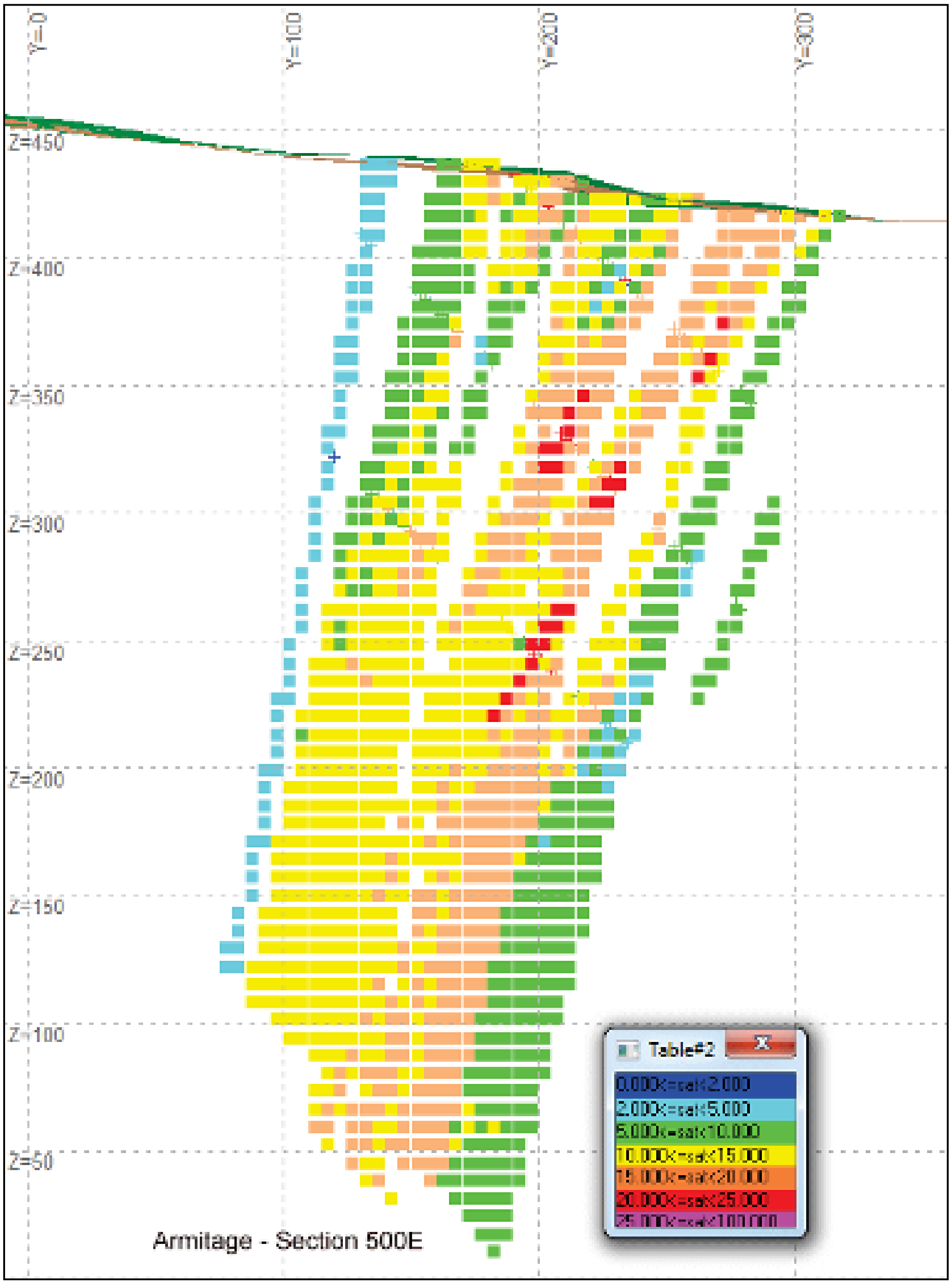


Figure 14-64: Armitage drill panel 500 through interpolated %Satmagan in mineralized blocks. Section is viewed looking WSW

Mineralized samples in the same section corridors are shown with a “+”.

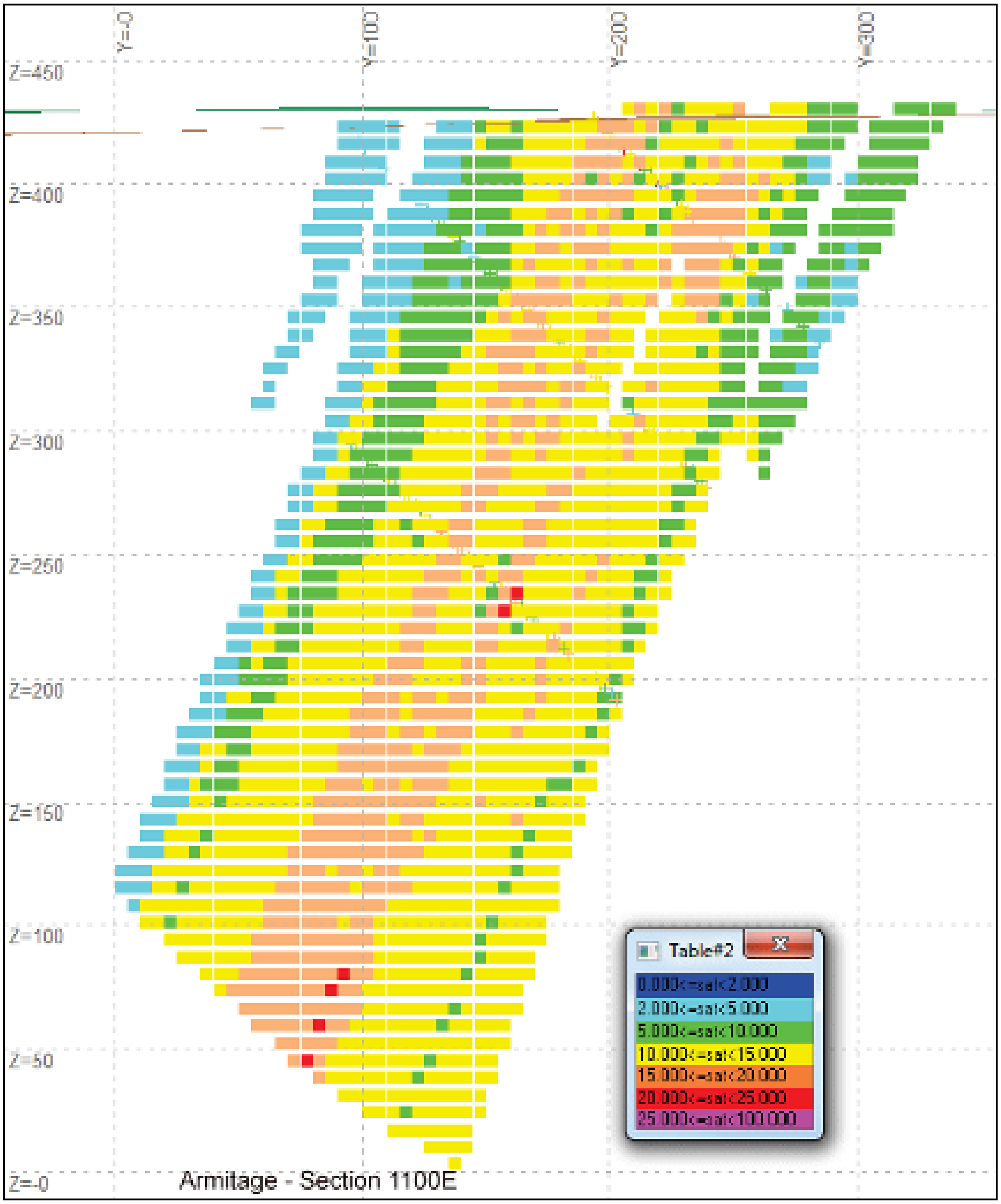


Figure 14-65: Armitage drill panel 1100 through interpolated %Satmagan in mineralized blocks. Section is viewed looking WSW

Mineralized samples in the same section corridors are shown with a "+".

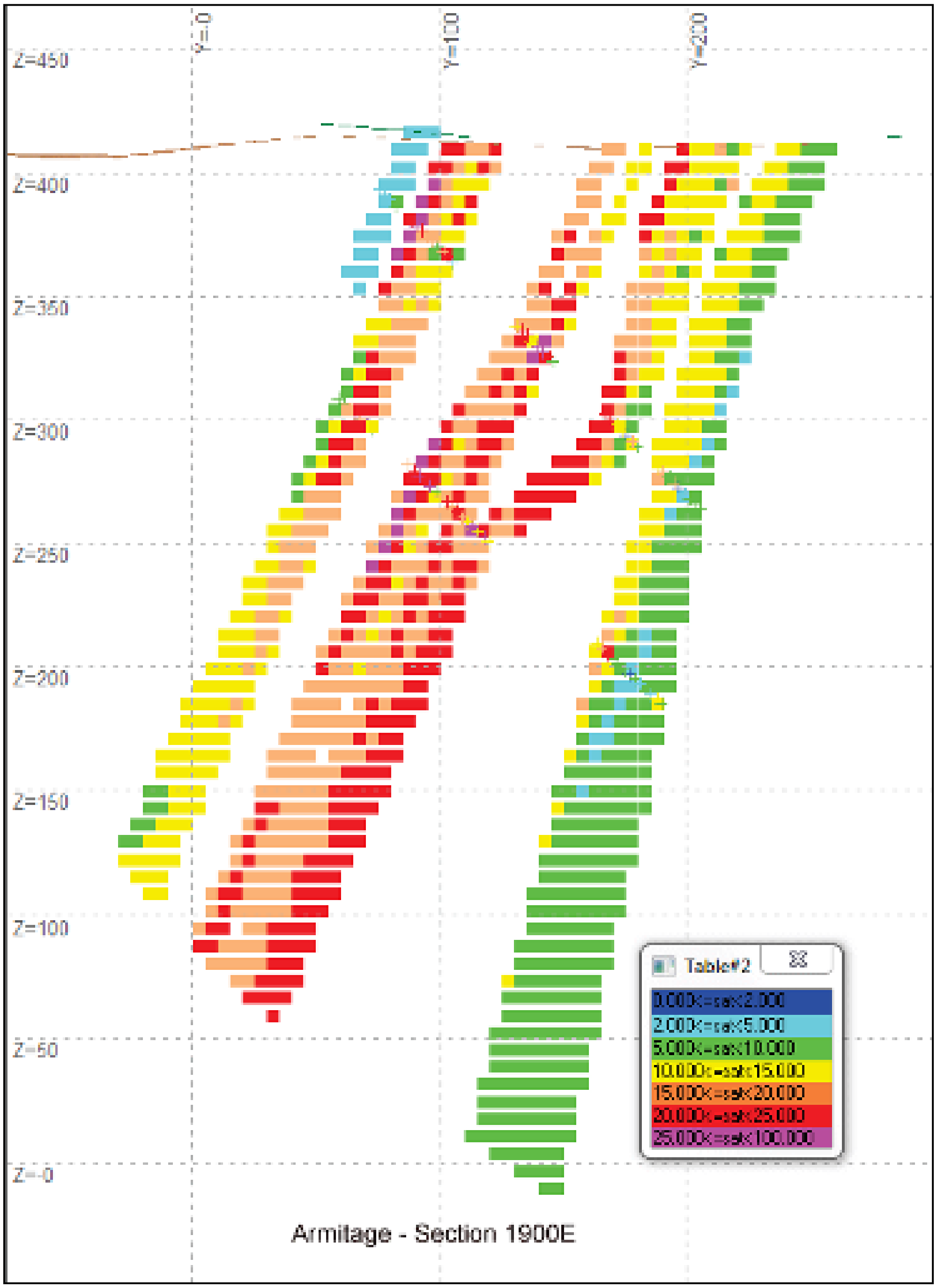


Figure 14-66: Armitage drill panel 500 through interpolated %Satmagan in mineralized blocks. Section is viewed looking WSW

Mineralized samples in the same section corridors are shown with a "+".

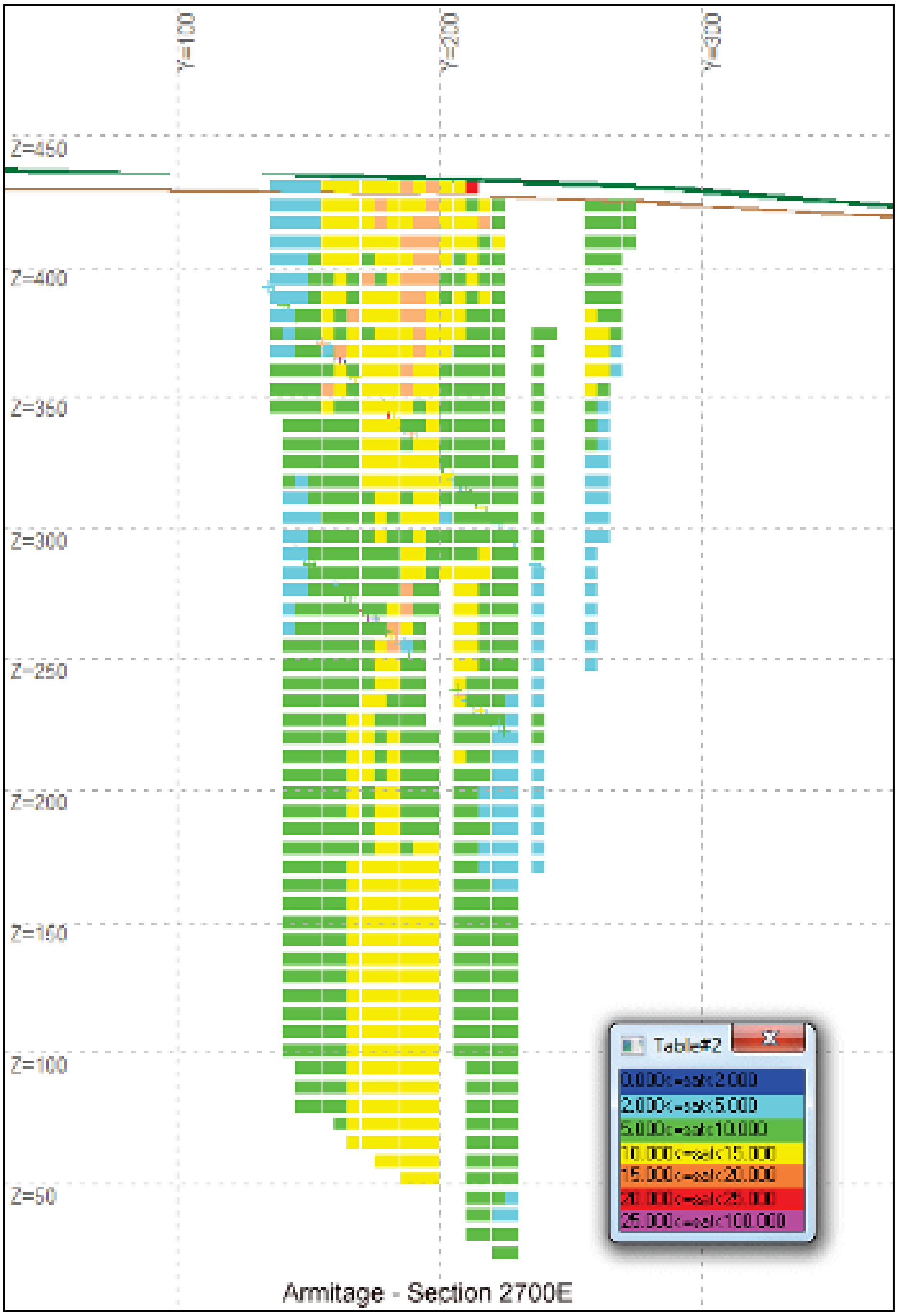


Figure 14-67: Armitage drill panel 2700 through interpolated %Satmagan in mineralized blocks. Section is viewed looking WSW

Mineralized samples in the same section corridors are shown with a “+”.

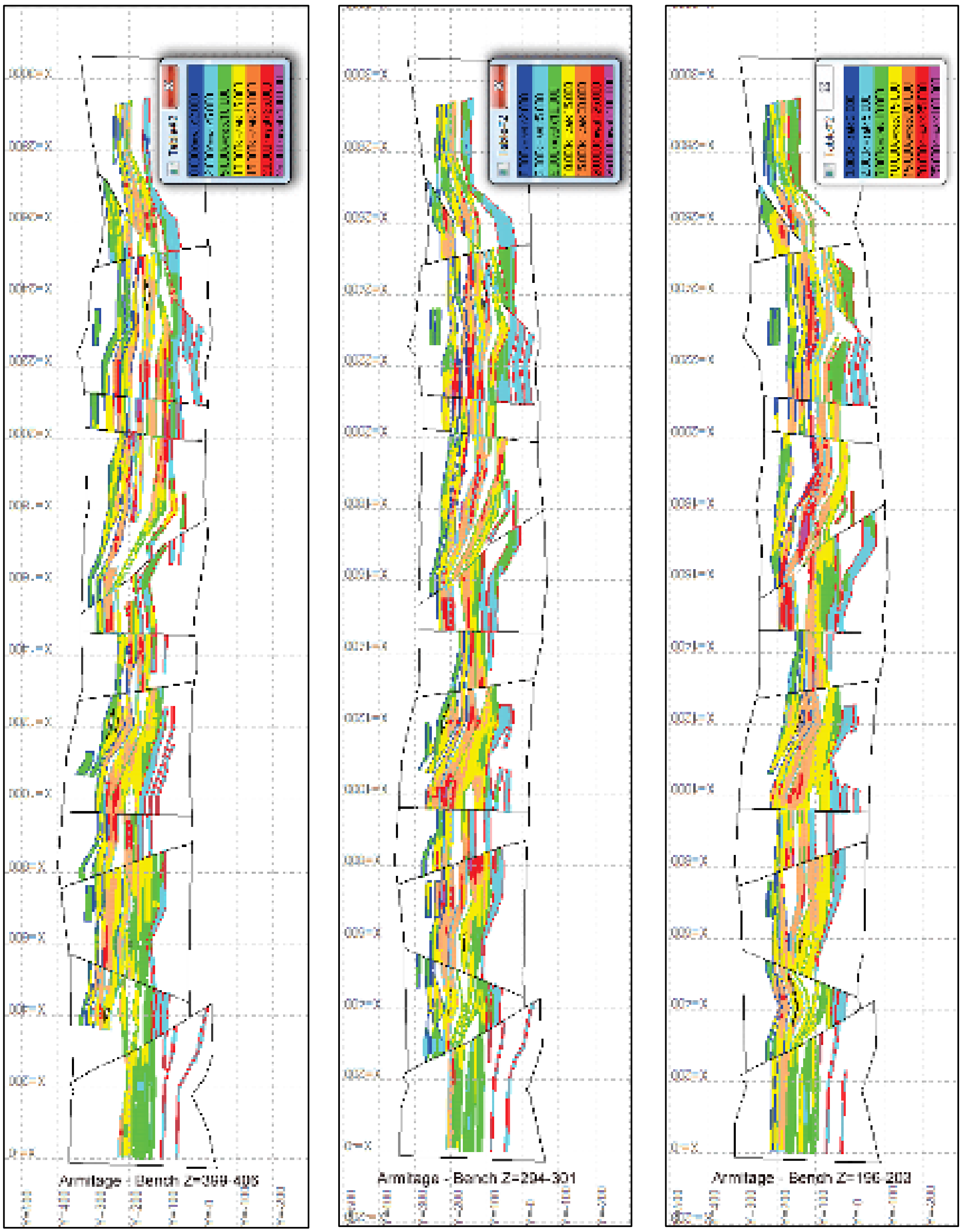


Figure 14-68: Armitage bench plans with interpolated %Satmagan in mineralized blocks

14.6. Validation of Estimated Block Grades

Most resource model validation techniques are based on the idea that block grade estimates should match available sample grades in the same sector. A first approach is simply to examine sections or benches with both block estimates and samples (with the same colour code, according to grade) and make sure that they visually match.

Another approach is to slice the deposits along strike and compare the average block estimate and the average sample grade in the same slice. By plotting those averages with the slice number, we produce a so-called "swath plot", which shows how well the block estimates follow the trends of sample data along strike.

Swath plots have been calculated with 100 m wide slices centered on drill sections. We have 26 slices in Southwest and 27 slices in Armitage. In each slice, we keep track of:

- The number of samples and blocks in any given unit, as well as total;
- The average %Satmagan and %TiO_2 of those samples and blocks.

Swath plots for specific units, as well as the combination of all four, are found in Figure 14-69 to Figure 14-72. As a general rule, the average block estimate closely follows the average sample value in the same unit and slice. As expected, there is reasonable smoothing with block averages, as explained in the next section.

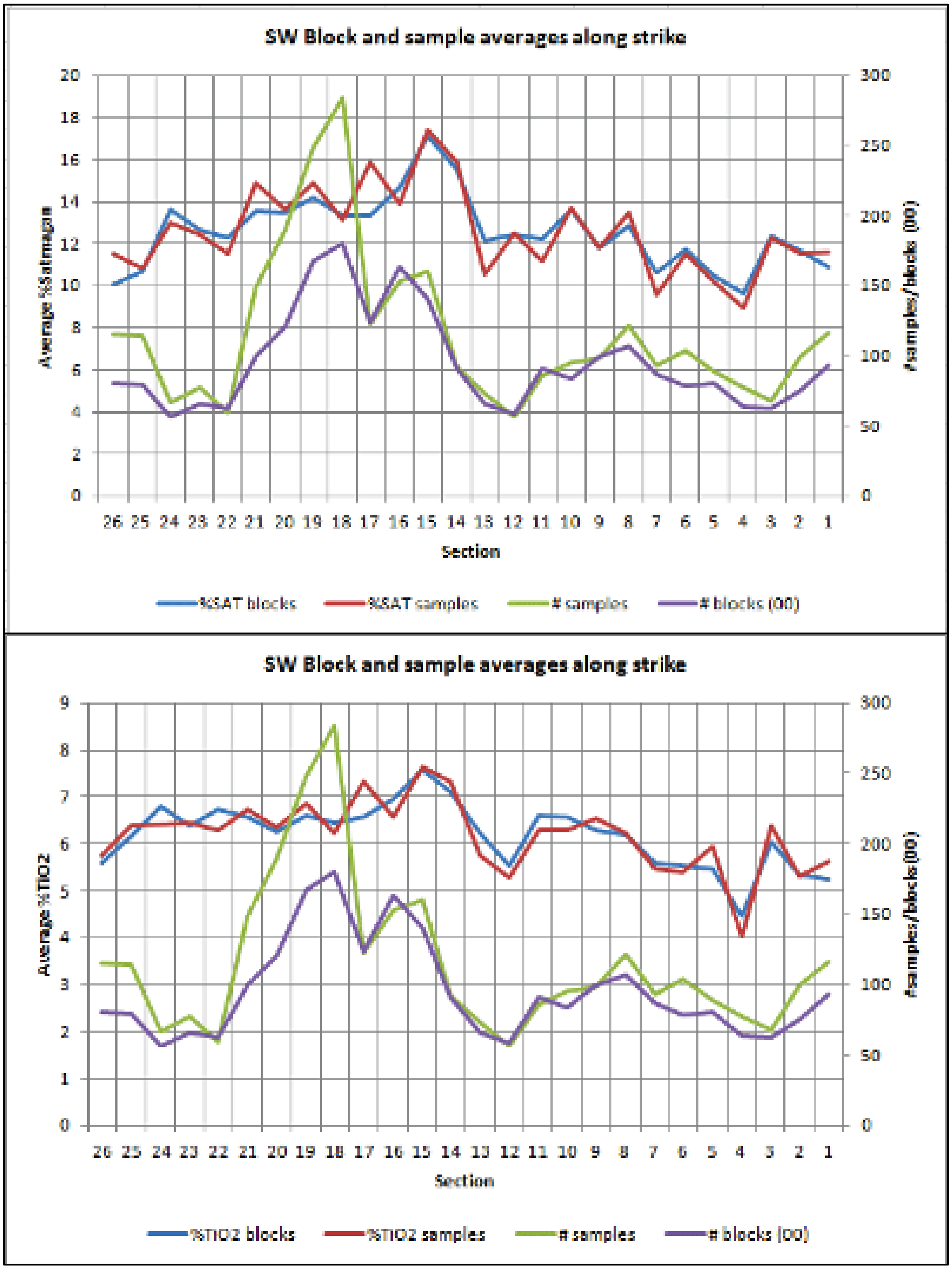


Figure 14-69: Swath plots in Southwest Deposit with all units together

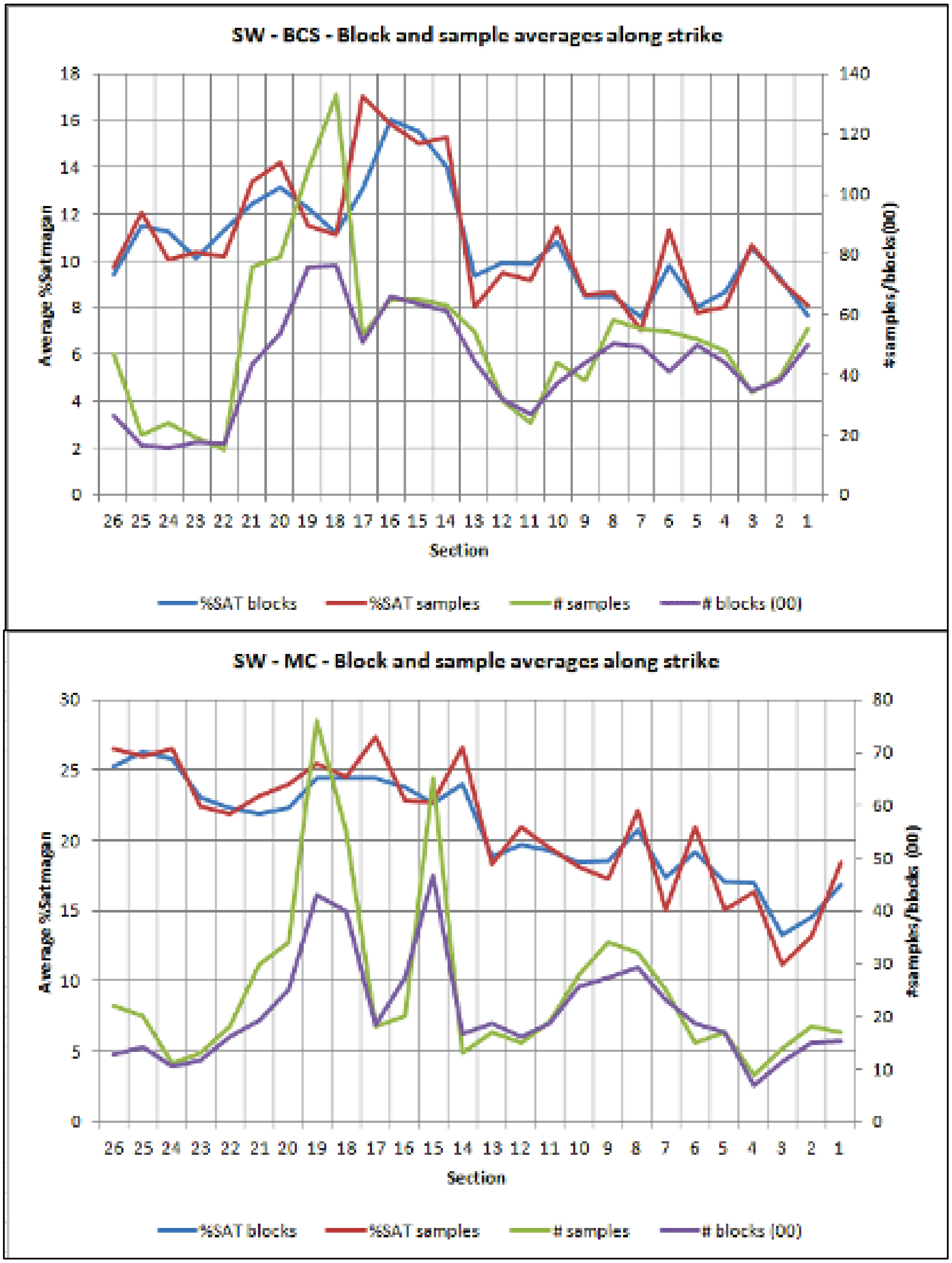


Figure 14-70: Swath plots of %Satmagan in BCS and MC units of the SW Deposit

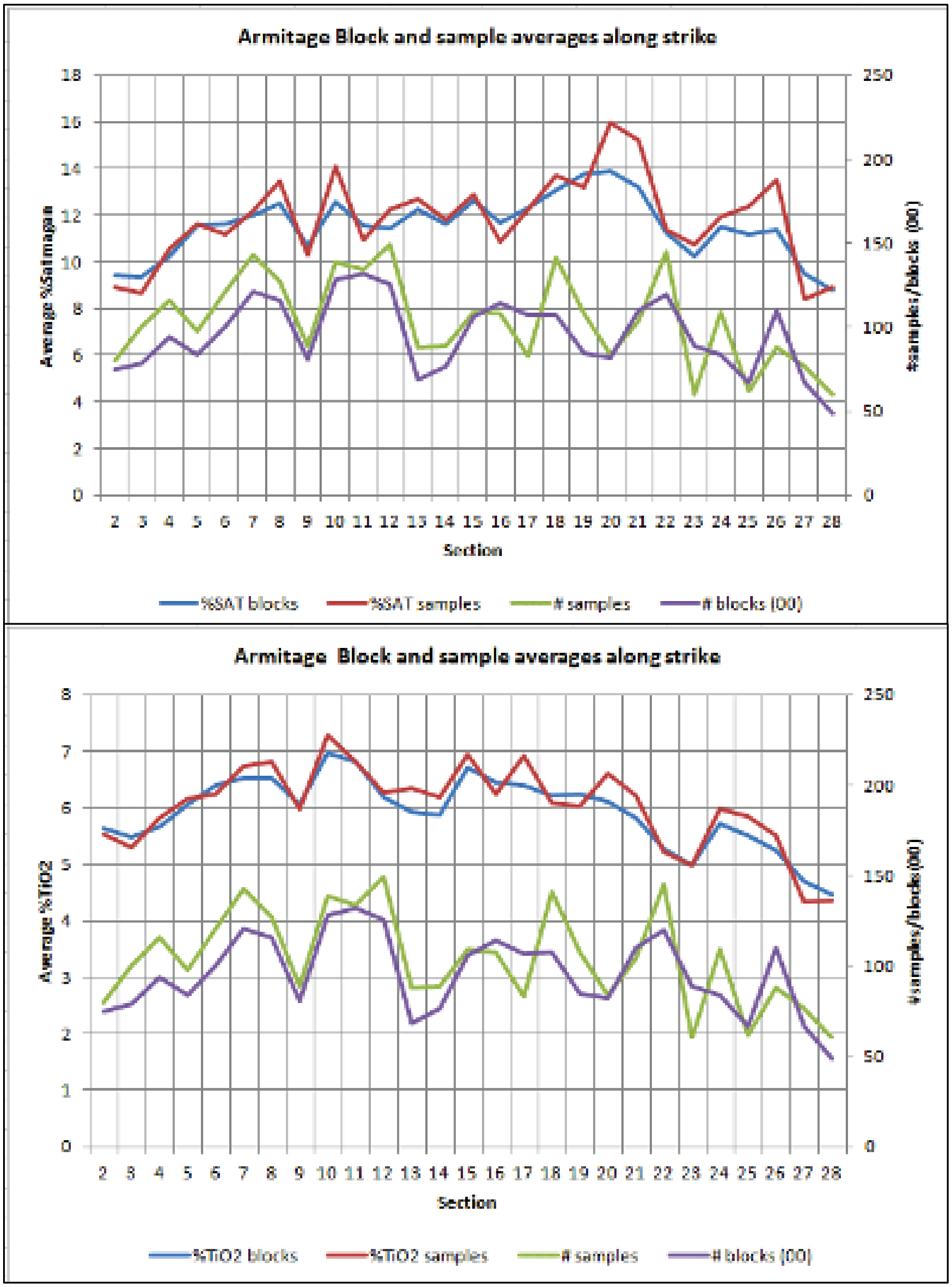


Figure 14-71: Swath plots in Armitage Deposit with all units together

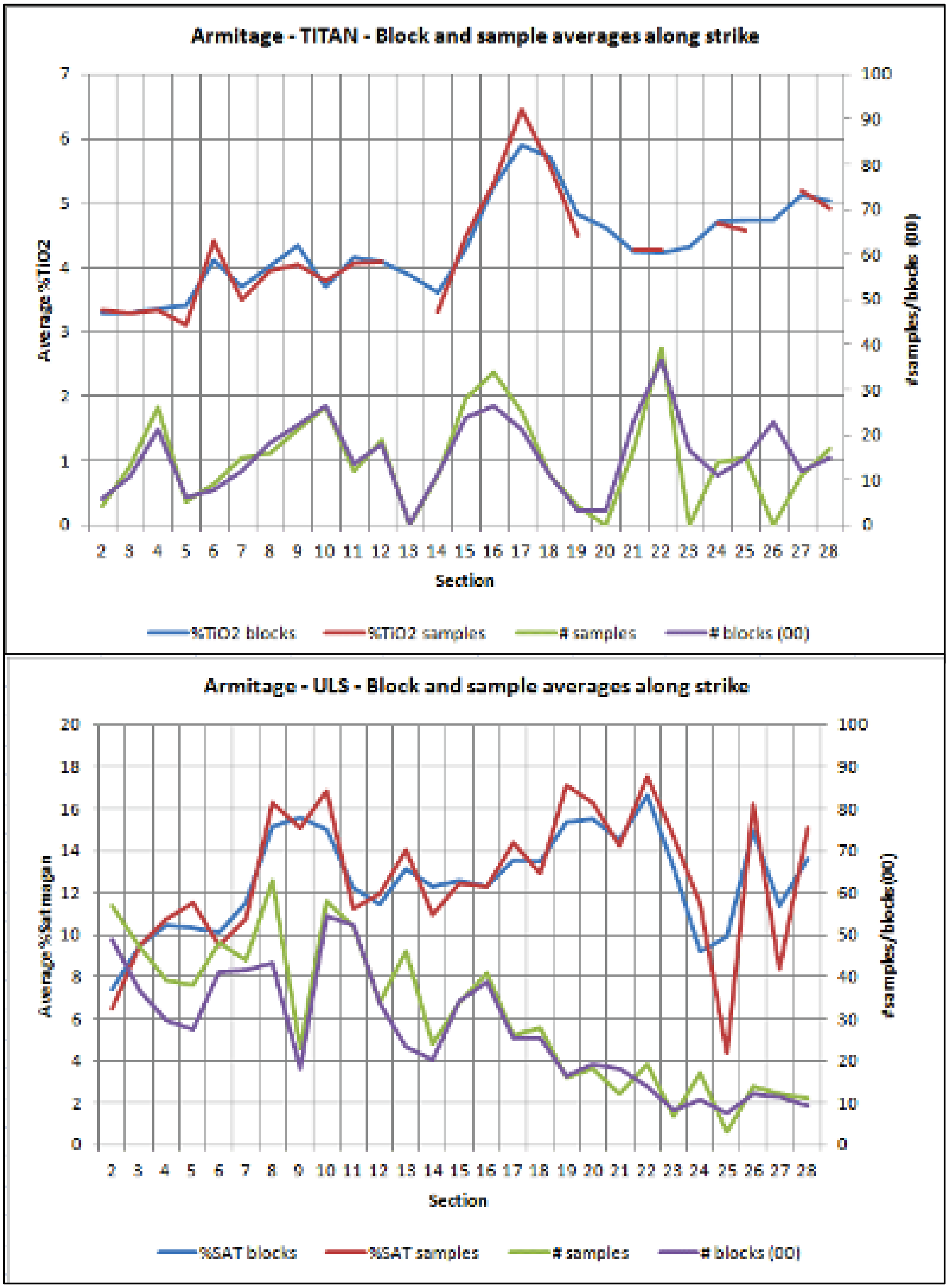


Figure 14-72: Swath plots in the ULS and TITAN units of the Armitage Deposit

14.7. Sensitivity of Block Grade Estimates and Dilution

After calculating and modelling the variograms of the principal variables in each unit of both the Southwest and Armitage deposits, a complete kriged block model was produced. Upon validating this model against the assay data, it was decided that the OK model is not representative of the grade distribution in the deposits, as it overly smoothes and dilutes the average block grades relative to the average assay grades. A series of specific interpolations were then made on a limited dataset (BCS in the Southwest) to test the sensitivity of the assay grade distribution to the estimator used.

Figure 14-73 illustrates the influence of the variability of block grade estimates with the estimation method, on predicted tonnage and grades above cut-off. This exercise is limited to the M+I blocks of the BCS zone in the SW deposit to assess the preferred methodology. With their low variability, kriged block grades give high tonnage + low-grade estimates above cut-off. Inverse distance (ID1) estimates are very similar to the OK. The data indicate that the Inverse distance to the cube (ID3) and nearest-neighbor (NN) or polygonal block estimates predict an unrealistically high selectivity, i.e., material above cut-off concentrated in a few very high-grade blocks.

Geostatistics were then employed to determine which set of estimates has the most realistic variability. Variability of block grades of any given size can be determined with a variogram model of the sample grades. More precisely, the variance of the blocks is equal to the variance of samples, less the average value of the variogram model function inside the block.

The average values (variance) of blocks displayed in column V(B) of Table 14-15 are derived by filling a 10 x 5 x 7 m block with discretization points, then calculating the variogram function value for the distance between any two discretization points and finally, averaging all those variogram values. The predicted variability of blocks under the form of a coefficient of variation (%CV(B)) is compared to the variability of inverse squared distance estimates (%CV(ID2)) and the variability of kriged grade estimates (%CV(OK)). With the exception of the BCS unit of the Southwest deposit, where %CV(ID2)>%CV(B), the variability of ID2 estimates, compared to the variability of kriged estimates, is closer to the predicted variability of blocks, hence the final selection of ID2 block estimates.

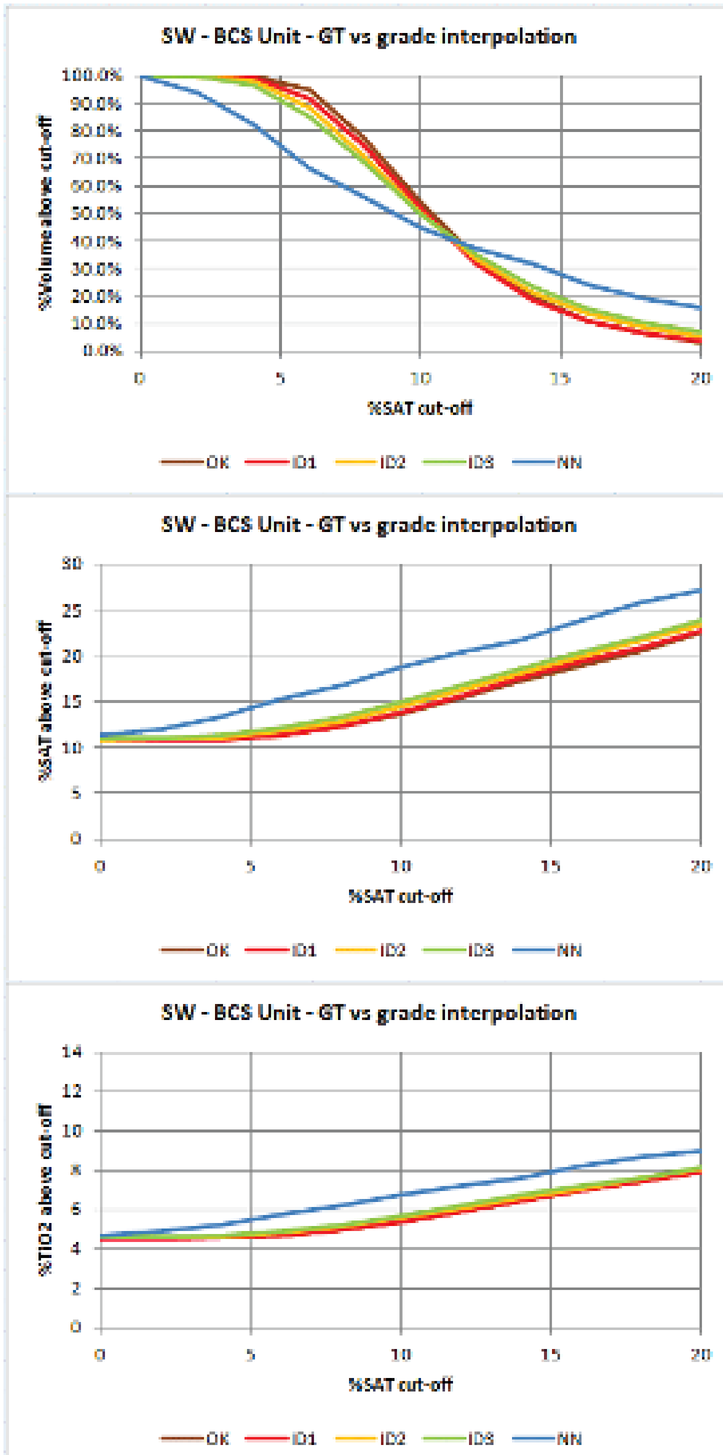


Figure 14-73: Grade-tonnage relationships with block grade estimation method

OK = kriging. IDn = inverse distance to the power n. NN= nearest-neighbor.

Table 14-15: Variability of samples, blocks and block estimates

Deposit	Unit	Variable	V(B)	%CV(S)	%CV(B)	%CV(ID2)	%CV(OK)
Southwest	BCS	%SAT	0.747	76.4	38.4	42.2	34.6
Southwest	BCS	$\%TiO_2$	0.709	53.4	28.8	30.2	24.9
Southwest	MC	%SAT	0.662	42.5	24.7	22.2	17.8
Southwest	MC	$\%TiO_2$	0.575	34.6	22.6	19.4	16.1
Southwest	ULS	%SAT	0.532	61.5	42.1	36.2	33.0
Southwest	ULS	$\%TiO_2$	0.539	43.9	29.8	26.7	23.8
Southwest	Titan	%SAT	0.449	85.3	63.3	58.5	52.6
Southwest	Titan	$\%TiO_2$	0.445	46.4	34.6	30.5	27.5
Armitage	BCS	%SAT	0.673	64.4	36.8	36.3	30.1
Armitage	BCS	$\%TiO_2$	0.580	46.5	30.1	27.4	23.3
Armitage	MC	%SAT	0.687	45.8	25.6	21.8	17.1
Armitage	MC	$\%TiO_2$	0.668	33.5	19.3	17.3	14.3
Armitage	ULS	%SAT	0.520	53.5	37.1	31.9	27.7
Armitage	ULS	$\%TiO_2$	0.625	32.0	19.6	18.6	15.9
Armitage	Titan	%SAT	0.556	63.5	42.3	35.0	31.9
Armitage	Titan	$\%TiO_2$	0.427	32.0	24.2	19.8	18.5

V(B) = average value of variogram model function in 10 x 5x 7 m blocks. %CV(S) = coefficient of variation of sample grades. %CV(B) = predicted coefficient of variation of blocks from V(B). %CV(ID2) = coefficient of variation of inverse squared distance block grade estimates. CV(OK) = coefficient of variation of ordinary kriged block grade estimates.

According to this analysis, the ID2 model appears to be the most representative of geology and grade distribution. It produces block estimates with variability closer to the theoretical values based on the results from the variogram models. There is a potential risk of data biasing in ID2 models that can stem from sample clustering around high grade intercepts. The systematic and uniform sampling protocols applied by BlackRock in both deposits has resulted in ID2 models that are unbiased in this respect.

Table 14-16 compares the statistics of block grade estimates by ID2 and OK for the two principal variables (%Satmagan and %TiO_2) in each geologic unit of the two deposits. These block estimates were derived from an older sample dataset. As a general rule, the mean block estimates are identical using both methods and are always similar to the mean sample value; hence, ID2 block estimates are not biased with respect to OK estimates. The difference between the two types of estimates is seen in their variability around the same mean.

The variability can also be assessed by examining the range of the estimates, i.e., the spread of values between minimum and maximum, or directly from the coefficient of variation in each case. Based on the significant nugget effect of variograms, the coefficient of variation of kriged estimates is, as expected, always less than that of inverse squared distance estimates, and both represent much less than the coefficient of variation of sample data.

Table 14-16: Statistics of samples, inverse squared distance and kriged block estimates

Deposit	Unit	Variable	Type	Min.	Max.	Average	%CV
Southwest	BCS	%SAT	ID2	0.1	35.8	11.0	42.2
			OK	1.7	32.8	11.0	34.6
			Sample	0.0	41.7	11.3	76.4
Southwest	BCS	%TiO_2	ID2	0.8	12.2	4.6	30.2
			OK	1.7	11.3	4.6	24.9
			Sample	0.3	12.6	4.7	53.4
Southwest	MC	%SAT	ID2	1.1	41.1	21.5	22.2
			OK	7.5	37.7	21.4	17.8
			Sample	0.0	42.2	21.8	42.5
Southwest	MC	%TiO_2	ID2	1.4	15.8	9.1	19.4
			OK	3.3	14.5	9.1	16.1
			Sample	0.8	16.5	9.2	34.6
Southwest	ULS	%SAT	ID2	0.9	35.4	15.2	36.2
			OK	2.8	32.8	14.8	33.0
			Sample	0.0	37.7	15.1	61.5
Southwest	ULS	%TiO_2	ID2	1.1	15.4	8.4	26.7
			OK	2.1	14.3	8.3	23.8
			Sample	0.8	15.8	8.3	43.9
Southwest	Titan	%SAT	ID2	0	24.4	5.6	58.5
			OK	1.0	21.3	5.7	52.6
			Sample	0.0	28.1	5.8	85.3

Deposit	Unit	Variable	Type	Min.	Max.	Average	%CV
Southwest	Titan	$\%TiO_2$	ID2	1.0	14.1	5.4	30.5
			OK	2.2	12.5	5.5	27.5
			Sample	0.8	14.7	5.3	46.4
Armitage	BCS	%SAT	ID2	0.8	31.2	11.9	36.3
			OK	2.7	25.9	12.0	30.1
			Sample	0.1	39.8	12.1	64.4
Armitage	BCS	$\%TiO_2$	ID2	0.5	10.3	4.9	27.4
			OK	2	9.4	4.9	23.3
			Sample	0.3	10.9	5	46.5
Armitage	MC	%SAT	ID2	2.6	33.9	16.1	21.8
			OK	4.8	27.3	15.9	17.1
			Sample	0	39.7	16.3	45.8
Armitage	MC	$\%TiO_2$	ID2	2.1	11.9	7.2	17.3
			OK	3.3	10.7	7.1	14.3
			Sample	0.7	13.2	7.3	33.5
Armitage	ULS	%SAT	ID2	0.5	30.8	12.0	31.9
			OK	2.9	25.8	12.0	27.7
			Sample	0.2	33.5	12.5	53.5
Armitage	ULS	$\%TiO_2$	ID2	1.5	12.3	7.3	18.6
			OK	2.9	10.7	7.3	15.9
			Sample	0.2	13	7.5	32
Armitage	Titan	%SAT	ID2	0.7	15.1	4.3	35.0
			OK	1.7	11.2	4.3	31.9
			Sample	0.3	15.9	4.3	63.5
Armitage	Titan	$\%TiO_2$	ID2	1.7	8.2	4.4	19.8
			OK	2.6	7.7	4.4	18.5
			Sample	0.2	8.3	4.4	32

ID2= inverse squared distance block estimates. OK = ordinary kriging block estimates. %CV = coefficient of variation, i.e. standard deviation divided by mean.

14.8. The Density of Mineralized Material

In the previous 2010-11 resource models, density measurements were scarce and the inference of the bulk density of mineralized blocks was based on a linear regression equation over %Satmagan, derived from those data (Davis Tube results for 38 SGS check samples confirmed by 31 pycnometer data in one drillhole for Southwest and 290 pycnometer data in Armitage).

As indicated in Section 14.1, there is now 697pycnometer data ranging from 2.68 to 4.63 t/m^3 (average 3.41 t/m^3) in the Southwest holes, and 776 pycnometer data ranging from 2.73 to 5.4 t/m^3 (average 3.34 t/m^3) in the Armitage holes. The 5.4t/m^3 maximum reported density in Armitage samples is an outlier that might need some review, although it is not directly used in the estimation of resource tonnages.

The improved systematic coverage in the density and WRA datasets has allowed BlackRock geologists to develop a density regression model based on a simplified modal mineralogy that is more robust than the Fe_2O_3 regression of the limited available data used in the 2010-2011 model (see Section 9.6). This new model uses %TiO_2 and %Fe_2O_3 data in addition to %Satmagan to derive a density of mineralization. More precisely, a simplified modal analysis converts grades into mineral concentrations with a fixed density for each mineral species/group.

The same density model applies across all units in both deposits. A comparison of Measured and Inferred density for mineralized samples in the four units of Southwest and Armitage (Table 14-17 and Figure 14-74) shows that the new density model provides a good prediction of grade density in all of the zones of the two deposits. Averages are very close and correlation coefficients remain around 0.9.

To apply the density model, a bulk density is Inferred in each mineralized block based on the ID2 estimates for %Satmagan, %TiO_2 and %Fe_2O_3 in that block. Another approach would be to first infer the density from grades in mineralized samples with no density value, and then interpolate the density in blocks from that complete sample dataset.

A test run on the 142,409 blocks in the BCS zone of Southwest shows very little differences between block density estimates from the two methods. The 142,409 densities directly Inferred from block grade estimates range from 3.06 to 4.07 t/m^3 (average 3.37 t/m^3) and have a coefficient of variation of 3.64%. The 142,409 interpolated densities range from 3.04 to 4.05 t/m^3 (average 3.37 t/m^3) and have a coefficient of variation of 3.66%. The correlation coefficient of the two block densities is R=0.97. The maximum relative differences range from -13.3% to +8.8%.

Table 14-17: Statistics of measured (pycnometer) & Inferred (from grades) sample densities

Deposit	Unit	# Data	Type	Min. (t/m3)	Average (t/m3)	Max. (t/m3)	Correlation Coefficient
Southwest	BCS	207	Measured	2.88	3.40	4.11	0.92
Southwest	BCS	207	Inferred	2.93	3.4	4.14	
Southwest	MC	114	Measured	2.68	3.75	4.39	0.79
Southwest	MC	114	Inferred	3.03	3.82	4.41	
Southwest	ULS	76	Measured	2.88	3.55	4.28	0.92
Southwest	ULS	76	Inferred	2.99	3.58	4.30	
Southwest	Titan	100	Measured	2.96	3.36	3.96	0.85
Southwest	Titan	100	Inferred	2.99	3.31	3.99	
Southwest	All	497	Measured	2.68	3.50	4.39	0.91
Southwest	All	497	Inferred	2.93	3.54	4.41	
Armitage	BCS	192	Measured	2.97	3.41	3.95	0.95
Armitage	BCS	192	Inferred	2.99	3.43	3.98	
Armitage	MC	112	Measured	3.00	3.58	4.21	0.94
Armitage	MC	112	Inferred	2.99	3.63	4.14	
Armitage	ULS	160	Measured	2.96	3.48	5.40	0.86
Armitage	ULS	160	Inferred	3.01	3.53	4.11	
Armitage	Titan	80	Measured	2.99	3.26	3.61	0.68
Armitage	Titan	80	Inferred	3.06	3.23	3.58	
Armitage	All	544	Measured	2.96	3.44	5.40	0.90
Armitage	All	544	Inferred	2.99	3.47	4.14	
All	All	1041	Measured	2.68	3.47	5.40	0.91
All	All	1041	Inferred	2.93	3.51	4.41	

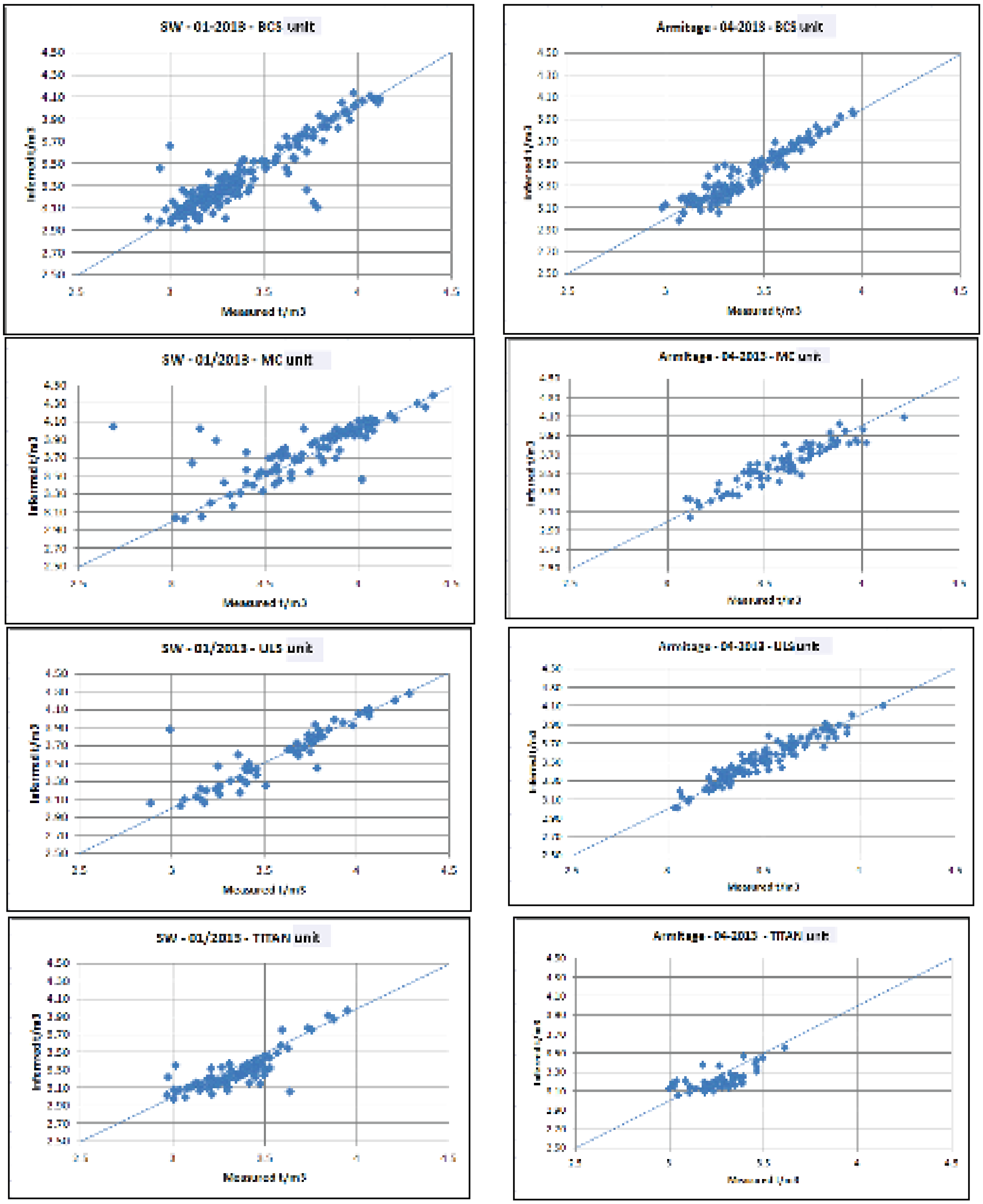


Figure 14-74: Measured (pycnometer) and Inferred (from grades) densities of samples

The qualitative fit of regression model is demonstrated by the tight clustering of these data to around +/-7% of the theoretical ideal, which is represented by the diagonal blue 1:1 line in the plots above. Complete statistics for each unit are provided in Table 14-17 above.

14.9. Categorization of Resources

14.9.1. Categorization in the Historical Resources of 2010-2011

The categorization of Southwest and Armitage estimated resource blocks conforms to the categorization performed in 2010-11. At the time, the northeastern end of Southwest, with drillholes on each section at 100 m, was classified in the Measured category, whereas the southwestern end of Southwest and the whole of Armitage, with only drillholes on sections at 200 m spacing, were assigned to the Indicated category. There was no material in the Inferred category, given the mineralized solids used at that time did not extend to more than 100 m from the deepest hole on each section, as well as more than 100 m away from the last section, on both extremities of the deposit.

14.9.2. Categorization in the Current Resource

The same rules are used to categorize the resources blocks of the new models. For the Measured category, a contour is drawn on each drill section, 50 m below the deepest hole. Those contours are connected to a "Measured solid" which is closed by faces on drill panels 50 and 2750 for Southwest, and Sections 150 and 2850 for Armitage. All the mineralized blocks within that solid are tagged as Measured.

Similarly, for the Indicated category, another contour is drawn on each drill panel 100 m below the deepest hole and those contours are connected to form an "Indicated solid", which is closed at its extremities by faces at drill panels 0 and 2800 for Southwest, and drill panels 100 and 2900 for Armitage. All the mineralized blocks within that solid are tagged as Indicated.

Finally, all blocks which are more than 100 m away from drillholes and not included within the Measured or Indicated solids are tagged as Inferred.

These rules are consistent with the CIM guidelines for resource classification. A strong geological continuity is demonstrated by the new geological model, with layered mineralized zones corresponding to a differential settling of minerals in a cooling magma chamber. However, within the mineralized stratigraphic units, the grade continuity is not as strong.

This is expected in mineralization that is constrained by a fairly tight geological framework. In other words, the grade continuity expressed in alternating bands of high and low grades no longer shows when one simply looks at grades in high grade bands. Nevertheless, all of the zone variograms of principal variables show some grade correlation over distances of at least 100 m, which supports the rationale for Measured and Indicated categorization.

The current drill spacing and sample density is sufficient to determine, with a high level of confidence, the grade distribution predicted by the M&I categorization. An empirical comparison was made between the results of the 2010/2011 models and those of the current 2013 model (see Section 14.10.3). The data strongly supports the current categorization with the vast majority of blocks reporting to the Measured.

14.9.2.1. Southwest

Block resource categorization is illustrated for Southwest drill panels in Figure 14-75 to Figure 14-78 and for benches in Figure 14-79.

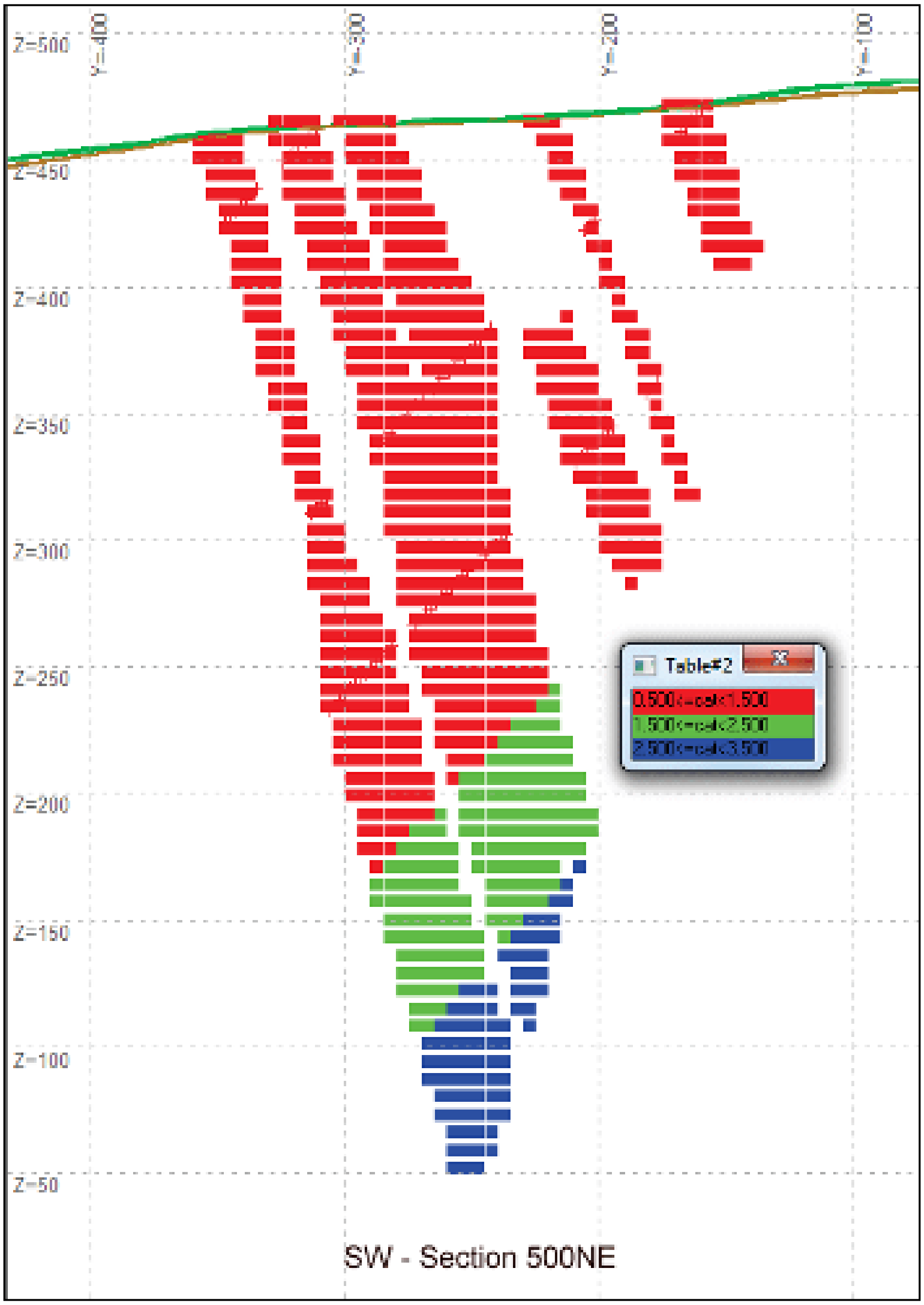


Figure 14-75: Southwest drill panel 500 showing resource categorization. Looking NE

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred. Mineralized samples in the same section corridors are shown with a "+".

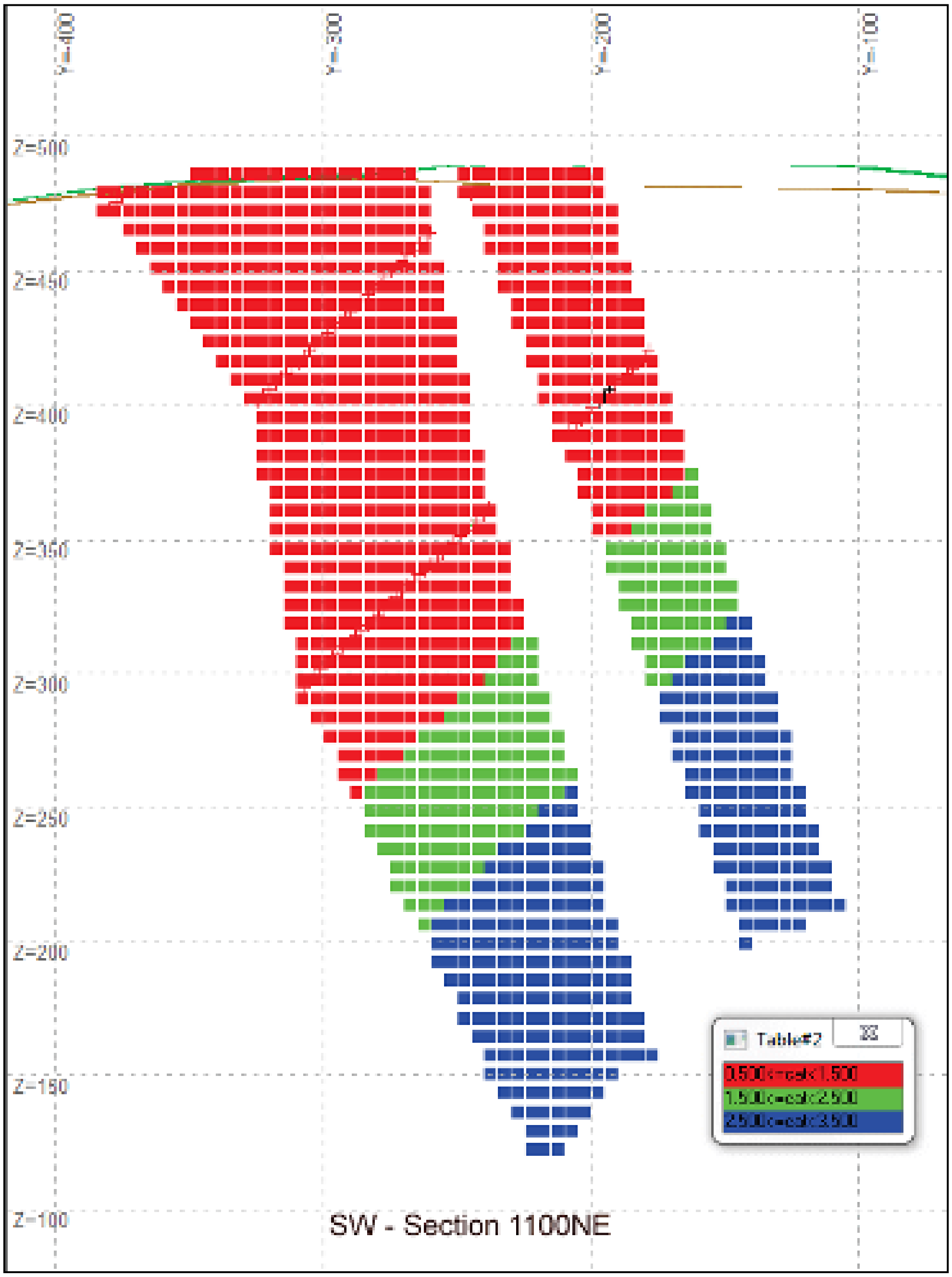


Figure 14-76: Southwest drill panel 1100 showing resource categorization. Looking NE

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred. Mineralized samples in the same section corridors are shown with a "+".

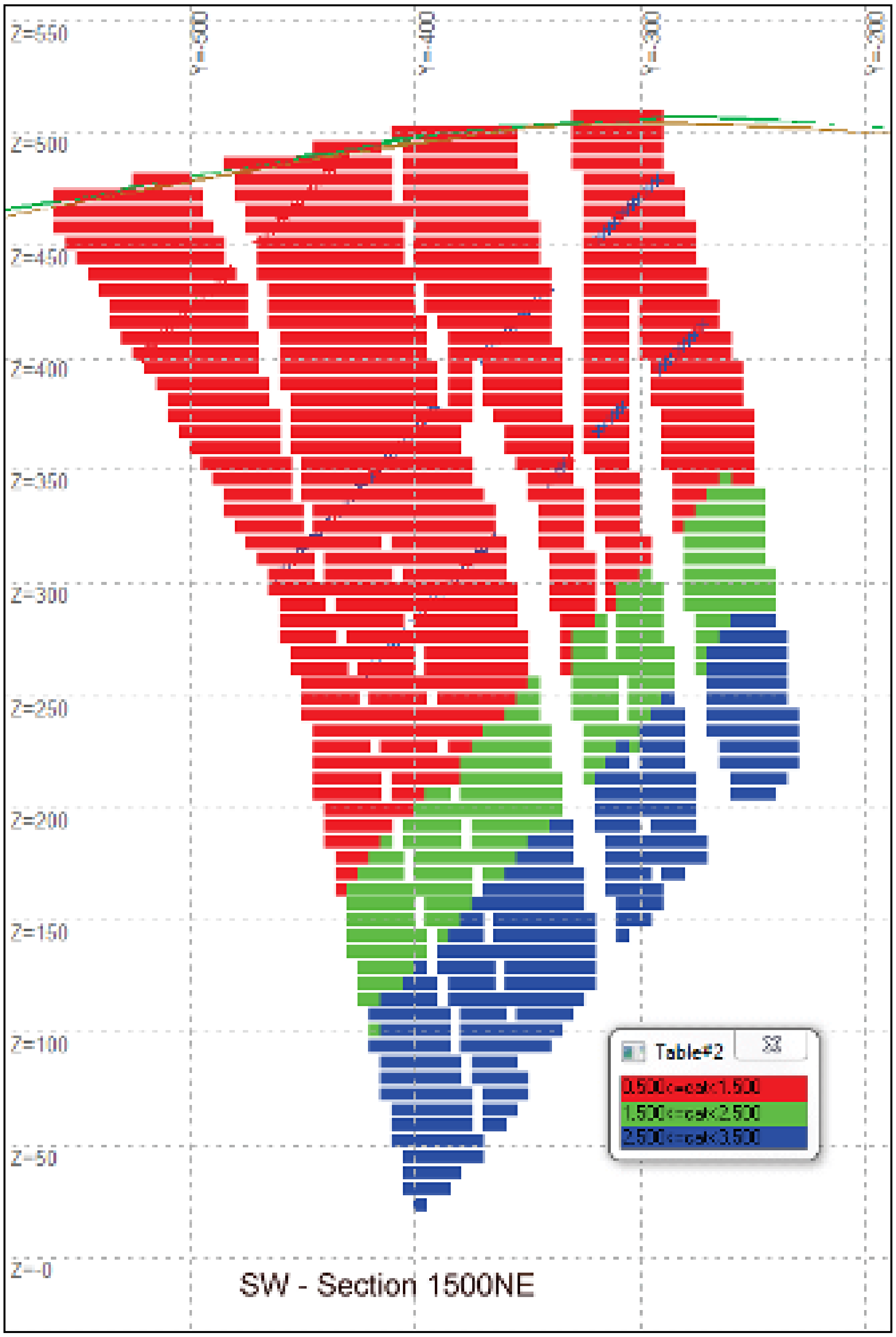


Figure 14-77: Southwest drill panel 1500 showing resource categorization. Looking NE

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred. Mineralized samples in the same section corridors are shown with a “+”.

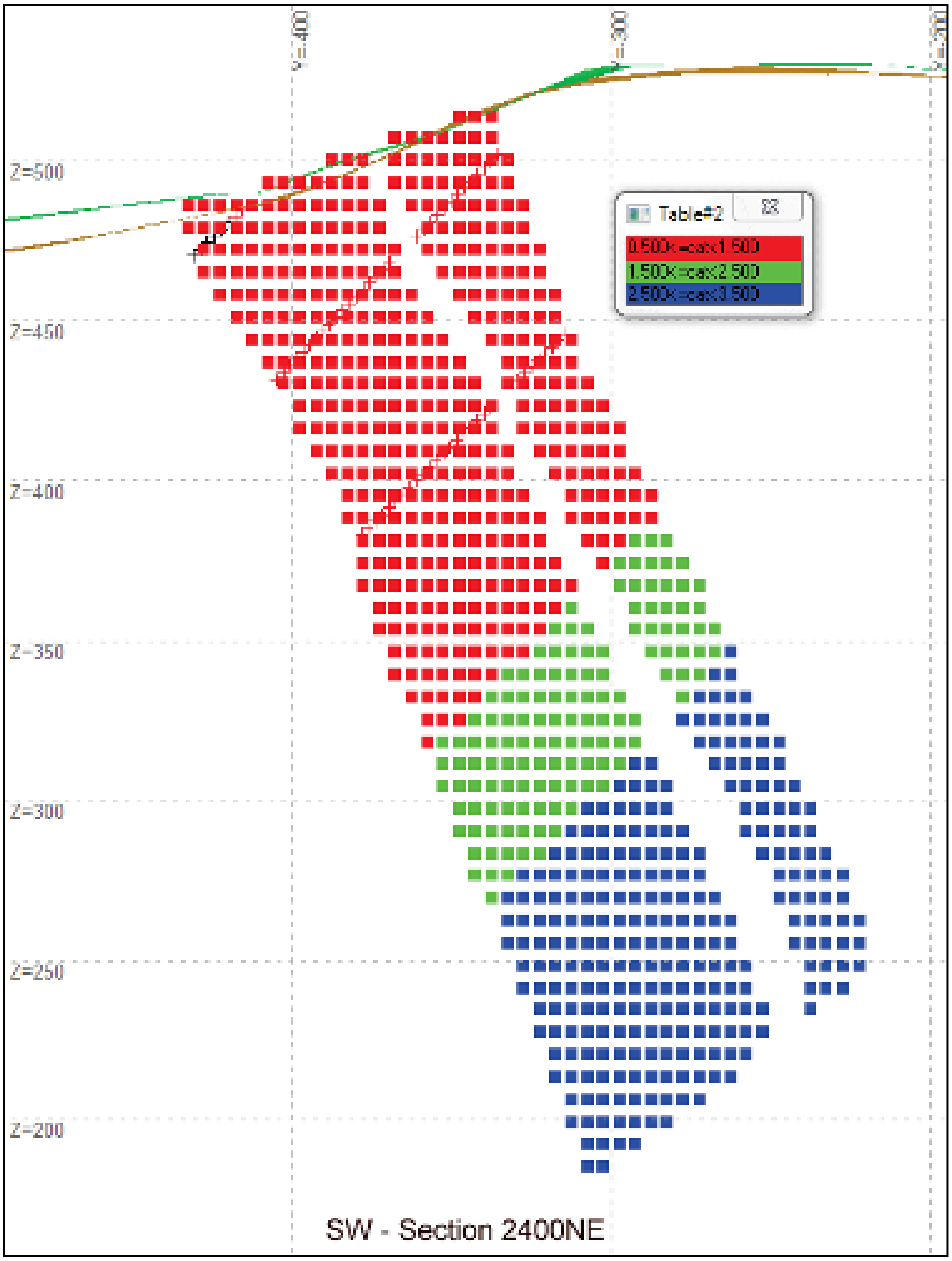


Figure 14-78: Southwest drill panel 2400 showing resource categorization. Looking NE

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred. Mineralized samples in the same section corridors are shown with a "+".

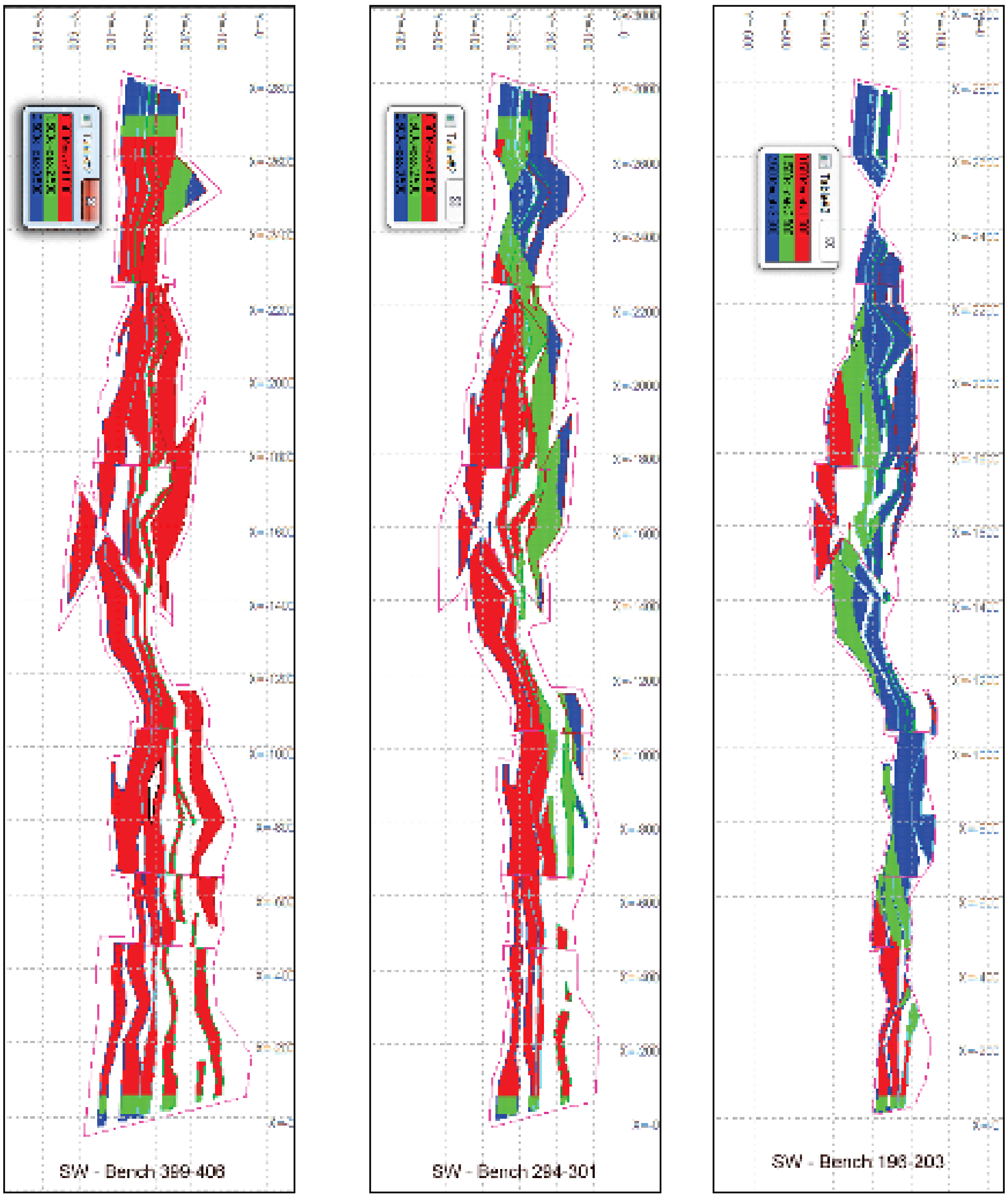


Figure 14-79: Southwest bench plans showing resource categorization

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred.

14.9.2.2. Armitage Deposit

Examples of the Armitage drill panels Resource categorization are provided in Figure 14-80 to Figure 14-83 and in several benches in Figure 14-84.

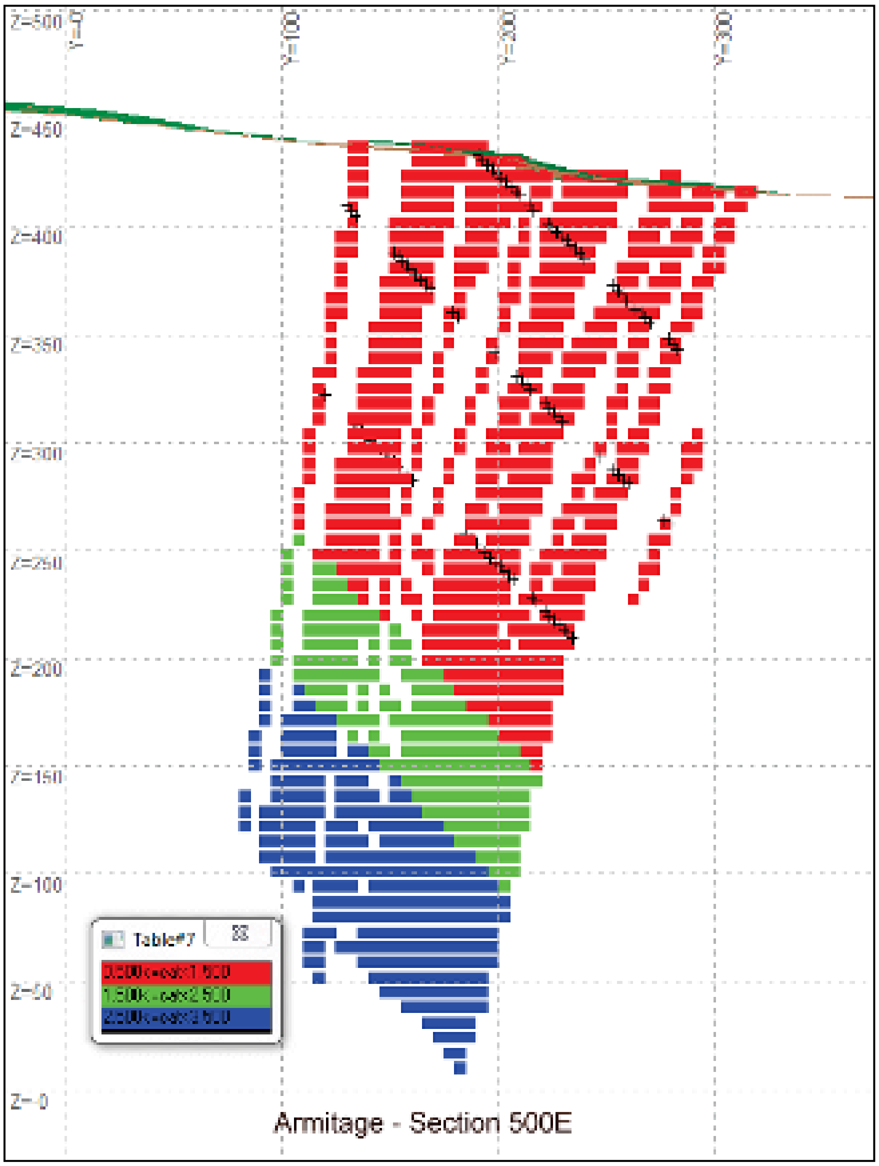


Figure 14-80: Armitage drill panel 500 showing resource categorization. Looking WSW

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred. Mineralized samples in the same section corridors are shown with a "+".

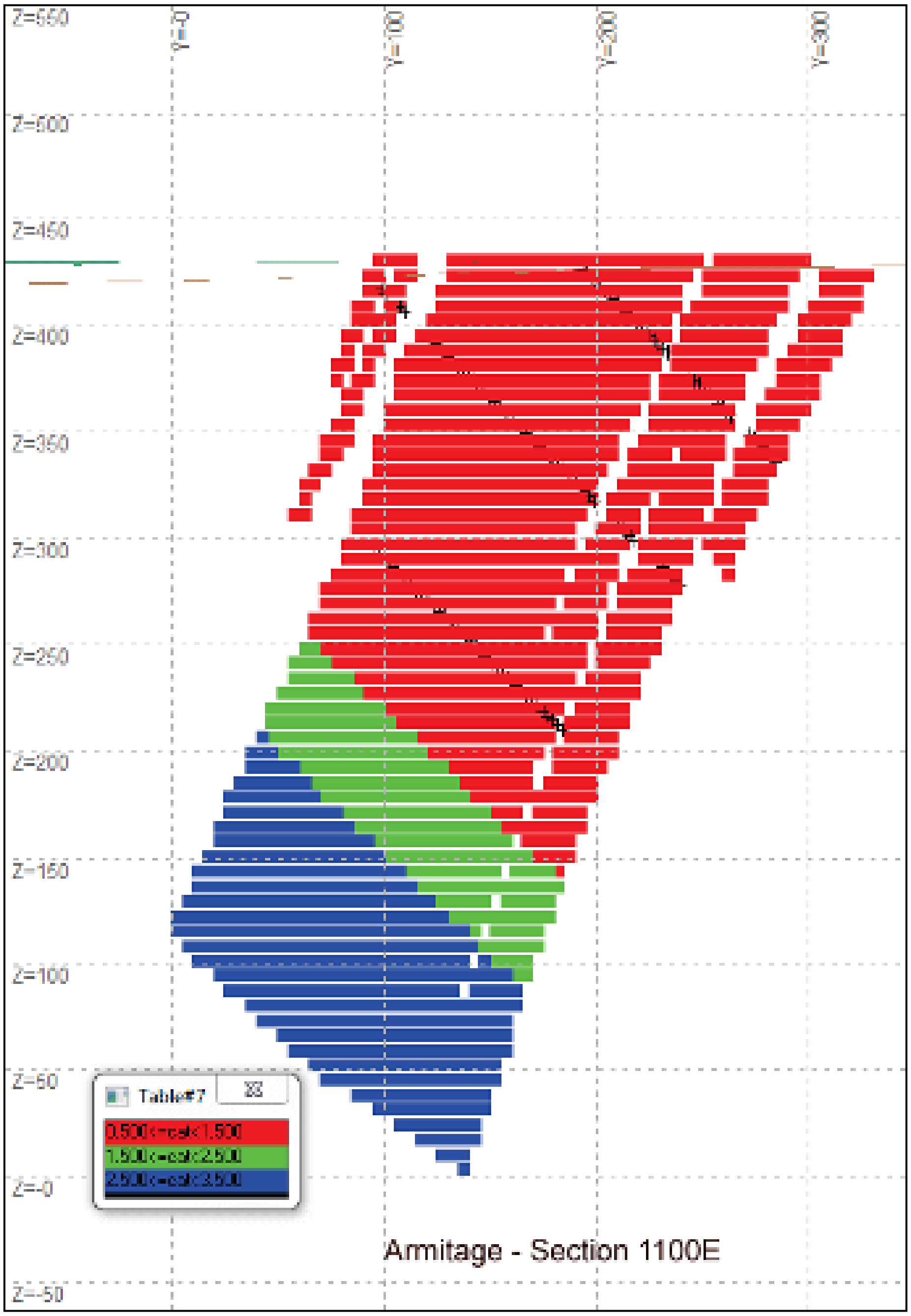


Figure 14-81: Armitage drill panel 1100 showing resource categorization. Looking WSW

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred. Mineralized samples in the same section corridors are shown with a "+".

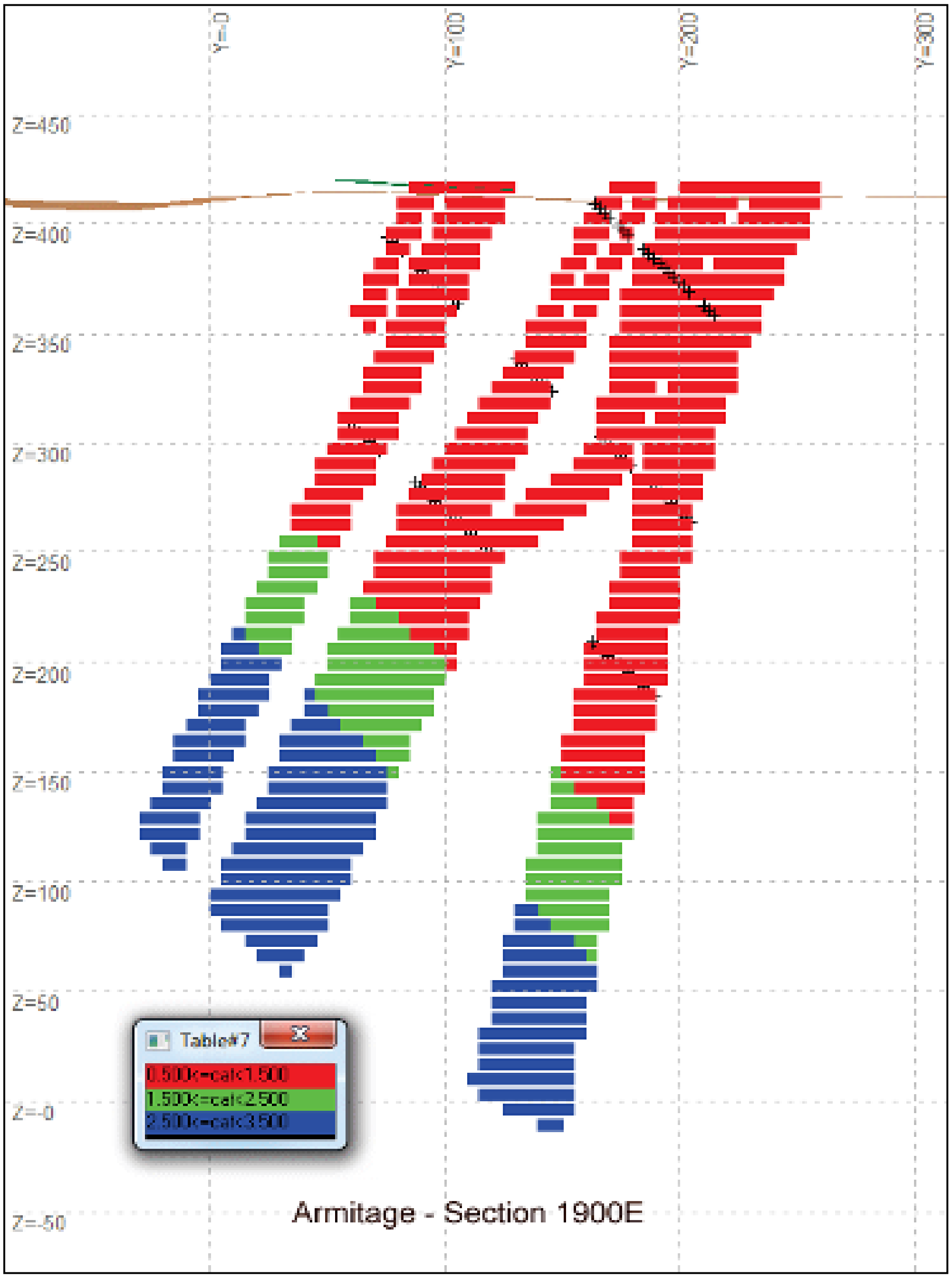


Figure 14-82: Armitage drill panel 1900 showing resource categorization. Looking WSW

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred. Mineralized samples in the same section corridors are shown with a "+".

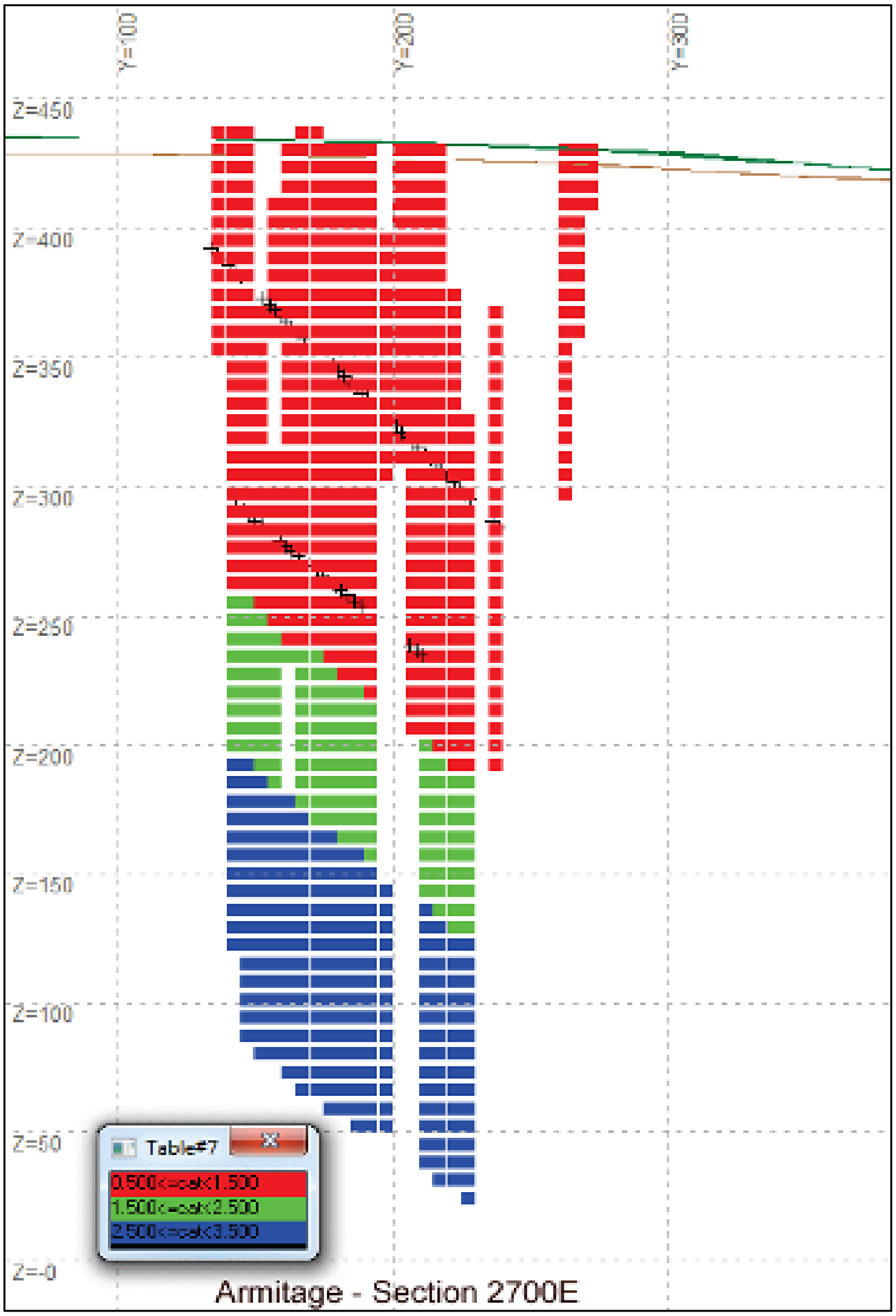


Figure 14-83: Armitage drill panel 2700 showing resource categorization. Looking WSW

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred. Mineralized samples in the same section corridors are shown with a "+".

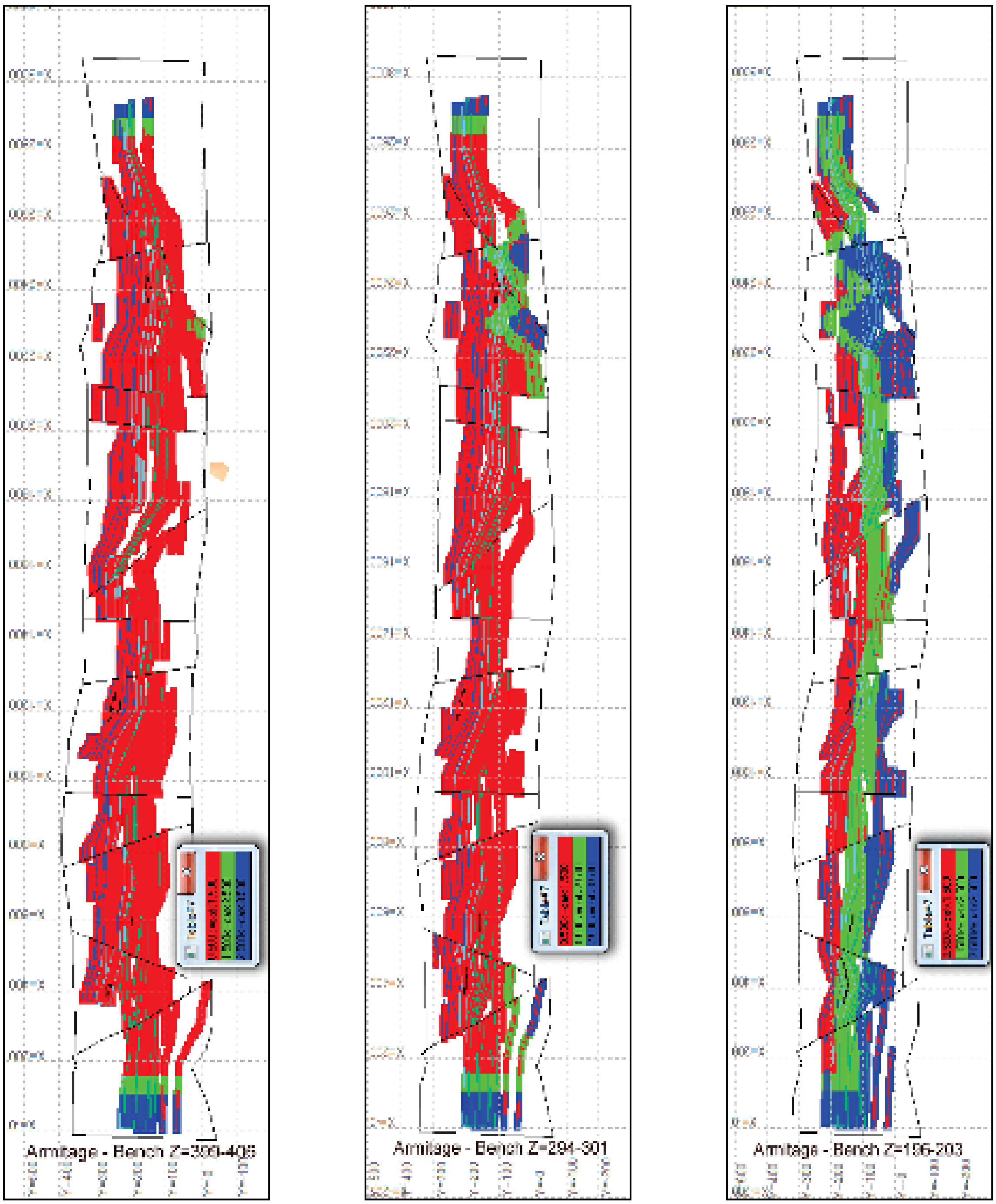


Figure 14-84: Armitage bench plans with resource categorization

Cat=1 (red) = Measured. Cat = 2 (green) = Indicated. Cat=3 (blue) = Inferred.

14.10. Mineral Inventory

A mineral inventory represents a compilation of block data according to category and cut-offs on any of the estimated block values. There are 325,805 blocks 10 x5 x 7 m (350 m^3) in Southwest and 376,169 blocks of the same size in Armitage. Estimates of the %Satmagan, %Fe_2O_3, %TiO_2, %V_2O_5, %Al_2O_3, %P_2O_5 and %S, as well as the density of mineralized material, have been assigned to each block. A categorization tag (Measured, Indicated or Inferred) has also been assigned to each block.

Values of the mineral inventories can be interpreted as "unconstrained resources", i.e., selected blocks are not restricted by any mining limits. True resources (see next section) must be limited by an optimized pit shell in order to demonstrate their "reasonable prospect of economic extraction".

14.10.1. Mineral Inventory of the Southwest Deposit

The mineral inventory of the Southwest deposit at %Satmagan cut-offs from 2% to 20% is found in Table 14-18. Graphs showing the variations of tonnage, %Satmagan and %TiO_2 with the Satmagan cut-off are found in Figure 14-85 and Figure 14-86.

Table 14-18: Mineral inventory of the Southwest Deposit with a %Satmagan cut-off

Cut-off %Sat	Cat.	Vol. (Mm^3)	Tonnes (Mt)	Dens. (t/m^3)	%Fe_2O_3	%TiO_2	%V_2O_5	%Sat	%Al_2O_3	%P_2O_5	%S
2.00	Meas	68.34	237.65	3.48	35.27	6.45	0.376	13.45	13.94	0.036	0.175
2.00	Ind	19.07	66.23	3.47	35.07	6.46	0.358	13.06	13.58	0.037	0.195
2.00	**M+I**	**87.41**	**303.88**	**3.48**	**35.23**	**6.45**	**0.372**	**13.37**	**13.86**	**0.036**	**0.179**
2.00	Inf	24.13	83.76	3.47	35.06	6.51	0.351	12.93	13.36	0.038	0.195
4.00	Meas	64.17	224.34	3.50	35.95	6.60	0.392	14.06	13.98	0.036	0.171
4.00	Ind	17.79	62.10	3.49	35.73	6.60	0.376	13.71	13.67	0.036	0.190
4.00	**M+I**	**81.96**	**286.44**	**3.49**	**35.90**	**6.60**	**0.389**	**13.98**	**13.91**	**0.036**	**0.175**
4.00	Inf	22.07	77.16	3.50	35.90	6.70	0.373	13.76	13.47	0.037	0.188
6.00	Meas	55.77	197.19	3.54	37.45	6.93	0.423	15.30	13.87	0.034	0.165
6.00	Ind	15.32	54.08	3.53	37.06	6.89	0.412	15.01	13.78	0.034	0.177
6.00	**M+I**	**71.09**	**251.27**	**3.53**	**37.37**	**6.92**	**0.421**	**15.24**	**13.85**	**0.034**	**0.168**
6.00	Inf	18.73	66.24	3.54	37.28	6.99	0.416	15.24	13.70	0.035	0.171

Cut-off %Sat	Cat.	Vol. (Mm^3)	Tonnes (Mt)	Dens. (t/m^3)	$\%Fe_2O_3$	$\%TiO_2$	$\%V_2O_5$	%Sat	$\%Al_2O_3$	$\%P_2O_5$	%S
8.00	Meas	47.88	171.37	3.58	39.18	7.35	0.451	16.55	13.52	0.033	0.166
8.00	Ind	13.13	46.91	3.57	38.74	7.31	0.439	16.24	13.44	0.033	0.176
8.00	**M+I**	**61.01**	**218.28**	**3.58**	**39.09**	**7.34**	**0.448**	**16.48**	**13.50**	**0.033**	**0.168**
8.00	Inf	15.87	56.88	3.58	39.15	7.46	0.444	16.57	13.34	0.032	0.169
10.00	Meas	40.08	145.43	3.63	41.14	7.86	0.479	17.90	13.05	0.033	0.170
10.00	Ind	10.63	38.57	3.63	40.87	7.86	0.472	17.81	12.99	0.032	0.176
10.00	**M+I**	**50.71**	**184.00**	**3.63**	**41.08**	**7.86**	**0.478**	**17.88**	**13.04**	**0.033**	**0.171**
10.00	Inf	13.21	48.01	3.63	41.11	7.98	0.472	17.96	12.88	0.032	0.171
12.00	Meas	32.69	120.36	3.68	43.23	8.41	0.507	19.35	12.51	0.032	0.175
12.00	Ind	8.88	32.63	3.67	42.68	8.35	0.497	19.05	12.52	0.031	0.180
12.00	**M+I**	**41.57**	**152.99**	**3.68**	**43.11**	**8.40**	**0.505**	**19.29**	**12.51**	**0.032**	**0.176**
12.00	Inf	10.98	40.46	3.68	43.02	8.51	0.497	19.27	12.35	0.030	0.177
14.00	Meas	26.65	99.43	3.73	45.16	8.94	0.531	20.69	12.00	0.031	0.181
14.00	Ind	7.46	27.73	3.72	44.27	8.80	0.516	20.11	12.07	0.030	0.187
14.00	**M+I**	**34.11**	**127.16**	**3.73**	**44.97**	**8.91**	**0.528**	**20.56**	**12.02**	**0.031**	**0.182**
14.00	Inf	9.19	34.25	3.73	44.74	9.01	0.518	20.41	11.85	0.029	0.185
16.00	Meas	21.65	81.69	3.77	46.77	9.35	0.555	21.93	11.60	0.030	0.182
16.00	Ind	6.21	23.27	3.75	45.48	9.09	0.538	21.08	11.82	0.030	0.185
16.00	**M+I**	**27.86**	**104.96**	**3.77**	**46.48**	**9.29**	**0.551**	**21.74**	**11.65**	**0.030**	**0.183**
16.00	Inf	7.84	29.46	3.76	45.93	9.31	0.537	21.28	11.56	0.029	0.185
18.00	Meas	17.06	65.07	3.81	48.31	9.73	0.580	23.19	11.22	0.029	0.182
18.00	Ind	4.72	17.86	3.79	47.10	9.45	0.567	22.34	11.46	0.029	0.180
18.00	**M+I**	**21.78**	**82.93**	**3.81**	**48.05**	**9.67**	**0.577**	**23.01**	**11.27**	**0.029**	**0.182**
18.00	Inf	6.12	23.21	3.79	47.28	9.53	0.569	22.42	11.29	0.029	0.178
20.00	Meas	12.91	49.77	3.86	49.76	10.11	0.603	24.48	10.83	0.029	0.181
20.00	Ind	3.40	13.03	3.83	48.55	9.85	0.589	23.57	11.07	0.029	0.180
20.00	**M+I**	**16.31**	**62.80**	**3.85**	**49.51**	**10.06**	**0.600**	**24.29**	**10.88**	**0.029**	**0.181**
20.00	Inf	4.52	17.29	3.83	48.65	9.83	0.594	23.59	10.97	0.028	0.175

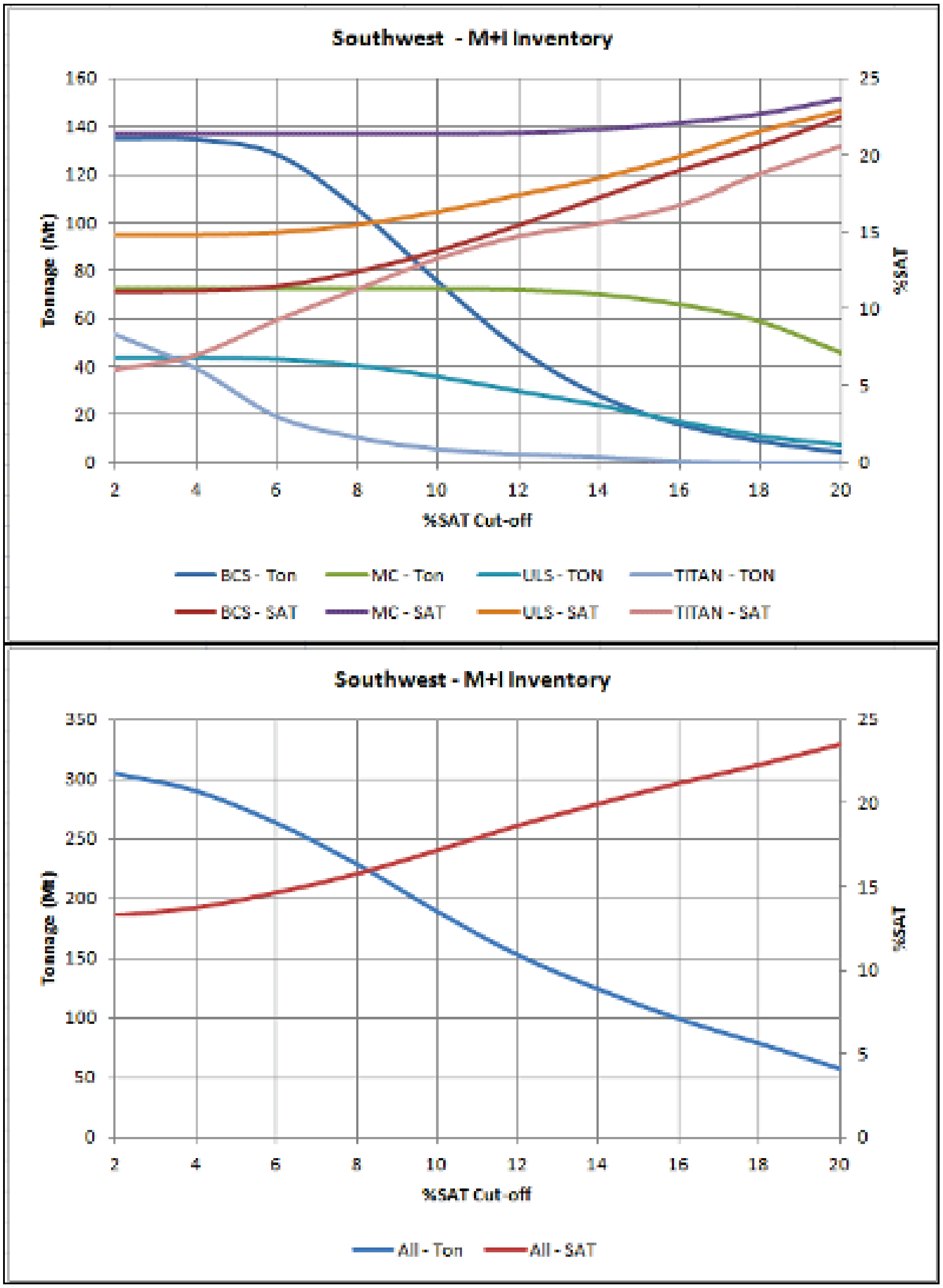
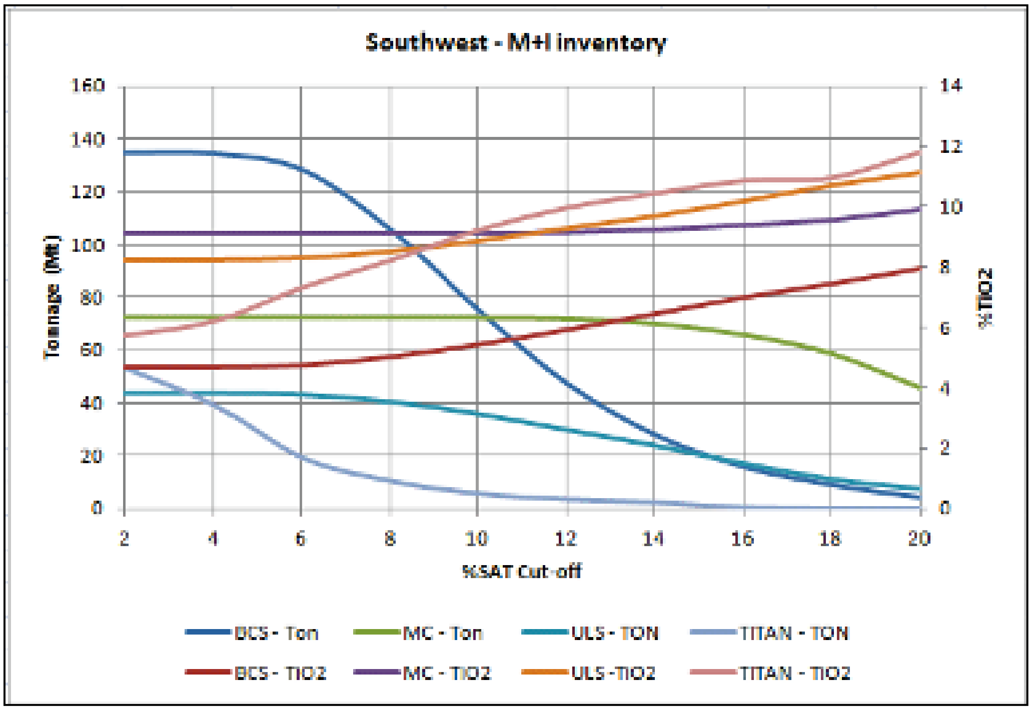


Figure 14-85: SAT Mineral inventory of the Southwest Deposit with a %Satmagan cut-off



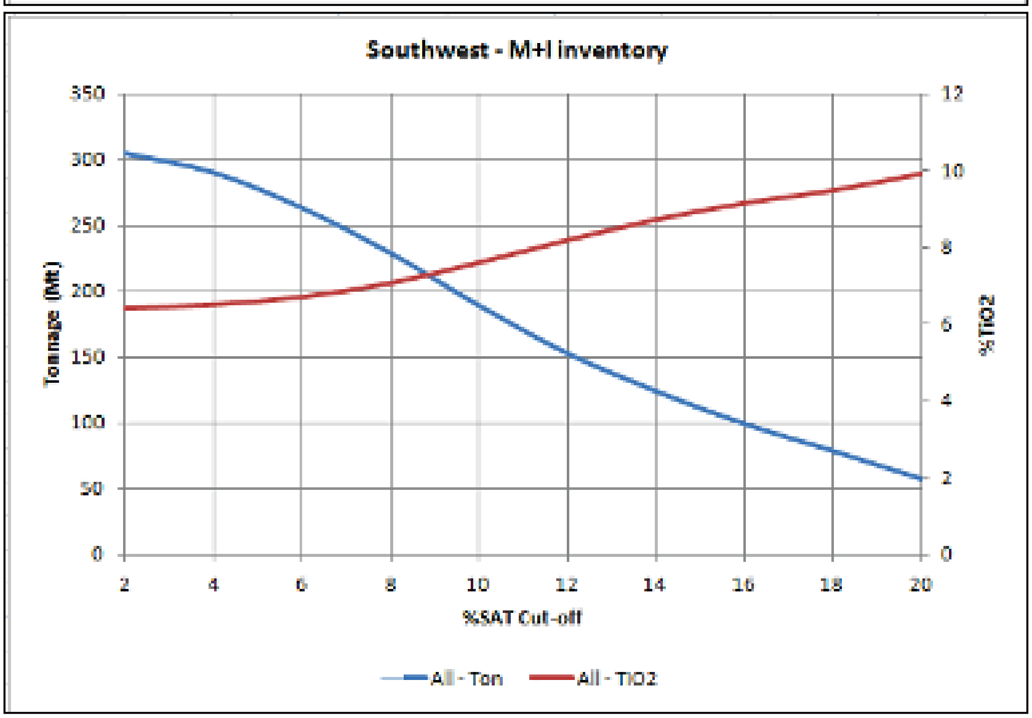


Figure 14-86: TiO_2 Mineral inventory of the Southwest Deposit with a %Satmagan cut-off

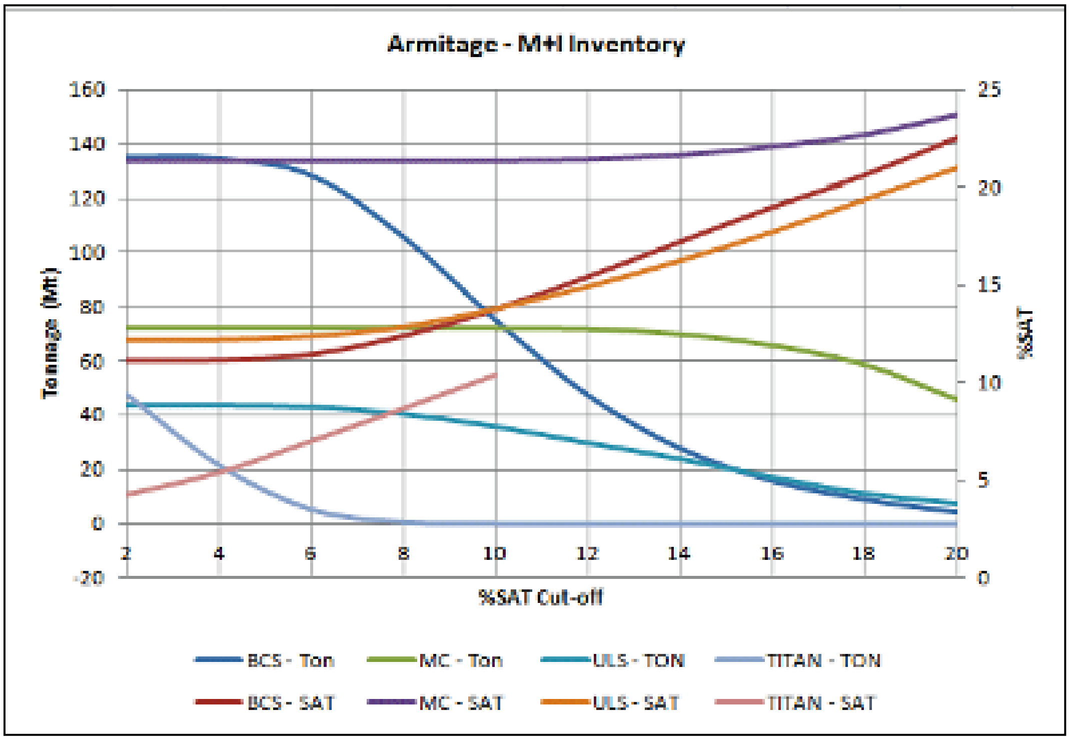
14.10.2. Mineral Inventory of the Armitage Deposit

The mineral inventory of the Armitage deposit at %Satmagan cut-offs from 2% to 20% is found in Table 14-19. Graphs showing the variations of tonnage, %Satmagan and %TiO_2 with the Satmagan cut-off are found in Figure 14-87 and Figure 14-88.

Table 14-19: Mineral inventory of the Armitage Deposit with a %Satmagan cut-off

Cut-off %Sat	Cat.	Vol. (Mm^3)	Tonnes (Mt)	Dens. (t/m^3)	%Fe_2O_3	%TiO_2	%V_2O_5	%Sat	%Al_2O_3	%P_2O_5	%S
2.00	Meas	69.39	238.83	3.44	32.99	6.13	0.334	12.08	14.67	0.029	0.279
2.00	Ind	21.65	74.03	3.42	32.04	5.94	0.315	11.30	14.59	0.029	0.285
2.00	**M+I**	**91.04**	**312.86**	**3.44**	**32.77**	**6.09**	**0.330**	**11.90**	**14.65**	**0.029**	**0.280**
2.00	Inf	39.16	133.87	3.42	31.96	5.95	0.312	11.22	14.57	0.028	0.293
4.00	Meas	63.33	219.46	3.47	33.73	6.33	0.356	12.86	14.81	0.029	0.271
4.00	Ind	19.29	66.50	3.45	32.84	6.16	0.341	12.22	14.81	0.028	0.274
4.00	**M+I**	**82.62**	**285.96**	**3.46**	**33.52**	**6.29**	**0.353**	**12.71**	**14.81**	**0.029**	**0.272**
4.00	Inf	35.46	122.06	3.44	32.68	6.16	0.334	11.99	14.76	0.028	0.283
6.00	Meas	57.40	200.28	3.49	34.54	6.53	0.375	13.62	14.84	0.028	0.268
6.00	Ind	17.33	60.11	3.47	33.54	6.33	0.363	12.99	14.97	0.028	0.269
6.00	**M+I**	**74.73**	**260.39**	**3.48**	**34.31**	**6.48**	**0.372**	**13.47**	**14.87**	**0.028**	**0.268**
6.00	Inf	32.13	111.22	3.46	33.24	6.30	0.354	12.68	14.95	0.027	0.277
8.00	Meas	50.34	177.09	3.52	35.60	6.79	0.394	14.48	14.61	0.028	0.271
8.00	Ind	15.31	53.47	3.49	34.40	6.54	0.381	13.74	14.86	0.028	0.268
8.00	**M+I**	**65.65**	**230.56**	**3.51**	**35.32**	**6.73**	**0.391**	**14.31**	**14.67**	**0.028**	**0.270**
8.00	Inf	27.90	97.29	3.49	34.15	6.53	0.375	13.49	14.91	0.027	0.276
10.00	Meas	42.39	150.44	3.55	36.78	7.06	0.414	15.45	14.32	0.028	0.270
10.00	Ind	12.67	44.65	3.52	35.52	6.80	0.399	14.67	14.55	0.028	0.271
10.00	**M+I**	**55.06**	**195.09**	**3.54**	**36.49**	**7.00**	**0.411**	**15.27**	**14.37**	**0.028**	**0.270**
10.00	Inf	22.77	80.08	3.52	35.27	6.78	0.394	14.43	14.59	0.028	0.279
12.00	Meas	33.93	121.52	3.58	38.03	7.34	0.435	16.51	14.00	0.028	0.267
12.00	Ind	9.63	34.25	3.56	36.88	7.12	0.421	15.78	14.18	0.028	0.274
12.00	**M+I**	**43.56**	**155.77**	**3.58**	**37.78**	**7.29**	**0.432**	**16.35**	**14.04**	**0.028**	**0.269**

Cut-off %Sat	Cat.	Vol. (Mm^3)	Tonnes (Mt)	Dens. (t/m^3)	$\%Fe_2O_3$	$\%TiO_2$	$\%V_2O_5$	%Sat	$\%Al_2O_3$	$\%P_2O_5$	%S
12.00	Inf	17.10	60.73	3.55	36.64	7.11	0.415	15.54	14.19	0.028	0.284
14.00	Meas	24.61	89.10	3.62	39.48	7.65	0.458	17.78	13.66	0.028	0.264
14.00	Ind	6.62	23.80	3.60	38.29	7.42	0.445	17.00	13.81	0.029	0.267
14.00	**M+I**	**31.23**	**112.90**	**3.62**	**39.23**	**7.60**	**0.455**	**17.62**	**13.69**	**0.028**	**0.265**
14.00	Inf	11.53	41.41	3.59	38.06	7.44	0.439	16.71	13.83	0.028	0.284
16.00	Meas	16.35	59.87	3.66	40.95	7.98	0.482	19.15	13.32	0.027	0.262
16.00	Ind	4.08	14.82	3.63	39.61	7.71	0.468	18.21	13.50	0.029	0.261
16.00	**M+I**	**20.43**	**74.69**	**3.66**	**40.68**	**7.93**	**0.479**	**18.96**	**13.36**	**0.027**	**0.262**
16.00	Inf	6.34	23.01	3.63	39.62	7.80	0.464	18.10	13.47	0.028	0.277
18.00	Meas	9.80	36.27	3.70	42.52	8.31	0.507	20.59	12.97	0.027	0.259
18.00	Ind	1.87	6.86	3.67	41.21	8.03	0.497	19.71	13.09	0.030	0.242
18.00	**M+I**	**11.67**	**43.13**	**3.70**	**42.31**	**8.27**	**0.505**	**20.45**	**12.99**	**0.027**	**0.256**
18.00	Inf	2.64	9.69	3.68	41.23	8.11	0.495	19.74	13.04	0.029	0.255
20.00	Meas	5.12	19.16	3.74	44.02	8.61	0.530	22.07	12.60	0.026	0.253
20.00	Ind	0.65	2.42	3.71	42.64	8.19	0.525	21.30	12.74	0.031	0.216
20.00	**M+I**	**5.77**	**21.58**	**3.74**	**43.87**	**8.56**	**0.529**	**21.98**	**12.62**	**0.027**	**0.249**
20.00	Inf	0.92	3.42	3.72	42.85	8.32	0.526	21.37	12.60	0.031	0.239



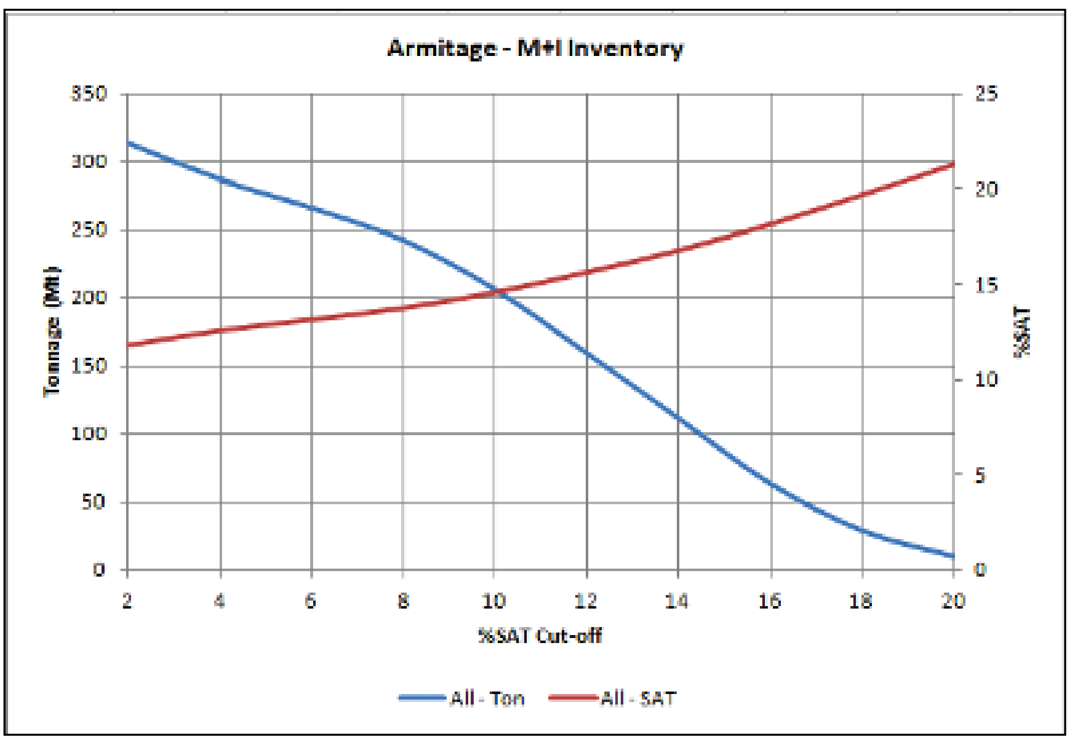


Figure 14-87: SAT Mineral inventory of the Armitage Deposit with a %Satmagan cut-off

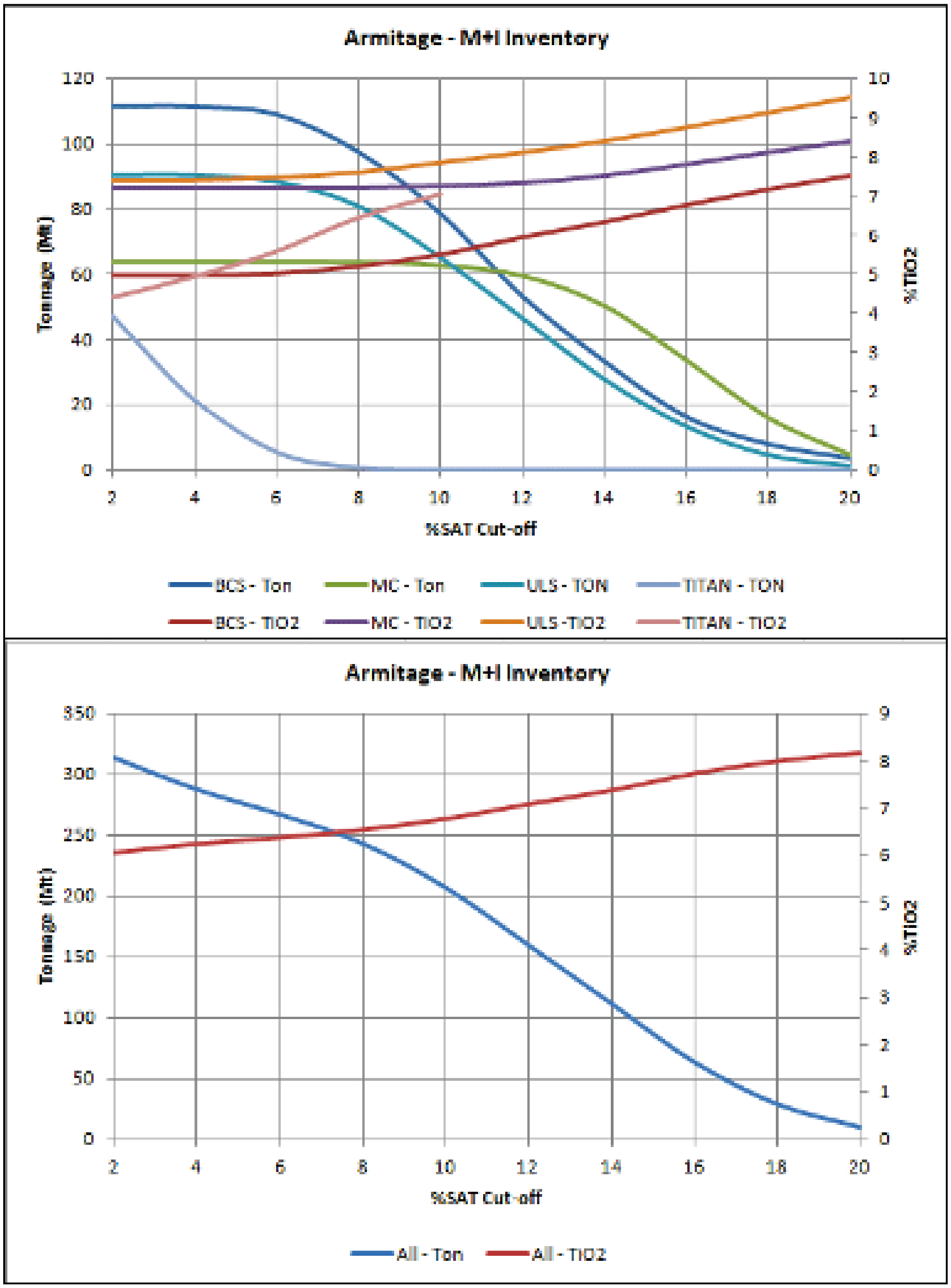


Figure 14-88: TiO_2 Mineral inventory of the Armitage Deposit with a %Satmagan cut-off

14.10.3. Mineral Inventory Comparison

The overall M+I mineral inventory evaluated at various %Satmagan cut-offs has not changed much, despite the twofold increase (in fact nearly a threefold increase in the case of Armitage) of the DH information between 2011 and 2013.

Southwest M+I inventory at various %Satmagan cut-offs is compared in Table 14-20, with visual support provided at the top of Figure 14-89.

In the probable economic cut-off range for %Satmagan (from 4% to 14%), the maximum difference in Southwest is less than 10% for Tonnage, $\%TiO_2$, %Satmagan and most other parameters modelled. The only element having a difference of more than 10% is sulphur, which is more than 30% higher in the 2013 model.

The Armitage M+I inventory, at various %Satmagan cut-offs, is compared in Table 14-21, with a graph at the bottom of Figure 14-89. At %Satmagan cut-offs between 4% and 14%, the maximum tonnage difference is less than 10%, with %Satmagan and $\%TiO_2$ grade differences even narrower, at ~5% or less. The only element with percent differences of more than 10% is phosphorus, which is more than 25% higher in the 2011 model.

Table 14-20: Comparison of 2011 and 2013 M+I Inventories for the Southwest Deposit

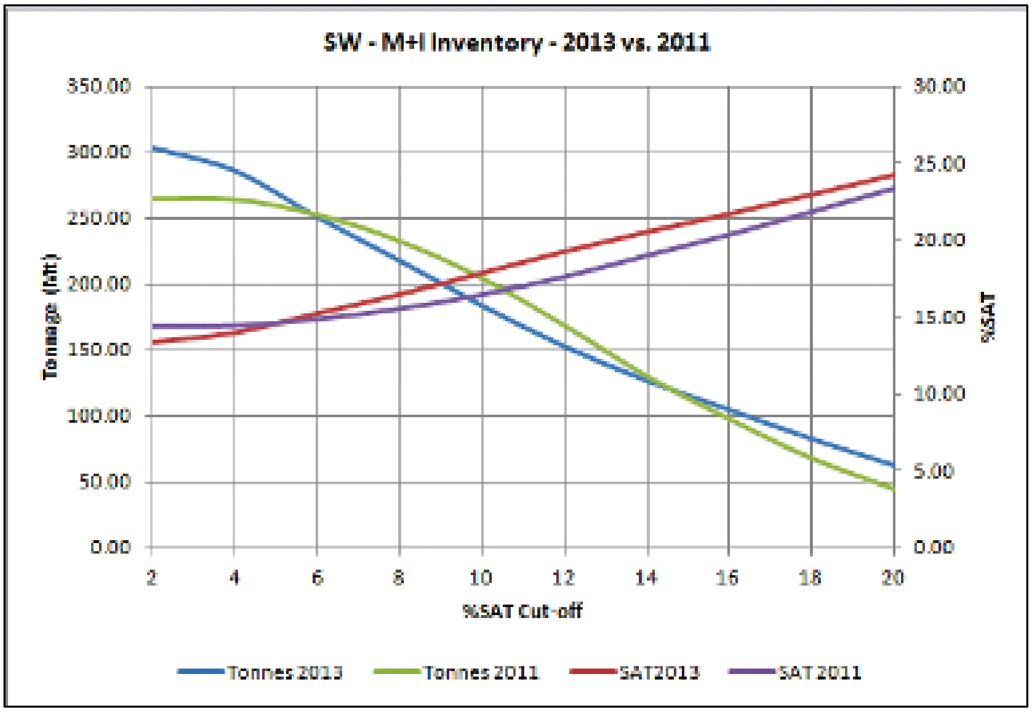
Dep.	Year	Cat	Cut-off (%Sat)	Vol. (Mm^3)	Ton.	Density (t/m^3)	TiO_2 (%)	Sat (%)	Al_2O_3 (%)	P_2O_5 (%)	S (%)
SW	2013	M+I	2	87.41	303.88	3.48	6.45	13.37	13.86	0.036	0.18
SW	2013	M+I	4	81.96	286.44	3.49	6.60	13.98	13.91	0.036	0.18
SW	2013	M+I	6	71.09	251.27	3.53	6.92	15.24	13.85	0.034	0.17
SW	2013	M+I	8	61.01	218.28	3.58	7.34	16.48	13.50	0.033	0.17
SW	2013	M+I	10	*50.71*	184.00	3.63	7.86	17.88	13.04	0.033	0.17
SW	2013	M+I	12	41.57	152.99	3.68	8.40	19.29	12.51	0.032	0.18
SW	2013	M+I	14	34.11	127.16	3.73	8.91	20.56	12.02	0.031	0.18
SW	2013	M+I	16	27.86	104.96	3.77	9.29	21.74	11.65	0.030	0.18
SW	2013	M+I	18	21.78	82.93	3.81	9.67	23.01	11.27	0.029	0.18
SW	2013	M+I	20	16.31	62.80	3.85	10.06	24.29	10.88	0.029	0.18
SW	2011	M+I	2	77.22	264.54	3.43	6.49	14.45	14.45	0.036	0.12
SW	2011	M+I	4	77.04	263.98	3.43	6.50	14.47	14.44	0.036	0.12
SW	2011	M+I	6	73.40	252.38	3.44	6.64	14.90	14.35	0.036	0.12

Dep.	Year	Cat	Cut-off (%Sat)	Vol. (Mm^3)	Ton.	Density (t/m^3)	TiO_2 (%)	Sat (%)	Al_2O_3 (%)	P_2O_5 (%)	S (%)
SW	2011	M+I	8	67.25	232.50	3.46	6.89	15.57	14.14	0.036	0.12
SW	2011	M+I	10	58.65	204.25	3.48	7.24	16.48	13.82	0.035	0.12
SW	2011	M+I	12	47.95	168.53	3.51	7.65	17.63	13.49	0.035	0.12
SW	2011	M+I	14	36.60	130.03	3.55	8.15	19.01	13.10	0.035	0.11
SW	2011	M+I	16	27.21	97.66	3.59	8.68	20.36	12.66	0.035	0.11
SW	2011	M+I	18	18.74	68.04	3.63	9.29	21.83	12.11	0.035	0.12
SW	2011	M+I	20	12.18	44.72	3.67	9.93	23.34	11.54	0.036	0.12
SW	Diff	M+I	2	12.4%	13.8%	1.3%	-0.6%	-7.8%	-4.2%	0.6%	39.7%
SW	Diff	M+I	4	6.2%	8.2%	1.9%	1.5%	-3.4%	-3.7%	0.0%	37.4%
SW	Diff	M+I	6	-3.2%	-0.4%	2.7%	4.1%	2.2%	-3.5%	-5.7%	33.1%
SW	Diff	M+I	8	-9.7%	-6.3%	3.3%	6.3%	5.7%	-4.6%	-8.7%	33.4%
SW	Diff	M+I	10	-14.5%	-10.4%	4.2%	8.2%	8.2%	-5.8%	-6.5%	35.2%
SW	Diff	M+I	12	-14.3%	-9.7%	4.7%	9.3%	9.0%	-7.5%	-9.6%	37.9%
SW	Diff	M+I	14	-7.1%	-2.2%	4.9%	8.9%	7.9%	-8.6%	-12.8%	49.5%
SW	Diff	M+I	16	2.4%	7.2%	4.8%	6.8%	6.6%	-8.3%	-15.4%	49.7%
SW	Diff	M+I	18	15.0%	19.7%	4.8%	4.0%	5.2%	-7.2%	-18.8%	40.8%
SW	Diff	M+I	20	29.0%	33.6%	4.8%	1.3%	4.0%	-5.9%	-21.5%	40.4%

Table 14-21: Comparison of 2011 and 2013 M+I Inventories for the Armitage Deposit

Dep.	Year	Cat	Cut-off (%Sat)	Vol. (Mm^3)	Ton. (Mt)	Density (t/m^3)	TiO_2 (%)	Sat (%)	Al_2O_3 (%)	P_2O_5 (%)	S (%)
AE	2013	M+I	2	91.04	312.86	3.44	6.09	11.90	14.65	0.029	0.28
AE	2013	M+I	4	82.62	285.96	3.46	6.29	12.71	14.81	0.029	0.27
AE	2013	M+I	6	74.73	260.39	3.48	6.48	13.47	14.87	0.028	0.27
AE	2013	M+I	8	65.65	230.56	3.51	6.73	14.31	14.67	0.028	0.27
AE	2013	M+I	10	55.06	195.09	3.54	7.00	15.27	14.37	0.028	0.27
AE	2013	M+I	12	43.56	155.77	3.58	7.29	16.35	14.04	0.028	0.27
AE	2013	M+I	14	31.23	112.90	3.62	7.60	17.62	13.69	0.028	0.26
AE	2013	M+I	16	20.43	74.69	3.66	7.93	18.96	13.36	0.027	0.26

Dep.	Year	Cat	Cut-off (%Sat)	Vol. (Mm^3)	Ton. (Mt)	Density (t/m^3)	TiO_2 (%)	Sat (%)	Al_2O_3 (%)	P_2O_5 (%)	S (%)
AE	2013	M+I	18	11.67	43.13	3.70	8.27	20.45	12.99	0.027	0.26
AE	2013	M+I	20	5.77	21.58	3.74	8.56	21.98	12.62	0.027	0.25
AE	2011	Indicated	2	87.40	301.22	3.45	6.13	12.41	14.03	0.036	0.29
AE	2011	Indicated	4	85.39	294.77	3.45	6.17	12.61	14.06	0.037	0.28
AE	2011	Indicated	6	80.34	278.35	3.46	6.25	13.06	14.10	0.037	0.28
AE	2011	Indicated	8	72.03	250.88	3.48	6.40	13.72	14.03	0.037	0.27
AE	2011	Indicated	10	60.92	213.57	3.51	6.63	14.54	13.86	0.037	0.27
AE	2011	Indicated	12	47.35	167.24	3.53	6.91	15.50	13.63	0.037	0.27
AE	2011	Indicated	14	31.62	112.75	3.57	7.33	16.72	13.26	0.037	0.29
AE	2011	Indicated	16	17.70	63.76	3.60	7.77	18.08	12.86	0.037	0.30
AE	2011	Indicated	18	7.06	25.76	3.65	8.14	19.72	12.52	0.036	0.29
AE	2011	Indicated	20	2.24	8.27	3.70	8.38	21.68	12.22	0.036	0.25
AE	Diff	M+I	2	4.1%	3.8%	-0.4%	-0.7%	-4.2%	4.3%	-21.5%	-3.4%
AE	Diff	M+I	4	-3.3%	-3.0%	0.3%	1.9%	0.8%	5.2%	-25.0%	-3.0%
AE	Diff	M+I	6	-7.2%	-6.7%	0.7%	3.7%	3.1%	5.3%	-27.7%	-4.3%
AE	Diff	M+I	8	-9.3%	-8.4%	0.9%	5.1%	4.2%	4.4%	-27.7%	0.1%
AE	Diff	M+I	10	-10.1%	-9.0%	0.9%	5.4%	4.9%	3.6%	-27.7%	0.1%
AE	Diff	M+I	12	-8.3%	-7.1%	1.3%	5.4%	5.3%	3.0%	-27.7%	-0.5%
AE	Diff	M+I	14	-1.2%	0.1%	1.3%	3.6%	5.2%	3.2%	-27.0%	-9.1%
AE	Diff	M+I	16	14.3%	15.8%	1.5%	2.0%	4.8%	3.8%	-29.8%	-13.6%
AE	Diff	M+I	18	49.2%	50.4%	1.2%	1.5%	3.6%	3.7%	-26.9%	-12.3%
AE	Diff	M+I	20	88.3%	89.1%	1.1%	2.2%	1.4%	3.2%	-30.2%	-0.5%



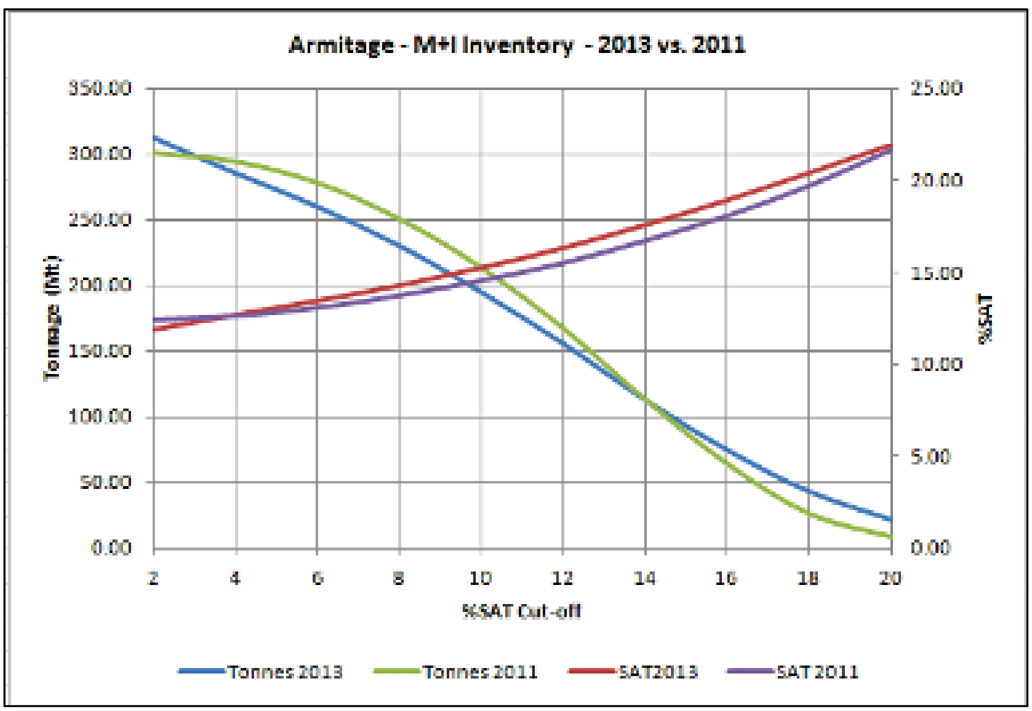


Figure 14-89: Comparison of 2011 and 2013 M+I inventories

14.11. Resource Estimates

Following CIM guidelines, resources are made part of the mineral inventory, with "a reasonable prospect of economic extraction". In practice, a final pit shell is optimized with reasonable technical and economic conditions, and resources are made of all blocks within that shell and above an economic cut-off corresponding to these conditions.

Optimized pit shells have been produced by BBA using technical and economic conditions defined in a previous iteration (2014). The shells for resources use Inferred resources, whereas those for reserves do not. They are shown in Figure 14-90. The cut-off used in the resource statement is 10% Satmagan, i.e. the same as the reserve cut-off and represents the mill COG. The mill COG was calculated using the parameters found in Table 15-2. A 10% Satmagan grade has been determined to meet metallurgical constraints. The statement of resources defined in those conditions is found in Table 14-22. The effective date of this Mineral Resource Statement prepared for the BlackRock Project is August 26, 2022.

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. Mineral resource estimates may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, and other relevant issues.

Table 14-22: Estimated resources of the BRM Deposits

Dep.	Cat.	Vol. (Mm^3)	Ton. (Mt)	Dens. (t/m^3)	%Sat	$\%TiO_2$	$\%V_2O_5$	$\%Fe_2O_3$	$\%Al_2O_3$	$\%P_2O_5$	%S
SW	Meas	40.0	145.0	3.63	17.9	7.9	0.48	41.2	13.0	0.033	0.17
SW	Ind	10.3	37.4	3.63	17.9	7.9	0.47	41.1	12.9	0.032	0.18
SW	M+I	50.2	182.4	3.63	17.9	7.9	0.48	41.1	13.0	0.032	0.17
SW	Inf	10.3	45.2	4.39	18.2	8.1	0.48	41.5	12.8	0.032	0.17
AE	Meas	40.1	142.2	3.55	15.4	7.1	0.41	36.9	14.3	0.028	0.28
AE	Ind	8.7	30.8	3.54	14.8	7.2	0.40	36.4	14.2	0.026	0.32
AE	M+I	48.8	173.1	3.55	15.3	7.1	0.41	36.8	14.3	0.028	0.29
AE	Inf	7.9	28.1	3.55	14.7	7.5	0.39	36.9	14.0	0.026	0.37
All	Meas	80.0	287.2	3.59	16.7	7.5	0.45	39.0	13.7	0.030	0.22
All	Ind	19.0	68.3	3.59	16.5	7.6	0.44	39.0	13.5	0.029	0.24
All	M+I	99.0	355.5	3.59	16.7	7.5	0.44	39.0	13.6	0.030	0.23
All	Inf	18.2	73.3	4.02	16.8	7.9	0.44	39.7	13.3	0.029	0.25

Resources are defined at a minimum cut-off of 10% Satmagan. Due to the necessary rounding of estimates, the rounded totals may slightly differ from the sum of rounded individual estimates.

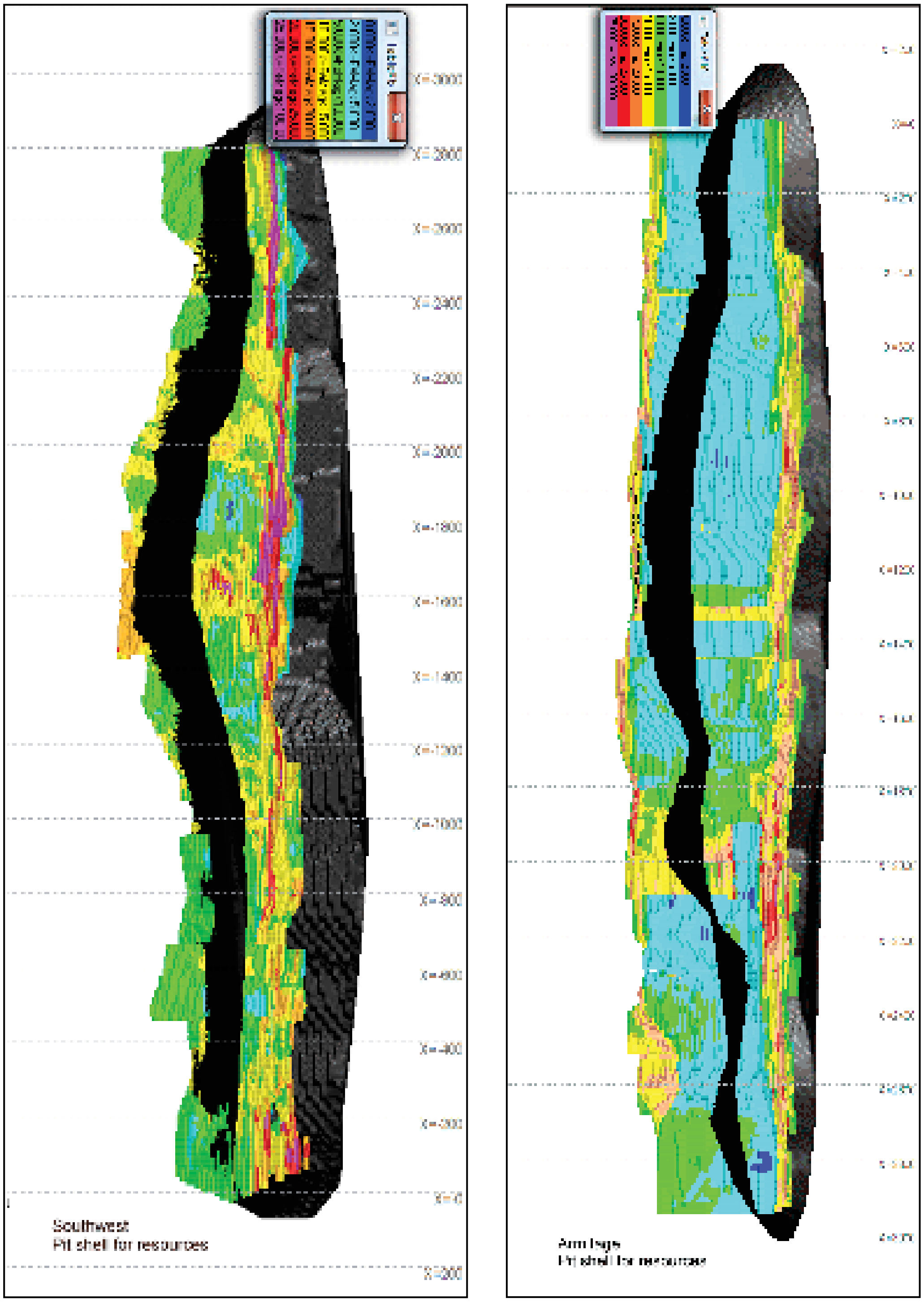


Figure 14-90: Pit shells for resources

15. Mineral Reserve Estimate

15.1 Introduction

Open pit optimization and detailed engineered pit design were carried out to convert Mineral Resources into Mineral Reserves for the Southwest Deposit. For the Armitage Deposit, the level of detail of the mine plan and quantity of metallurgical testworks and geotechnical analysis are not sufficient to convert the In-pit Resources into Mineral Reserves.

The author is of the opinion that no other known risks including legal, political or environmental, would materially affect potential development of the Mineral Reserves, except for those already discussed in this report.

15.2 CIM Definition of Mineral Reserves for NI 43-101 Technical Report

As defined by the Canadian Institute of Mining, Metallurgy and Petroleum within the CIM Definition Standards on Mineral Resources and Mineral Reserves (adopted by CIM Council in May 2014), the definition of a Mineral Reserve is as follows:

> *"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."*

For the purpose of this Feasibility Study, the pit optimization has used only mineralized blocks classified in the Measured and Indicated categories to drive the computer-assisted pit optimizer algorithm and to be classified as Proven and Probable Mineral Reserves, respectively.

15.3 Resource Block Model

SGS Geostat (SGS) was mandated by BlackRock for the preparation of the resource block model for the Project (*RESSW24022014.xlsx*). The block size used in the model is x=10 m x y=5 m x z=7 m. The model was transferred to BBA as a Comma Separated Value file (CSV) for input into the MineSight software suite.

SGS provided the following variables in the resource block model:

- IX, IY, IZ representing the (I, j, k) indices of the block;
- Mineralized block centres in rotated local coordinates (x, y, z);
- Mineralized block centres in UTM coordinates (x, y, z);
- ZONE representing the four mineralized units (BCS, MC, ULS, TITAN);
- FRAC is the percentage of material in a given block under the bedrock surface;
- DMAIN represents the structural domains (7 domains for Southwest);
- CAT is the resource category (1=Measured, 2=Indicated, 3=Inferred);
- Grades: %Fe_2O_3, %TiO_2, %V2O5, %SAT (i.e., %Satmagan), %Al_2O_3, %P_2O_5, %S.

More details on Mineral Resources and the preparation of the block model are presented in Chapter 14 of this report.

15.4 Block Model Coordinate System

Block coordinates were provided by SGS in two types of coordinate systems as follows:

- A rotated mine local coordinate system, based on the geological cross-section locations perpendicular to the long axis of the ore body;
- A Universal Transverse Mercator (UTM) coordinate system.

The transformation required to move from the local coordinate system to the UTM coordinate system is given below:

- Southwest Local to UTM: (0E, 0N) = (568,508E, 5, 516,648N), 130° clockwise rotation.

The model coordinate system used by BBA in MineSight is the local (rotated) system.

15.5 Model Surfaces

In addition to the block model file, two surface files were provided to BBA in the form of gridded (x, y, z) ASCII files in the UTM coordinate system:

- Topography surface; and
- Overburden surface (interface bedrock/overburden).

The provided overburden surface was created based on drillhole collars and the average depth was calculated to be roughly 2.5 m, which did not correspond to observed overburden thickness in the field on two separate site visits by BBA in the fall of 2018. At BBA's request, Blackrock hired a local surveyor to measure the depth of overburden in the area of the starter pit. The surveyor measured the overburden at 39 drillhole locations and an additional 65 random points in the vicinity of the starter pit. Within the area of the starter pit the average overburden thickness was calculated to be 0.52 m (0.30 m of organics and 0.22 m of till). In the absence of more accurate measurement for the rest of the ultimate pit, the average thickness of 0.52 m was used to calculate the total overburden removal required. The topography surface was interpreted as top of overburden, and the overburden amount based on the 0.52 m average thickness. This approach allows conservative waste vs. overburden quantities.

15.6 Modifying Factors

For the conversion of Mineral Resources to Mineral Reserves, it is necessary to apply a variety of modifying factors.

15.6.1 Model Density

The densities for mineralized blocks were provided by SGS in the resource block model. The density for material within the mineralized units changes as a function of variations in the modal mineralogy. The regression relationship between the calculated modal mineralogy and density was derived in comparison with 1,043 pycnometer tests for 3-metre composites that conformed to the assay intervals in the four stratigraphic mineralized units, as well as the interstitial waste material. These values ranged from 3.00 t/m^3 to 4.40 t/m^3 for the Southwest Pit (see Chapter 11.3 for details regarding the calculation of density, and Chapter 14.6 for details on how the regression equation was applied to the individual blocks in the deposit).

The average densities for waste blocks were derived through a separate sampling program focused on the layed-back areas of the pit, away from the assayed mineralization. These were grouped separated into three general categories:

- Hanging Wall;
- Footwall;
- Internal (i.e., waste between mineralized units).

The resulting average waste rock densities are summarized in Table 15-1. A full description of the waste rock density program can be found in Section 11.3 of this report.

Table 15-1: Waste specific gravity for Southwest Pit

Category	Southwest
Hanging Wall (t/m^3)	3.02
Footwall (t/m^3)	3.10
Internal Waste (t/m^3)	3.12
Overburden (t/m^3)	2.00

Blackrock carried out specific gravity (SG) immersion testing using Archimedes principle in two separate programs (2010 and 2013) to complement the pycnometer density measurements. Both of these programs found a positive bias in the pycnometer density measurements (heavier pycnometer values) over the water immersion measurements. The pycnometer information was accepted as more representative and subsequently used as the basis of the regression equations for density determinations in the block model. However, the pycnometer method for density measurement is prone to potential bias due to the loss of natural pore space and voids in the rock during the pulverization process.

It is recommended to further assess the density of the mineralized material and host rock at the detailed engineering phase of the project.

15.6.2 Metallurgical Recoveries

The metallurgical weight recoveries were determined from the results of metallurgical testworks performed on samples from the Southwest Deposit to produce a magnetite concentrate of 62% Fe.

The grade of V_2O_5 in the magnetite concentrate was determined based on the V_2O_5 and Satmagan feed grade. A vanadium correlation equation was developed for each of the four lithological zones using data from 294 Davis tube test results.

The magnetite weight recovery and grade of V_2O_5 in the magnetite concentrate equations have been updated for this Feasibility Study and vary according to ore zones. More information on the calculation of magnetite as well as grades of V_2O_5 in the magnetite concentrate can be found in Chapter 13.

15.6.3 Mill Cut-off Grade Calculation

The mill cut-off grade (COG) is used to classify the material inside the pit limits as rock or waste. For material located inside the pit, the mill cut-off is the grade required to cover the costs for processing, G&A, other costs related to concentrate production and transport as well as ferrovanadium (FeV) and pig iron (HPPI) second transformation production costs. Since the concentrator produces a magnetite concentrate, the mill COG is quantified in % Satmagan grade. Table 15-2 illustrates the parameters used to determine the Satmagan mill COG.

Table 15-2: COG Calculation parameters

Parameters	Unit	Value
Fe Processing and Other Costs	C$/t milled	11.03
Magnetite Concentrate Transport and Rail Car Maintenance	C$/t conc.	19.15
Concentrate Handling at Port	C$/t conc.	2.50
HPPI Selling Price	C$/t Product	1,022
HPPI Selling Cost	C$/t Product	1,022
FeV Selling Price	C$/t Product	49.621
Fev Production Cost	C$/t Product	22,711
Selected Mill COG	**%SAT**	**10%**

For the Study, a mill cut-off grade of 10% Satmagan has been selected. This COG is higher than the calculated mill COG. Indeed, a 10% Satmagan grade has been determined to meet metallurgical constraints.

To finalize the COG selection, a grade/tonnage distribution curve has been developed for the Southwest Deposit and V_2O_5 grade in concentrate and tonnage of concentrate have been analyzed at varying COG.

The COG is applied after dilution.

15.6.4 Mining Dilution and Mining Ore Loss

Mining dilution and mining ore loss were calculated on a block-by-block basis using a numerical approach in MineSight mining software. For each mineralized block, waste blocks (or mineralized blocks below cut-off grade) that share a contact edge with the mineralized block in one or more of the four cardinal directions on the same mining bench are used in the dilution calculations. Based on the choice of loading equipment (9 m^3 capacity), a dilution of 1.0 m has been applied to each contact edge taking into account the bench height. It is assumed that 50% of the time, under-digging will result in ore loss and 50% of the time, over-digging will result in added waste. The average mining loss calculated is 3.4%. The average added waste dilution calculated is 2.8%. The amount of Measured and Indicated Resources that fall below cut-off due to the added waste dilution is 1.8%. There is thus a net loss of 2.3% of reserves due to dilution.

15.7 Open Pit Optimization

In order to develop the detailed engineered pit design and mine plan for the Southwest Deposit, optimized pit shells were first prepared.

In accordance with the NI 43-101 guidelines and the Canadian Institute of Mine, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves, only blocks classified as either Measured or Indicated are allowed to drive the pit optimizer for a Feasibility Study.

15.7.1 Methodology

Pit optimizations were carried out using the MineSight Economic Planner Module and its Lerchs-Grossman 3D algorithm. The pit optimization algorithm is used to produce pit shells that are physical representations of the optimal pit to be mined, assuming a given set of parameters and 3D block model. Using a variety of input parameters such as mining costs, processing costs, weight recovery values and pit slopes, the algorithm outputs the pit shell that maximizes the undiscounted value. These shells are devoid of geotechnical and operational features such as ramps and proper benching arrangements and are to be used as a basis and guide for the design of an engineered open pit. No capital expenses are considered by the pit optimization tool.

A series of pit optimizations are produced using a range of net revenue factors (reduction factors on selling price minus costs) from 15% to 120% in order to produce the industry standard pit-by-pit graph. The revenue factor is used to measure the sensitivity of the pit optimizations to changes in mineral selling prices, as well as to evaluate the effect of the pit size and stripping ratios on the project NPV.

The optimization pit shells will produce a series of nested pit shells that will prioritize the mining of the most economic material and progressively grow in size, while less profitable material is mined as the revenue factor increases.

The results of the pit optimizations are subsequently compared on the basis of the calculated NPV, undiscounted value and tonnes of ore and waste material. From these results, a final pit optimization shell that meets project requirements is selected. Examples of the important project requirements include the NPV, overall pit stripping ratio, mine life and average grade.

15.7.2 Pit Optimization Parameters for the Southwest Deposit

For the Southwest Deposit a pit optimization series have been generated based on selling HPPI and FeV, using updated Fe weight recovery equation per zones and updated V_2O_5 recovery equations per zones.

The main pit optimization parameters for the Southwest Deposit used in the pit optimization runs are listed in Table 15-3. The selling prices and costs used for the pit optimization process were based on the best available information at the time of the study, including escalated costs from the 2017 Study, costs from similar existing mining operations, BBA's experience and geotechnical information from three main sources: LVM report (2013), Mine Concept report (2013) and recent discussion with third party consultant on the Southwest geotechnical recommendations.

Table 15-3: Pit Optimization parameters for the Southwest Deposit

Parameter	Base Value	Unit	Source
PIT SLOPE			
Overall Pit Slope Angle	27° - 54°	degree	LVM 2013 - 4 to - 5° adjustment.
Mining Costs			
Ore/Waste Base Mining Cost	3.71	$/t mined	BBA FS 2017 with incremental adjustments of +4%/years to final cost
Ore/Waste Incremental Mining Cost	0.04	$/t mined	
Overburden Mining Cost	2.51	$/t mined	BBA FS 2017 - Final Cost
Processing & G&A Costs			
G&A Cost (Fe Concentrate)	2.12	$/t milled	BBA 2021
Processing Cost (Fe Concentrate)	7.11	$/t milled	BBA 2021

Parameter	Base Value	Unit	Source
Other Cost (coarse tailing, env., leasing)	1.8	$/t milled	BBA FS 2017
Total Operating Cost	11.03	$/t milled	
Total Operating Cost	**41.61**	**$/t conc.**	**Assuming 26.5% weight recovery**
Net Value & Payment			
HPPI Selling Price	670	$/t Product	BlackRock (Aug. 2021)
HPPI Production Cost	360	$/t Product	BlackRock (Aug. 2021)
FeV Selling Price	54,341	$/t Product	BlackRock (Aug. 2021)
FeV Production Cost	22,711	$/t Product	BlackRock (Aug. 2021)
Fe Concentrate Transport	19.15	$/t conc.	BlackRock (2021)
Fe Concentrate Port	2.50	$/t conc.	BlackRock (2021)

Notes:
1. All costs are in Canadian dollars.

The geotechnical consultant, LVM, provided recommendations for the Bench Face Angle (BFA), Inter Ramp Angle (IRA) and catch bench. BBA adjusted the values to include the assumed ramp placement.

The mineralogy of the mineralized material was reviewed, and the content of secondary elements deemed acceptable. Thus, no penalties were applied on the selling price of the final concentrate. Furthermore, no contaminant limits were considered in the optimization and mine planning processes.

15.7.3 Pit Optimization Results for Southwest

For the Southwest Pit, the final pit optimization retained for the design of the ultimate pit was the one calculated at a net revenue factor of 0.29. This selection was based on:

- This pit shell presented one of the highest Net Present Value (NPV);
- It contained sufficient ore tonnage for a mine life of over 40 years (at a concentrate production rate of 856 ktpy).

Table 15-4 presents the pit optimization series pit shell material inventory and potential operational costs and revenues.

Table 15-4: Pit Optimization results for Southwest

PIT	Net Revenue Factor	Mineralized Material			Fe conc. (Mt)	HPPI (Mt)	FeV80 (Mt)	Waste Rock (Mt)	Total Material (Mt)	Strip Ratio	Potential Operational Costs and Revenues							
		Tonnes (Mt)	SAT Grade (%)	Fe WR (%)							Mining Cost (M$)	Processing Cost (M$)	Revenue (M$)	Mine Life (y)	Un-discounted Value (M$)	NPV* (exc. Capex) (M$)	Incremental NPV (M$)	% of Max NPV (%)
PIT 100	0.15	12.1	23.0%	32.0%	3.9	2.4	0.03	3.6	15.7	0.29	53	134	1,565	4.5	1,378	1,119	0.0	35%
PIT 125	0.16	17.9	22.3%	31.0%	5.5	3.5	0.04	6.4	24.3	0.36	83	198	2,222	6.5	1,941	1,470	60.6	46%
PIT 126	0.17	22.7	21.8%	30.3%	6.9	4.3	0.05	10.0	32.7	0.44	112	250	2,747	8.0	2,384	1,710	50.0	54%
PIT 127	0.18	30.1	21.2%	29.4%	8.8	5.5	0.06	16.0	46.1	0.53	159	332	3,516	10.3	3,025	2,008	40.3	63%
PIT 128	0.19	36.7	20.8%	28.9%	10.6	6.6	0.07	23.1	59.8	0.63	207	405	4,200	12.4	3,588	2,225	33.0	70%
PIT 101	0.20	44.1	20.5%	28.3%	12.5	7.8	0.09	32.3	76.4	0.73	266	486	4,945	14.6	4,192	2,422	26.5	76%
PIT 130	0.21	52.9	20.1%	27.8%	14.7	9.2	0.10	45.3	98.2	0.86	345	583	5,816	17.2	4,888	2,608	21.3	82%
PIT 131	0.22	61.1	19.8%	27.4%	16.8	10.5	0.12	58.1	119.2	0.95	422	674	6,615	19.6	5,519	2,743	16.3	86%
PIT 132	0.23	73.6	19.7%	27.2%	20.0	12.5	0.14	84.2	157.8	1.15	565	811	7,880	23.3	6,504	2,905	13.1	91%
PIT 133	0.24	87.3	19.5%	26.9%	23.5	14.7	0.16	115.1	202.4	1.32	733	963	9,259	27.5	7,563	3,027	8.8	95%
PIT 102	0.25	97.5	19.4%	26.8%	26.1	16.4	0.18	141.5	239.0	1.45	872	1,075	10,291	30.5	8,345	3,090	6.2	97%
PIT 135	0.26	111.9	19.4%	26.8%	29.9	18.8	0.21	183.0	294.9	1.64	1,086	1,234	11,762	35.0	9,442	3,146	3.9	99%
PIT 136	0.27	119.6	19.3%	26.7%	31.9	20.0	0.22	206.5	326.1	1.73	1,208	1,319	12,542	37.3	10,015	3,166	2.5	99%
PIT 137	0.28	127.4	19.2%	26.6%	33.8	21.2	0.24	229.3	356.7	1.80	1,329	1,405	13,292	39.5	10,558	3,179	1.7	100%
PIT 138	0.29	135.3	19.2%	26.5%	35.8	22.4	0.25	255.4	390.7	1.89	1,462	1,492	14,060	41.8	11,106	3,185	0.8	100%
PIT 103	0.30	141.5	19.1%	26.4%	37.3	23.4	0.26	277.1	418.6	1.96	1,573	1,561	14,661	43.6	11,528	3,188	0.3	100%
PIT 104	0.35	159.5	18.9%	26.1%	41.5	26.0	0.29	342.4	501.9	2.15	1,908	1,758	16,309	48.5	12,642	3,179	-0.5	100%
PIT 105	0.40	168.4	18.8%	25.9%	43.6	27.3	0.30	384.7	553.1	2.28	2,115	1,857	17,113	50.9	13,141	3,164	-1.6	99%
PIT 106	0.45	174.8	18.7%	25.7%	44.9	28.2	0.31	418.6	593.4	2.40	2,283	1,927	17,664	52.5	13,454	3,148	-2.5	99%

PIT	Net Revenue Factor	Mineralized Material			Fe conc. (Mt)	HPPI (Mt)	FeV80 (Mt)	Waste Rock (Mt)	Total Material (Mt)	Strip Ratio	Potential Operational Costs and Revenues							
		Tonnes (Mt)	SAT Grade (%)	Fe WR (%)							Mining Cost (M$)	Processing Cost (M$)	Revenue (M$)	Mine Life (y)	Un-discounted Value (M$)	NPV* (exc. Capex) (M$)	Incremental NPV (M$)	% of Max NPV (%)
PIT 107	0.50	179.2	18.6%	25.6%	45.8	28.7	0.32	445.5	624.8	2.49	2,413	1,976	18,033	53.5	13,643	3,133	-3.3	98%
PIT 108	0.55	182.1	18.5%	25.5%	46.4	29.1	0.32	465.5	647.6	2.56	2,509	2,008	18,270	54.2	13,753	3,121	-4.3	98%
PIT 109	0.60	183.8	18.5%	25.5%	46.8	29.3	0.33	481.6	665.4	2.62	2,582	2,027	18,426	54.7	13,817	3,111	-5.6	98%
PIT 110	0.65	185.9	18.4%	25.4%	47.2	29.6	0.33	498.3	684.2	2.68	2,661	2,050	18,585	55.2	13,874	3,100	-5.6	97%
PIT 111	0.70	187.6	18.4%	25.4%	47.6	29.8	0.33	516.1	703.7	2.75	2,742	2,069	18,726	55.6	13,915	3,087	-7.2	97%
PIT 112	0.80	189.2	18.4%	25.3%	47.9	30.0	0.33	534.4	723.7	2.82	2,826	2,087	18,861	56.0	13,948	3,074	-8.0	96%
PIT 113	0.90	190.8	18.3%	25.3%	48.2	30.2	0.34	555.2	746.0	2.91	2,919	2,104	18,983	56.3	13,960	3,058	-10.0	96%
PIT 114	1.00	191.5	18.3%	25.2%	48.3	30.3	0.34	564.5	756.0	2.95	2,962	2,112	19,037	56.5	13,962	3,051	-10.0	96%
PIT 115	1.10	192.1	18.3%	25.2%	48.4	30.4	0.34	573.2	765.3	2.98	3,000	2,118	19,077	56.6	13,959	3,044	-12.3	96%
PIT 116	1.20	192.9	18.3%	25.2%	48.6	30.4	0.34	588.8	781.6	3.05	3,069	2,127	19,140	56.8	13,945	3,032	-16.0	95%

Notes :

1. Based on the PFS 2014 regularized block model prepared by SGS Geostat ''RESSW24022014.xlsx'';
2. Measured and indicated resources included as ''ore'' for the LG optimization routine. Inferred treated as non-economic material;
3. Resources are based on a cut-off grade of 10% Satmagan;
4. Mining Dilution and ore losses are included;
5. NPV values are calculated based on the base case HPPI (670$/tonnes) and FeV (54,341$/tonnes);
6. NPV estimated assuming an average strip ratio and constant annual production over estimated LOM. NPV values exclude all CAPEX
7. FeV80 tonne based on V_2O_5 in concentrate (equation by zone), 74% recovery for second transformation.
8. Highest NPV pit shown in **BLUE**, RF = 1.0 pit shown in **BEIGE** and the pit selected is in **RED**.
9. NPV* excludes project Capex. NPV* should be used to compare different pit shells and not as an indication of the project economics.

15.8 Southwest Pit Detailed Mine Design

15.8.1 Southwest Open Pit Geotechnical

The geotechnical requirements for the Feasibility Pit Design were prepared by LVM (2013) and recommendations are provided in a report entitled, "Pit Slope Design Report for Southwest Pit". Recommendations were provided for the inter-ramp angle (IRA), bench face angle (BFA) and catch bench width. The angles vary depending on the sector identified in the LVM report. A second geotechnical report was produced by Mine Concept in 2013 with slightly different recommendations. There was also discussion with a third-party geotechnical consultant in 2021 that recommended more conservative overall slope angles (OSA). Consequently, BBA decided to modify the OSA recommended by LVM by -5 degrees for Zone 4, no modification for Zone 5 and Zone 8, and by -4 degrees for the other zones.

A 28 m bench arrangement was retained and the catch bench width was calculated based on these values. The summary of the proposed pit slopes by sector is found in Table 15-5 while sectors are identified in Figure 15-1. These sectors are identical in size and location to those used for the pit optimization.

Table 15-5: Pit design geotechnical parameters

Pit Slope Sector	IRA	BFA	Catch Bench Width (m)
1	54	70	10.0
2	54	70	10.0
3	54°	70	10.0
4	42°	53	10.0
5	54	70	10.0
6	54	70	10.0
7	54	70	10.0
8	54	70	10.0
9	54	70	10.0
Overburden	**-**	**26.6° (2H:1V)**	**-**

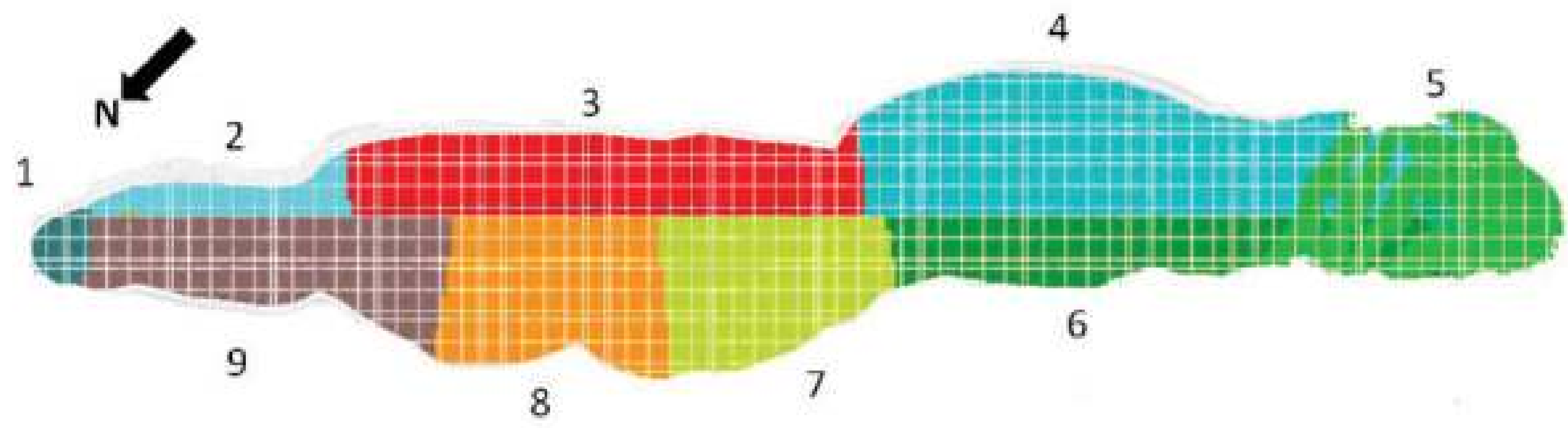


Figure 15-1: Geotechnical slope sectors

15.8.2 Southwest Pit Design Parameters

The detailed mine design was carried out using the selected pit shell as a guide. The proposed pit design includes the practical geometry required in a mine, including pit access and haulage ramps to all pit benches, pit slope designs, benching configurations, smoothed pit walls and catch benches. The major design parameters used are described in Table 15-6.

Table 15-6: Detailed engineered pit design parameters

Parameter	Value
Minimum Mining Width	40 m
Parameter in Fresh Rock	
Benching Arrangement [(1)]	4 x 7 m
Berm Width	10 m
Inter-Ramp Angle (IRA)	54°
Bench Face Angle (BFA)	70°
Ramp Width (1-lane)	18 m
Ramp Width (2-lane)	27 m
Ramp Grade	10°
Parameter in Overburden	
Benching Arrangement	-
Berm Width	10.5 m
Inter-Ramp Angle (IRA)	25°
Bench Face Angle (BFA)	30°
Ramp Width (1-lane)	18 m
Ramp Width (2-lane)	27 m
Ramp Grade	10°

The in-pit haulage roads are 27 m wide to accommodate the proposed 90-tonne haul trucks. This ramp will provide sufficient room for 2-way traffic. All in-pit ramps have been restricted to a maximum 10% gradient.

Two haulage ramps have been included in the ultimate pit design to provide haul access to either end of the waste pile as well as towards the crusher. A third ramp provides additional access to the upper 56 metres of the south end of the pit. Figure 15-2 presents an isometric view of the ultimate Southwest Pit.

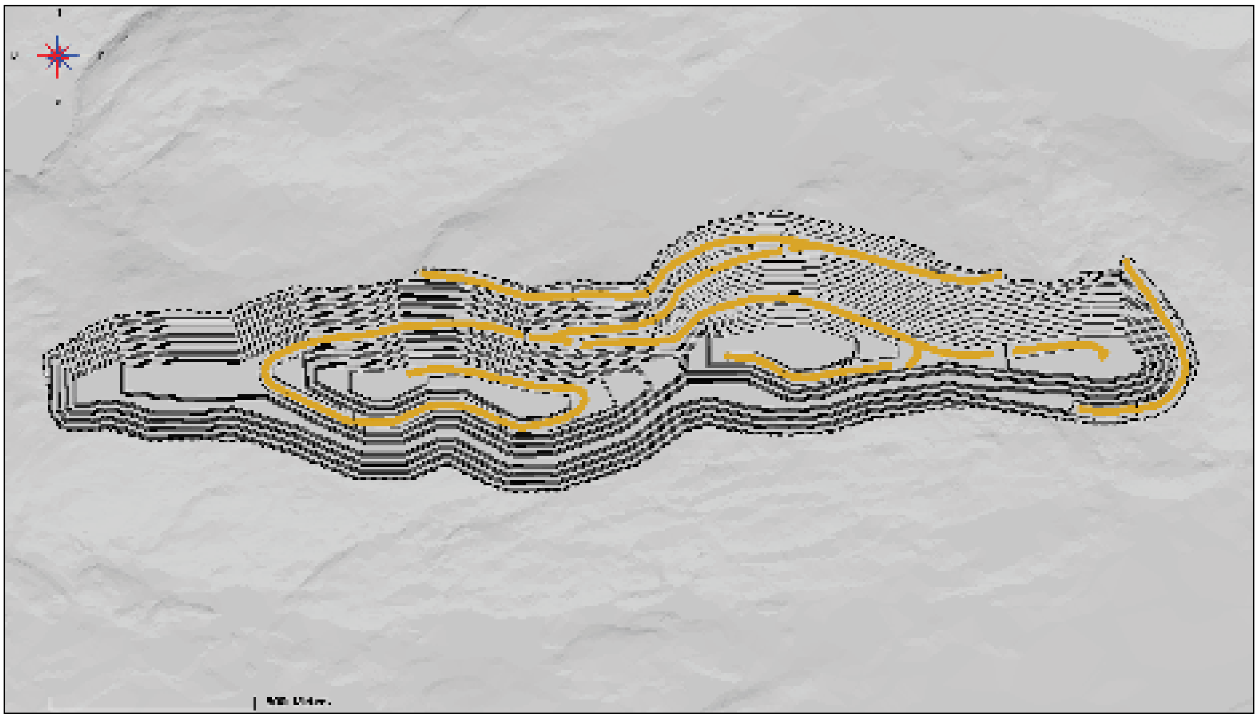


Figure 15-2: Ultimate Southwest Pit isometric view

The sketches from Figure 15-3 to Figure 15-5 present typical bench plans and cross-sections of the detailed pit versus the selected optimized pit (RF 0.29).

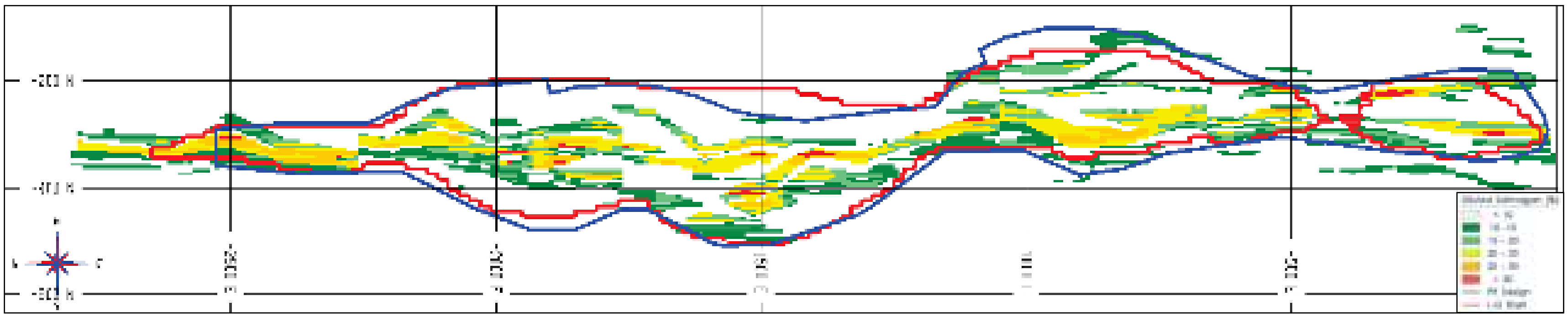


Figure 15-3: Selected pit optimization (blue lines) and final pit design (red lines) at elevation plan view 357

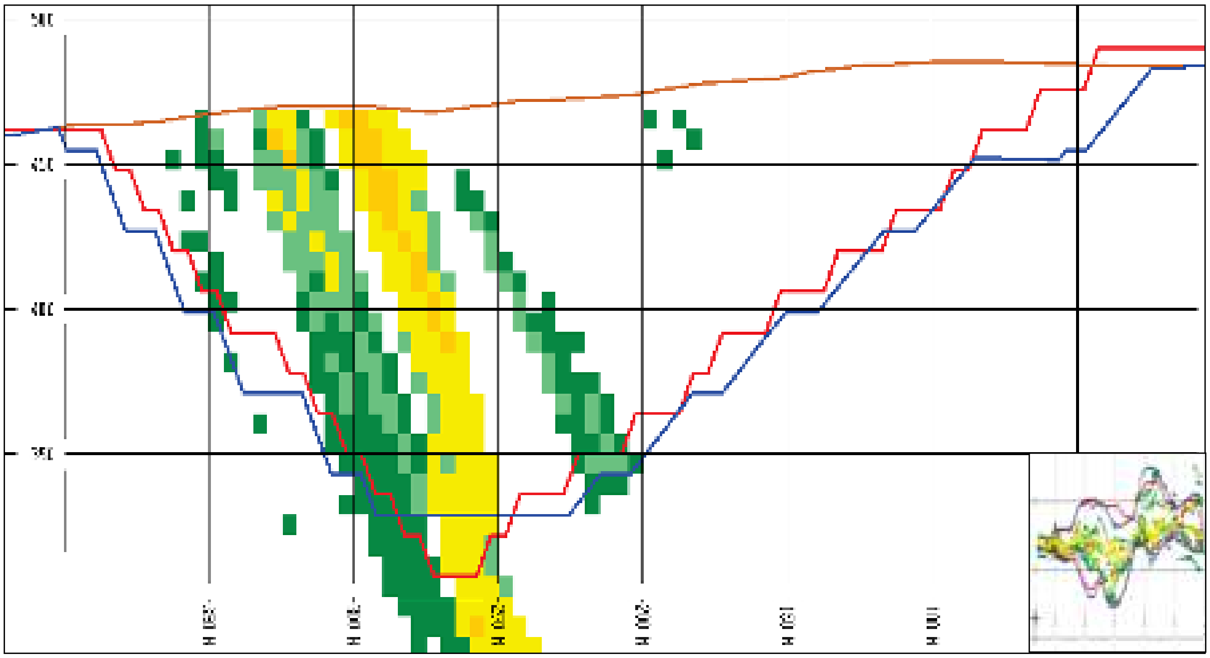


Figure 15-4: Selected pit optimization (blue lines) and final pit design (red lines) at section view -570 east

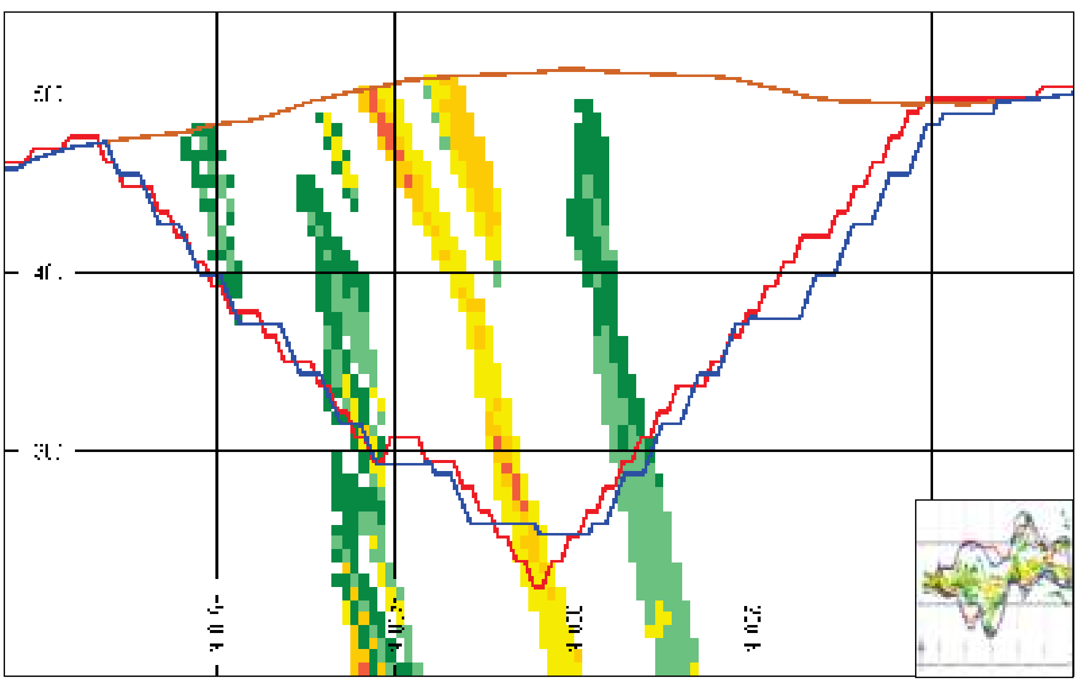


Figure 15-5: Selected pit optimization (blue lines) and final pit design (red lines) at section view -1680 east

15.9 Open Pit Mineral Reserves

The Southwest detailed mine planning is presented in Chapter 16.

In accordance with the NI 43-101 standards of mineral classification, the measured and indicated resources inside the final pit limits of Southwest Pit have been transferred into Proven and Probable reserves after the application of the modifying factors. The open pit Mineral Reserves for the Southwest Pit are shown in Table 15-7.

The total Mineral Reserves for the Southwest Pit amount to 127.8 Mt proven and probable at a grade of 18.8% Sat, 7.8% TiO_2 and 0.46% V_2O_5 based on a cut-off grade of 10% Satmagan. The Southwest Pit reserves are sufficient for approximately a 39-year mine life at an average magnetite concentrate production of 856 ktpa.

Table 15-7: Southwest Pit Mineral Reserves

Mineral Category	Tonne (kt)	SAT (%)	Fe_2O_3 (%)	TiO_2 (%)	V_2O_5 (%)	Expected V_2O_5 in concentrate (%)	Expected metallurgical weight recovery (%)
Proven	123,900	18.9	40.2	7.7	0.46	1.34	26.0
Probable	3,900	17.9	40.3	8.1	0.42	1.24	25.0
Total P&P Reserve	127,800	18.8	40.2	7.8	0.46	1.33	26.0

Notes:
1. Resources are defined at a minimum cut-off of 10% Saturation Magnetic Analysis ("Satmagan"). Due to the necessary rounding of estimates, the rounded totals may slightly differ from the sum of rounded individual estimates.
2. The Mineral Resource estimate was completed by Claude Bisaillon, P.Eng. (OIQ #116407) from DRA Americas formerly from SGS Geostat at the time of writing the present report, an independent Qualified Person as defined in NI 43-101.
3. The effective date of the Mineral Reserve estimate is October 30, 2022.
4. The Mineral Reserves 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 in May 2014.
5. Qualified Person: The Mineral Reserve statement was prepared by Isabelle Leblanc (OIQ #144395) of BBA, an "independent qualified person", as that term is defined by National Instrument 43-101.
6. Open pit Mineral Reserves have been estimated using a 0.29 net revenue factor apply on High Purity Pig Iron (HPPI) price of 670 CAD/t of product, a Ferrovanadium (FeV) price of 54,341CAD/t of product, a foreign exchange rate of CAD1.33 to USD1.00.
7. Open pit reserves have been estimated using a cut-off grade of 10% Satmagan.
8. The LOM strip ratio is 2.2.
9. Reserves are derived from the Satmagan Resources Statement (182.4Mt of resources in the Measured and Indicated categories at a cut-off grade of 10%) prepared by Claude Bisaillon (OIQ #116407) from DRA Americas formerly from SGS Geostat.
10. The reference point for the Mineral Reserves is the crusher feed.
11. Expected %V2O5 in concentrate and % metallurgical weight recovery are based on Davis Tube Analysis (DTA) metallurgical testwork. The formulas by mineralized units, are presented in Chapter 13.1.3.
12. BBA is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Reserves estimate, except for those already discussed in this report.

16. Open Pit Mining Methods

The Southwest Pit will be mined using a conventional open pit drill and blast, load and haul mining method using a drill, truck and shovel and loader mining fleet. To reduce power plant load requirements and also for ease of movement around the pit, a fleet of diesel-powered equipment has been selected. The current feasibility study is based on an "Owner-Operator" approach except for the pre-production period which will use a contractor for the drilling and extraction of the rock as well as construction of the infrastructure.

16.1. Mine Planning

The objective of the life of mine schedule is to provide a basis for the orderly development of the Southwest Pit while maximizing the return on investment and the use of the mineral reserves.

16.1.1. Mine Phases

To maximize the Net Present Value (NPV) of the Project, a series of six staged pits was developed (including the ultimate pit pushback) to delay unnecessary stripping. Engineered phase designs (pushbacks) were developed for all the six phases to help define the sequence of mining as well as to maximize the cash flow of the project. These designs were then used for mine planning. The following additional criteria were used in the phase selection process:

- Minimum offset of 60 m between phases;
- Ease of access to different mining areas; wide benches facilitating equipment movement between phases to maximize operating flexibility;
- SAT Grade distribution constraints (i.e., to allow flexible ore blending options);
- Mining and processing production rate constraints;
- Postpone mining of Phase 3 until after Year 12 due to a water stream that runs through this area.

Figure 16-1 shows all the phases for the mining areas.

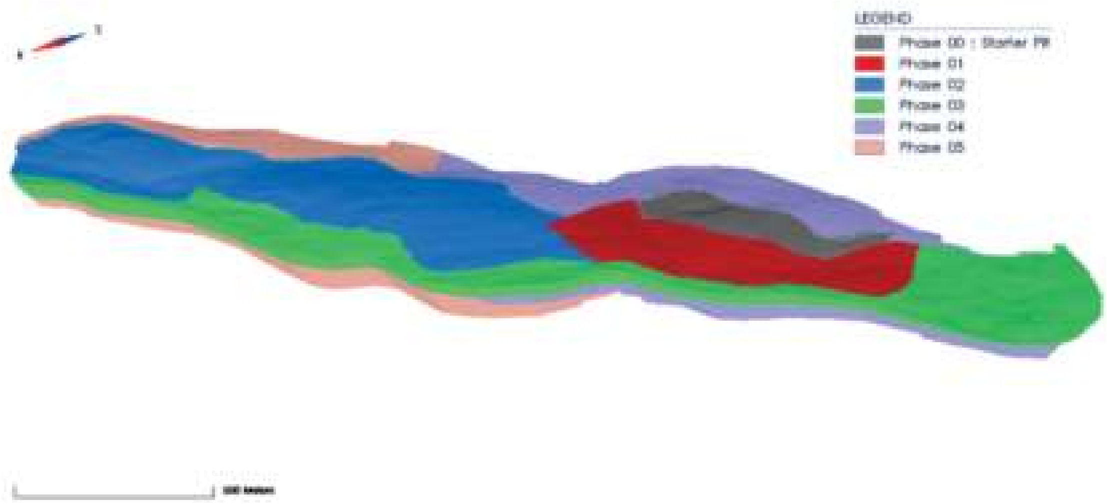


Figure 16-1: Southwest mine phases

The various mining phases are presented as follows:

- Phase 00 is a pre-production phase to mine waste required for construction of tailings dykes, roads, and civil infrastructure. This phase is contained within the Phase 01 limits to reduce the amount of waste mined in that phase. The phase mines 4.5 Mt of waste and as little ore as possible (0.25 Mt), which is stockpiled and later reclaimed for processing. This ore is low grade ore with an average SAT of 14.8%.

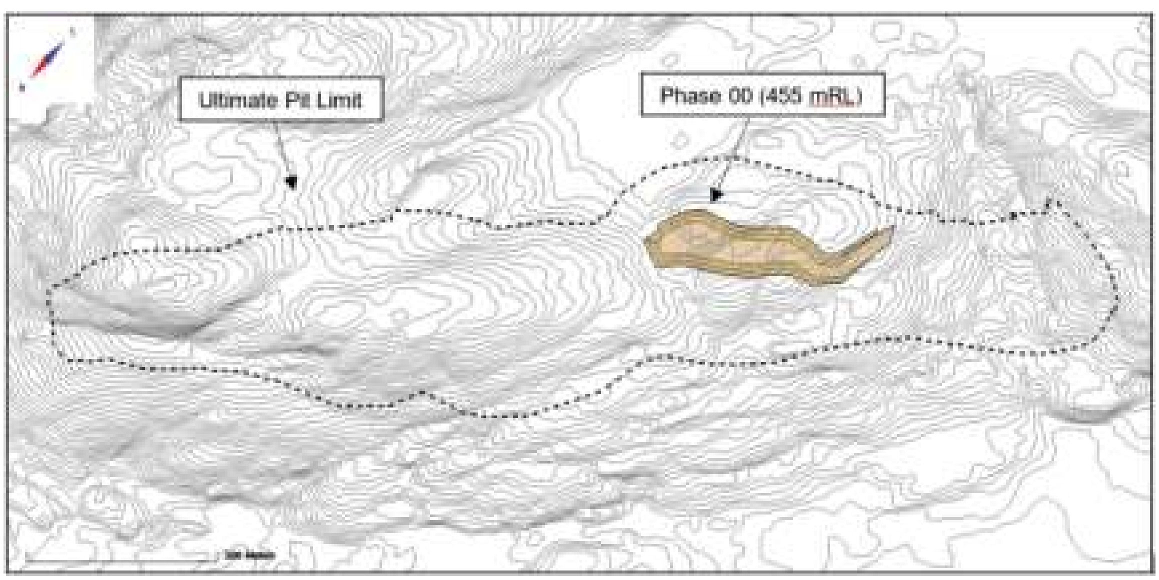


Figure 16-2: Southwest pre-production phase (Phase 00) design

- Phase 01 is located near the concentrator while avoiding the area at the south end of the pit where an active stream is located. This minimizes the haul to the primary crusher while also providing the most amount of time for planning water diversion. The phase contains 13.1 Mt (approximately 10%) of the proven and probable reserves in the Southwest Pit, with a low stripping ratio of 1.38. The Phase 01 design is illustrated in Figure 16-3 below.

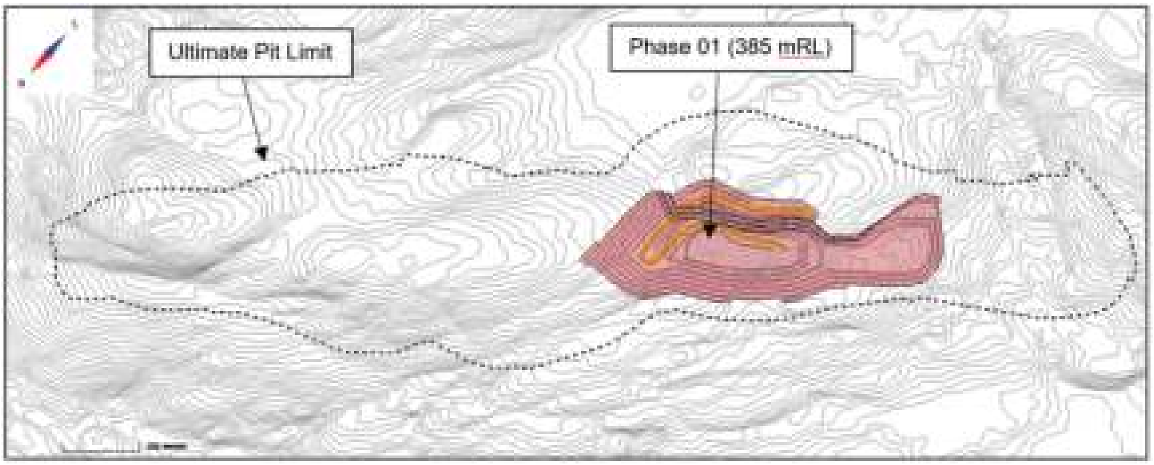


Figure 16-3: Southwest Phase 01 design

- Phase 02 is developed as an extension of Phase 01 towards the Northeast. A dual lane haulage ramp is used to access the bottom benches. The phase contains 23.7 Mt (roughly 18%) of proven and probable reserves. The average ore grade is 20.0% SAT, which is slightly higher than in Phase 01. The strip ratio is lower at 1.35. Phase 02 design is illustrated in Figure 16-4 below.

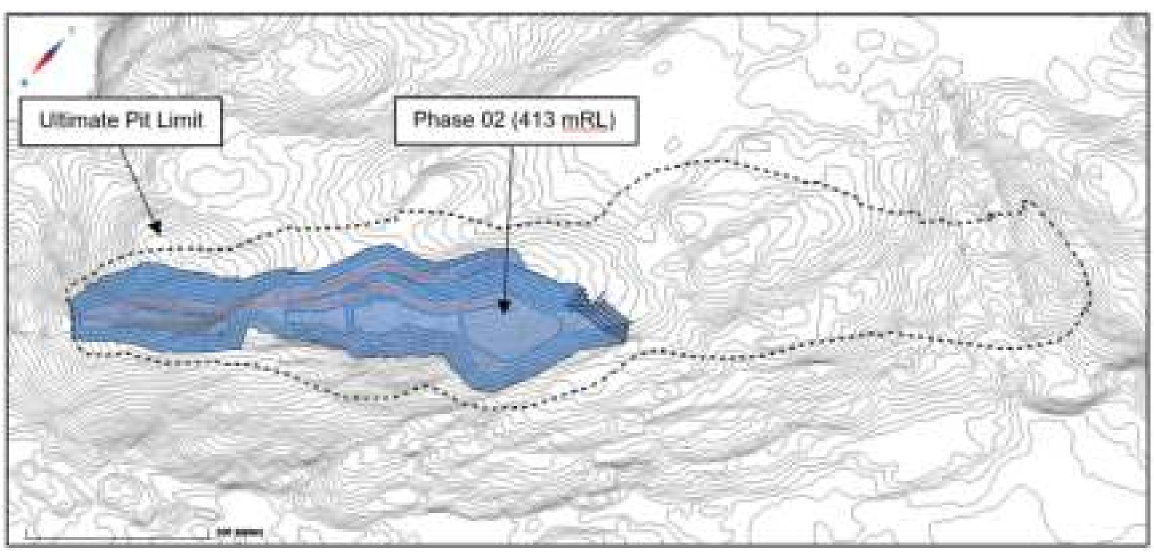


Figure 16-4: Southwest Phase 02 design

- Phase 03 extends on the west as well as the southwestern limits of the ultimate pit, mining through the area where the water stream is located. An additional ramp system is created to mine this phase to take down the material that is on a hillside and to provide redundant access to the pit. Phase 03 contains 11.9 Mt (or roughly 9.3%) of proven and probable reserves. The average ore grade is 16.8% SAT and the phase has a stripping ratio of 4.0. Phase 3 design is illustrated in Figure 16-5 below.

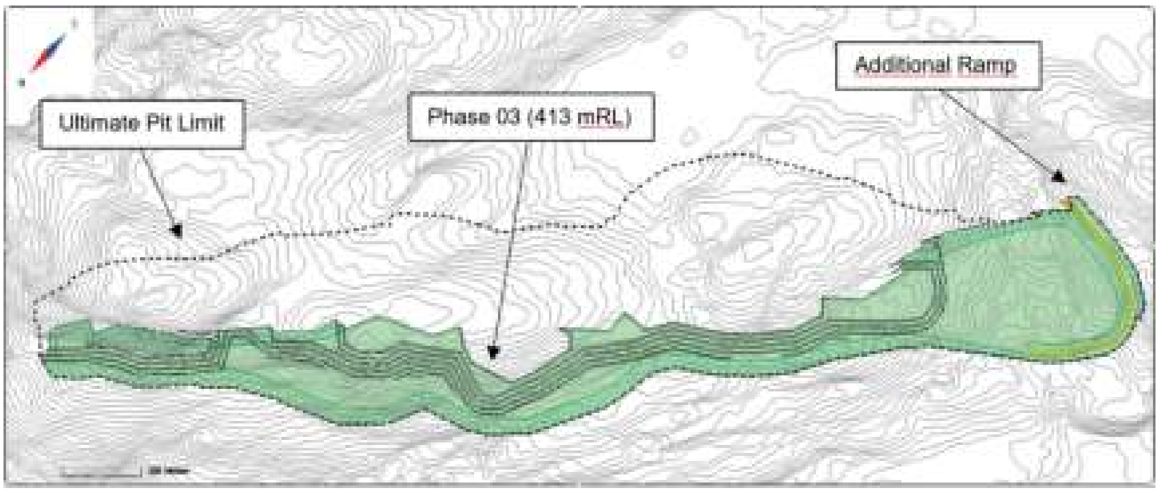


Figure 16-5: Southwest Phase 03 design

- Phase 04 reaches the south-east pit limits surface. Phase 04 contains 32.0 Mt (roughly 25%) of proven and probable reserves. The average ore grade is 17.8% SAT and the phase has a stripping ratio of 2.7. The Phase 04 design is illustrated in Figure 16-6 below.

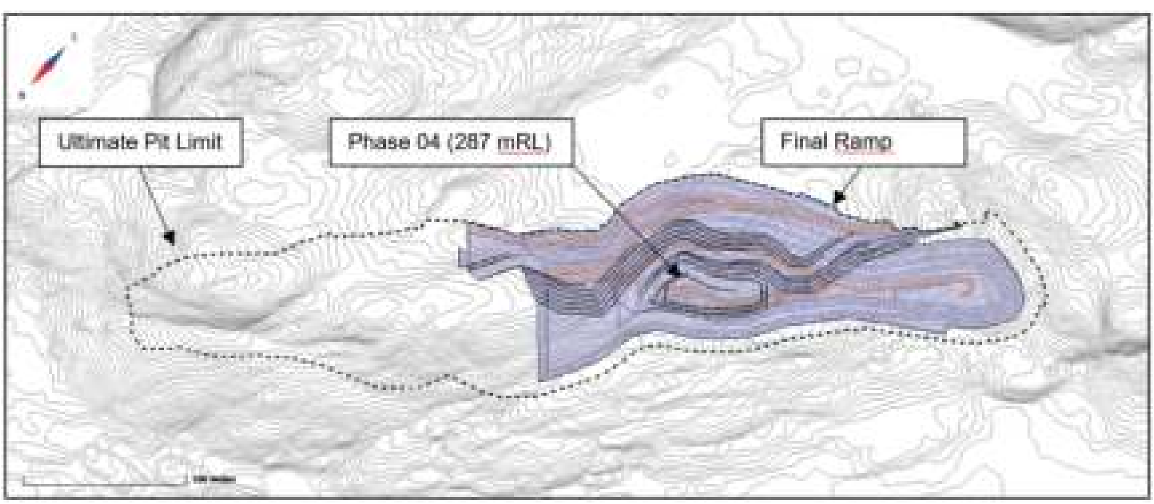


Figure 16-6: Southwest Phase 04 design

- Phase 05 develops the pit to the north and mined to its ultimate depth of approximately 250 metres (elevation 245 m). The top benches of Phase 05 are mined synchronously with Phase 04 to avoid developing an additional ramp access. The phase contains 46.8 Mt (or roughly 37%) of proven and probable reserves. The average ore grade is 19.6% SAT and the phase has a stripping ratio of 2.0.

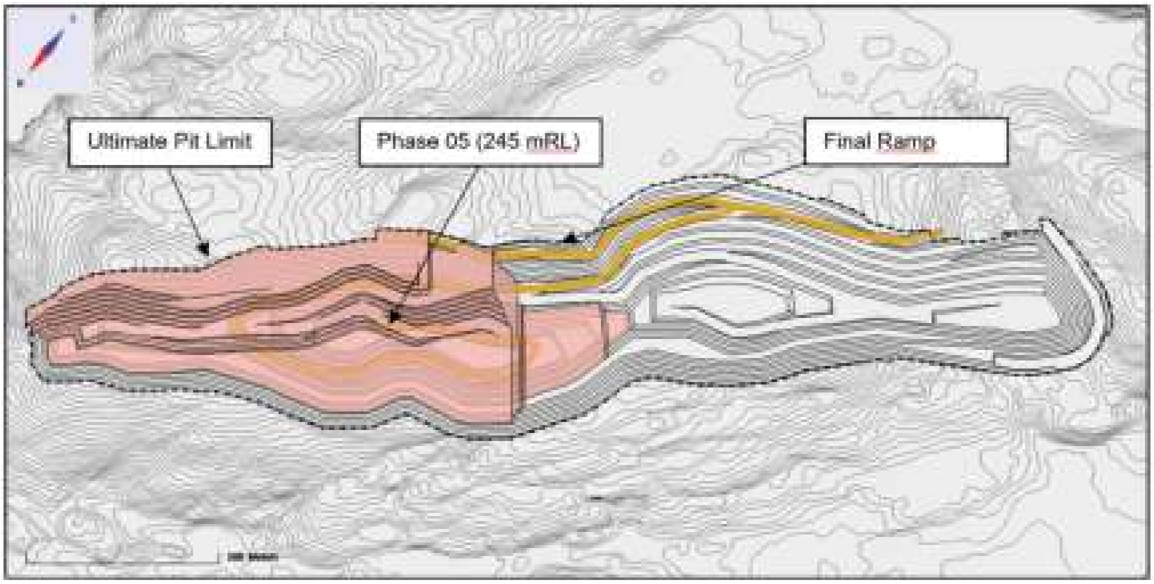


Figure 16-7: Southwest Phase 05 design (Final engineered pit)

A summary of Proven and Probable Reserves for each phase in the Southwest Pit is presented in Table 16-1.

Table 16-1: Mining phases reserves in Southwest Pit

Parameters Units	Proven & Probable Mt	SAT (Satmagan) %	Fe_2O_3 %	V_2O_5 %	TiO_2 %	Inferred, Waste & OB Mt	Strip Ratio t waste/t ore
Phase 00	0.25	14.8	40.2	0.28	9.0	4.5	
Phase 01	13.1	18.4	38.4	0.44	7.3	17.9	1.38
Phase 02	23.7	20.0	42.6	0.49	8.4	31.6	1.35
Phase 03	11.9	16.8	36.0	0.42	6.4	47.1	4.0
Phase 04	32.0	17.8	38.0	0.43	7.1	87.4	2.7
Phase 05	46.8	19.6	42.1	0.48	8.4	94.4	2.0
Total	**127.8**	**18.8**	**40.2**	**0.46**	**7.8**	**282.9**	**2.22**

Notes:
1. Cut-Off Grade: 10% Satmagan.
2. Dilution of 1.0 m into waste blocks adjacent to ore blocks with 50% ore loss and 50% dilution.
3. Totals may not add due to rounding.

16.1.2. Blending Strategy

An ore blending strategy was developed to mitigate two primary risks pertaining to fluctuations in the mill feed: One changing input grades could create inconsistent concentrate grades, resulting in either a product that does not meet specification or fails to yield high quality pellets; and two changes in weight recovery and minerology could require the plant to grind more or less than on average, ultimately creating a risk of too low or high circulation of material and problems from a material handling perspective.

An evaluation of the SAT% grade distribution and lithology was conducted, and the product quality variation was evaluated on a monthly basis. There are two main mining controls available to lower variability in the mill feed characteristics.

The first is to feed the crusher from different areas of the pit in order to achieve the desired blend. This can be accomplished by using mobile equipment like wheel loaders that can move around the pit to different faces, or by having multiple shovels/excavators located in different areas of the pit. It was determined that the number of loading equipment were not sufficient to guaranty a proper mill feed.

The second mining control available to lower variability in the mill feed is the use of stockpiles. Instead of trucks feeding the primary crusher directly, some portion or all of the ore coming from the pit can be delivered to the stockpile. The loading equipment can reclaim the ore from different sides of the pile in order to control what is delivered to the crusher.

The stockpiling strategy is to maintain a blended high-grade stockpile. Since there is much less high-grade material, this is the material that should be set aside when mined and blended in with the more common lower-grade material.

The blending requirement was determined based on the mill constraint and ROM material availability and variability. It was estimated that a 75kt capacity stockpile would be used to handle 29% of the ROM material.

16.1.3. Southwest Pit Life of Mine (LOM) Production Plan

The primary objective of LOM production planning is to sequence the extraction of in-situ reserves.

- The following conventions were adopted to define the mine planning phases:
- Mine pre-production begins at the start of pioneering activities for the open pit and ends once first ore is delivered to the concentrator;
- Mine production begins with first ore to the concentrator and ends once the reserves have been exhausted.

Planning was completed on a quarterly basis from beginning of production to the end of Year 2 and on an annual basis from Year 3 to Year 20. From the Year 21 to the end of mine production in Year 39, planning was completed on increments of three years.

The LOM plan was carried out using MPSO Scheduler in MineSight software using the following criteria:

- Ensure the average SAT% delivered to the crusher during each period is between 18% and 20% to avoid bottlenecking in the process plant;
- Stockpiling of low-grade ore when necessary, in order to maintain the average grade between 18% and 20% SAT;
- Reclaiming of stockpiled ore as soon as possible (avoid maintaining long term stockpiles);
- Defer stripping as long as possible to maximize NPV (however, avoiding drastic “highs and lows” in annual waste tonnages as much as possible);

- A quantity of 1.3 Mm^3 of waste is required for construction (pre-production phase) of the initial tailings dam and an additional 0.3 Mm^3 of waste for the roads construction;
- Year 1 ramp-up is established at 642 ktpa of concentrate or 75% of the expected production of 856 ktpa for Southwest;
- Subsequent production of 856 ktpa of Fe magnetite concentrate;
- Maximum annual sinking rate observed was five to six benches per phase, which is deemed acceptable for an operation of this scale.
- The objectives listed above must be combined with the following practical constraints:
- Mill feed rate constraints;
- Mining rate constraints (related to the "highs and lows" mentioned above) in an attempt to evenly distribute the annual mining fleet requirement, e.g., drills, shovels and trucks;
- Sinking rate constraints based on dewatering and practical mining limitations.

Low grade ore stockpiles will be temporarily placed within the dump footprint. In addition to low grade stockpiles, two 75 kt pre-crusher stockpiles will be employed and located adjacent to the crusher. In any given month, one of these stockpiles will be under construction with the other being reclaimed. The stockpiles will be blended as they are built. This reclaimed ore will be used to reduce variability in the mill feed and approximately 29% of the feed to the crusher will be re-handled ore.

A summary of the annual LOM production plan for the Southwest Pit is presented in Table 16-2

Table 16-2: Annual LOM Plan

Description	Units	Preprod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21-22-23	Year 24-25-26	Year 27-28-29	Year 30-31-32	Year 33-34-35	Year 36-37-38	Year 39-40-41	Total
Total Ore Milled	kt	0	2,500	3,400	3,500	3,200	3,100	3,100	3,100	3,200	3,100	3,400	3,500	3,500	3,400	3,100	3,300	3,500	3,500	3,500	3,500	3,400	10,100	10,200	10,300	9,900	9,500	9,300	2,900	127,800
Average Grade Milled (SAT)	%	0.0	18.5	18.0	18.0	19.3	19.9	20.0	20.0	19.7	20.0	18.4	18.1	18.0	18.1	19.8	18.8	18.0	18.0	18.0	18.0	18.2	18.3	18.2	18.1	18.8	19.7	20.0	21.5	18.8
Weight Recovery	%	0.0	25.7	25.0	24.9	26.9	27.6	27.6	27.6	27.1	27.7	25.1	24.8	24.7	24.8	27.4	25.8	24.7	24.7	24.9	24.9	25.5	25.5	25.1	24.9	25.8	27.1	27.5	29.9	26.0
Average Grade Milled (Fe_2O_3)	%	0	38	37	38	40	43	43	42	41	42	40	39	39	39	42	38	41	37	40	38	40	38.9	38.5	39.7	40.9	42.0	42.5	45.1	40
Average Grade Milled (V_2O_5)	%	0.00	0.43	0.42	0.42	0.48	0.48	0.48	0.49	0.50	0.50	0.45	0.46	0.44	0.42	0.49	0.45	0.40	0.43	0.44	0.42	0.45	0.45	0.45	0.46	0.47	0.48	0.49	0.52	0.46
Average Grade Milled (TiO_2)	%	0.0	7.2	6.9	7.1	7.7	8.8	8.9	8.3	7.7	8.2	7.6	7.3	7.4	7.4	8.0	7.0	8.4	6.9	7.8	7.1	7.7	7.3	7.1	7.6	8.0	8.4	8.5	9.3	7.8
Average Grade Milled (Al_2O_3)	%	0	13	13	13	13	11	12	12	13	12	13	13	13	12	13	13	12	13	13	13	12	12.9	13.2	12.7	12.5	12.3	11.9	11.3	13
Average Grade Milled (P_2O_5)	%	0.00	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.04	0.03	0.03	0.03	0.03	0.03
Average Grade Milled (S)	%	0.00	0.13	0.12	0.14	0.14	0.21	0.20	0.18	0.14	0.17	0.17	0.15	0.16	0.16	0.16	0.14	0.21	0.14	0.18	0.16	0.16	0.16	0.16	0.18	0.18	0.19	0.18	0.19	0.17
Total Concentrate Produced	kt	0	636	852	864	856	857	856	856	862	856	855	856	856	856	856	856	857	856	856	860	856	2,568	2,568	2,568	2,568	2,568	2,568	863	33,186
V_2O_5 in Concentrate	%	0.00	1.34	1.35	1.35	1.43	1.26	1.27	1.32	1.38	1.35	1.31	1.35	1.33	1.32	1.35	1.40	1.19	1.38	1.30	1.34	1.36	1.36	1.37	1.33	1.31	1.31	1.31	1.30	1.33
Direct Mill Feed	kt	0	1,800	2,400	2,400	2,300	2,000	2,200	2,200	2,200	2,200	2,300	2,500	2,500	2,200	1,300	2,400	2,200	2,500	2,200	2,400	2,400	7,200	7,300	7,400	7,100	6,700	5,700	1,900	87,601
ROM to Mill	kt	0	2,480	3,387	3,309	3,182	2,774	3,104	3,103	3,026	3,088	3,155	3,455	3,471	3,121	1,846	3,315	3,092	3,459	3,022	3,400	3,360	10,061	10,224	10,313	9,969	9,413	7,941	2,628	122,701
ROM to Stockpile	kt	249	275	134	0	0	0	0	103	0	249	11	497	751	671	16	490	649	269	296	397	0	0	0	0	0	0	0	0	5,064
Stockpile to Mill (excluding blending at ROM pad)	kt	0	0	19	162	0	326	0	0	153	0	249	0	24	325	1,279	0	380	0	421	50	0	0	3	0	0	51	1,387	255	5,064
Waste Rock	kt	4,500	4,300	4,300	5,700	5,600	8,500	5,000	5,000	6,800	7,800	16,500	15,000	14,000	18,000	17,000	17,000	18,900	10,600	8,700	8,300	6,000	11,800	13,900	21,400	13,000	9,400	4,600	1,100	282,900
Overburden	kt	98	31	38	38	82	44	44	44	86	86	99	99	55	68	68	68	25	25	25	25	12	0	0	0	0	0	0		1,188

The following summarize the key highlights of the LOM plan developed for the Southwest Pit:

- A total mine life of 12 months of pre-production and 39 years of production;
- A total of 4.5 Mt of waste rock will be excavated during the pre-production period and used as construction material for tailings dams and roads;
- The ramp-up in Year 1 for the ore production is 75% of the annual expected concentrate production of 856 ktpa;
- Average strip ratio of 1.80 for the first seven years of operation (excluding the PP phase) and an overall LOM strip ratio of 2.22 (including the PP phase);
- The mill feed varies from 2.4 Mt and 3.5 Mt, with an average of 3.3 Mt over LOM;
- For the first five years, the average Satmagan and V_2O_5 grade are 18.7% and 0.45%, respectively;
- For the first five years, the average V_2O_5 grade in concentrate is 1.35%;
- For the first ten years, the average Satmagan and V_2O_5 grade are 19.2% and 0.47%, respectively;
- For the first ten years, the average V_2O_5 grade in concentrate is 1.34%;
- Relatively smooth and stable year-to-year production to avoid short-term peaks for equipment requirements.

16.2. Waste Rock & Overburden Piles Design

This section presents the waste and overburden pile required for the development of the Southwest Pit.

LVM's current mandate did not extend to the design of the waste rock and overburden pile designs. As a result, the design criteria used for the BlackRock Project were developed based on BBA's experience with similar projects and data gathered from operating mine sites. A swell factor of 30% was used for waste rock and 20% for overburden material when calculating pile capacity requirements. A summary of the waste rock and overburden pile design criteria are presented in Table 16-3.

Table 16-3: Pile Design Parameters

Pile Type	Bench Face Angle	Catch Bench Width	Lift Height	Ramp Width & Max Gradient
Waste Rock	37°	33.4 m	20 m	27 m, 10%
Overburden	26.6° (2H:1V)	25 m	10 m	-

The waste rock pile will be constructed from waste rock and has a design capacity of 116.0 Mm^3. The total ex-pit waste rock is 88.0 Mm^3 (excluding the PP phase), which corresponds to 115 Mm^3 with 30% swell. This pile is located to the southeast of the Southwest Pit. Material will be placed in successive 20 m lifts as required per the LOM Production Plan.

An overburden stockpile was designed between the Southwest Pit and the waste pile. The pile has been built in two lifts of 10 m with face angles of 26 degrees. No berms have been included as the total height is 20m or less in all areas.

Assuming 20% swell of the placed overburden, the total required storage capacity is 0.6 Mm^3. The designed overburden stockpile has a capacity of 1.6 Mm^3, which provides additional capacity of 1.0 Mm^3 to handle any variation between the excavated and estimated quantities.

The location of this pile has been chosen to minimize cycle times during stripping activities and for future mine reclamation.

The design capacity of the waste rock and overburden pile is sufficient to contain the total volume of waste material from the Southwest Pit. A plan view of the site layout for the Southwest Pit and associated waste piles is shown in Figure 16-8.

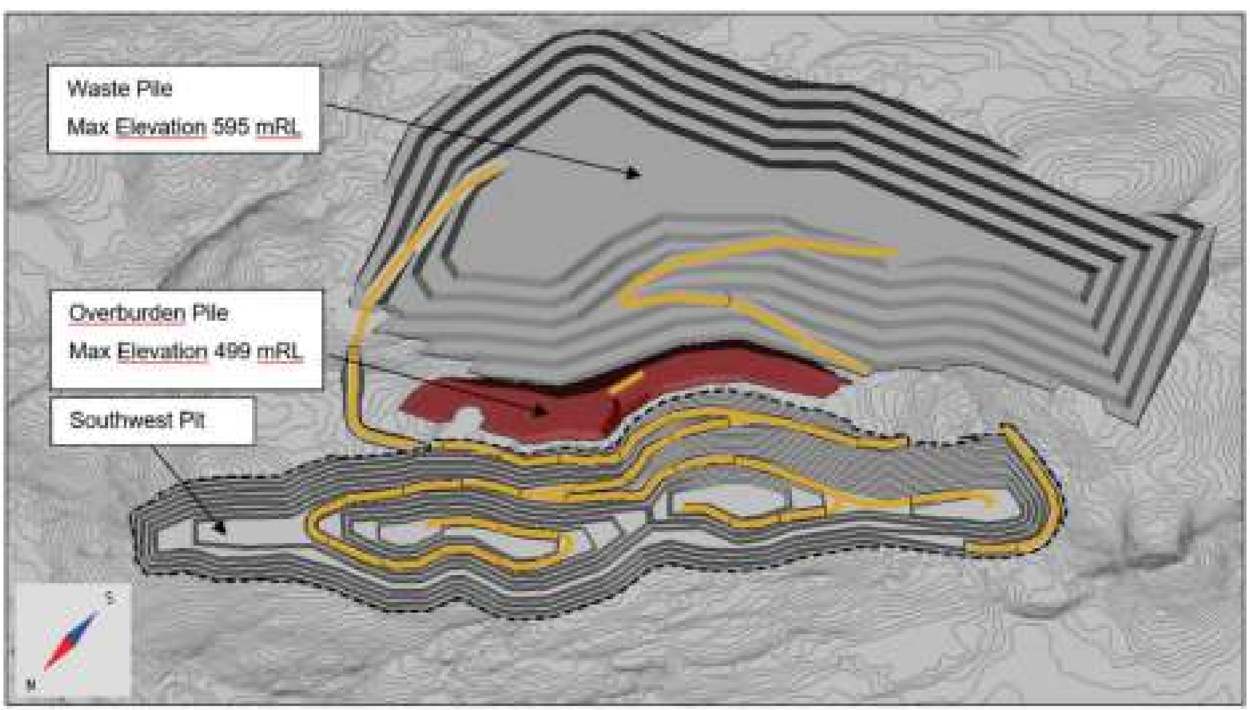


Figure 16-8: Southwest Pit site layout

16.3. Mine Equipment and Operations

The Southwest Pit will be mined using a conventional "Drill & Blast", "Truck & Shovel" mining method. For more flexibility and due to the smaller size of the mining equipment, diesel powered units have been considered for mine operations. The primary and auxiliary mining fleets of Southwest Pit only have been estimated based on the LOM Production Plan in Table 16-3. Operations are based on 2 x 12-hour shifts per day, 7 days per week for 360 days per year, assuming 5 days lost per year due to weather or major shutdowns for maintenance. The current Feasibility Study is based on an "Owner-Operator" approach except for the pre-production period, which will be operated by contractors.

Overburden removal has been scheduled, as required in the LOM Production Plan, in order to expose the material to be moved in subsequent periods.

The waste and overburden material excavated during the pre-production period will be completely used as construction material. Waste material will also be required during the production stage for tailings dam construction.

Selection of the major mining equipment and support equipment fleets is based on operating time assumptions, equipment mechanical availability profiles, machine utilization assumptions, haulage distance and cycle time estimations, as well, performance profiles regarding equipment speeds and fuel consumption. These profiles are based on manufacturer supplied data, information from similar operating mines and BBA experience.

The major equipment fleet consists of the following:

- Blast Hole Drills: 215.9 mm (8 ½ in.) diesel powered DTH blast hole drills;
- Loading Equipment: 9 m^3 capacity hydraulic shovels (diesel) and 10.7 m^3 capacity wheel loaders;
- Haul Trucks: 90 t capacity.

The support equipment fleet consists of the following:

- Track dozers: 475 HP class machine;
- Wheel Dozer: 570 HP class machine;
- Motor Graders: 16 ft class machine;
- Additional service equipment includes a water truck, an in-pit service truck, a lube truck, a 3 m^3 bucket support excavator, a skid steer, light vehicles, a backup drill, dewatering pumps, and lighting towers.

16.3.1. Drilling and Blasting

Production blast hole drilling will be carried out with 215.9 mm (8 1/2 in) diesel powered down-the-hole hammer drills. A 5.4 m x 5.4 m pattern will be utilized for ore, while a slightly wider 5.9 m x 5.9 m pattern will be used in waste rock.

Production blasts will occur on 7 m bench heights. Sub-drilling of 1.2 m will be used to ensure that the fragmentation extends the full bench height. Both ore and waste blast hole will be filled with 2.2 m stemming material in order to maximize charge confinement and minimize fly rock. In addition, a 5% re-drill factor has been added to account for productivity lost to collapsed holes or lost drill steels. The total number of production blast hole drill rigs will vary from one to three units throughout the LOM Production Plan.

Pre-split drilling and blasting will be carried out on all final and interim walls. This practice involves drilling tightly spaced 139.7 mm diameter (5 1/2 in) holes loaded with a lighter charge and detonated to produce uniform fracturing or "splitting" at the final wall location. This will ensure that maximum stability can be maintained along the final wall throughout the mine life.

Blasting requirements have been developed in conjunction with local blasting services providers. The proposed arrangement would involve the use of a 100% emulsion blasting agent with an average density (in the hole) of 1.2 g/cm^3; detonated by two blasting cap and booster per hole. Surface connections and tie-ins would be made using traditional shock tube.

Based on the drilling patterns listed above, the powder factor is estimated to be 0.352 kg/t in ore and 0.347 kg/t in waste. The explosives will be manufactured on-site by the explosive's supplier in purpose-built facilities. The explosives contractor will also be responsible for providing all down-the-hole services. The drilling and blasting parameters and specifications mentioned above have been summarized in Table 16-8. The fragmentation results presented in Table 16-4 have been taken from the fragmentation study performed by BBA during the feasibility study.

Table 16-4: Drill and blast specifications

Parameter	Ore	Waste	Waste in PP	Pre-Split	Units
Drilling Specifications			**Over-blasting**		
Hole Diameter	8 1/2	8 1/2	8 1/2	5 1/2	inches
Bench Height	7.0	7.0	7.0	14.0	m
Sub drill Length	1.2	1.2	1.2		m
Hole Spacing	5.4	5.9	5.6		m
Burden	5.4	5.9	5.6		m
Rock Mass per Hole	749	760	685		tonne/hole
Penetration Rate	25	25	25	22	m/hr
Re-drill	5%	5%	5%	5%	%
Blasting Specifications					
Stemming Length	2.2	2.2	2.2		m
Loaded Length	6.0	6.0	6.0	14.0	m
Emulsion Density	1200	1200	1200		gm/cc
Kg/Hole	264	264	264		kg
Explosive Average Density	1200	1200	1200		gm/cc
Powder Factor	0.352	0.347	0.385		kg/tonne
Fragmentation					
K20	55	76	63		mm
K50	217	304	274		mm
K80	412	550	490		mm

16.3.2. Loading and Hauling

Mine production will be carried out using two to four 9 m^3 hydraulic front-shovels with two typically operating in waste rock and the other in ore. Shovels will be matched to a fleet of 90 t haul trucks based on a 4-pass loading scenario in ore, 5-pass loading in waste and 6-pass loading in overburden.

A wheel loader 10.7 m^3 capacity will be used in-pit in conjunction with the hydraulic front-shovels to maximize flexibility and minimize ore feed delays due to blasting activities. This loader will also be used to re-handle from stockpiles to the crusher.

Haulage profiles have been designed in MineSight 3D using the Haulage component for truck cycle times estimates. Speed data was based on both manufacturer data and BBA's internal equipment database (compilation of data from operating mines and similar projects completed by BBA).

16.3.3. Annual Equipment Fleet Requirements

The annual primary mining fleet requirements were based on the scale of the operation, optimization of the LOM Production Plan and operating efficiency and reliability. The maximum annual fleet requirements occur during years 13 to 16 of operations. Peak requirements for the primary mining fleet are as follows:

- 20 x 90-tonne diesel haul trucks;
- 4 x 9 m^3 diesel-hydraulic shovels;
- 1 x 10.7 m^3 front end loaders;
- 3 x 215.9 mm diesel down-the-hole hammer blast hole drills.

The annual haul truck fleet requirements are shown in Figure 16-9 and are a function of the LOM Production Plan and annual haul cycle distances.

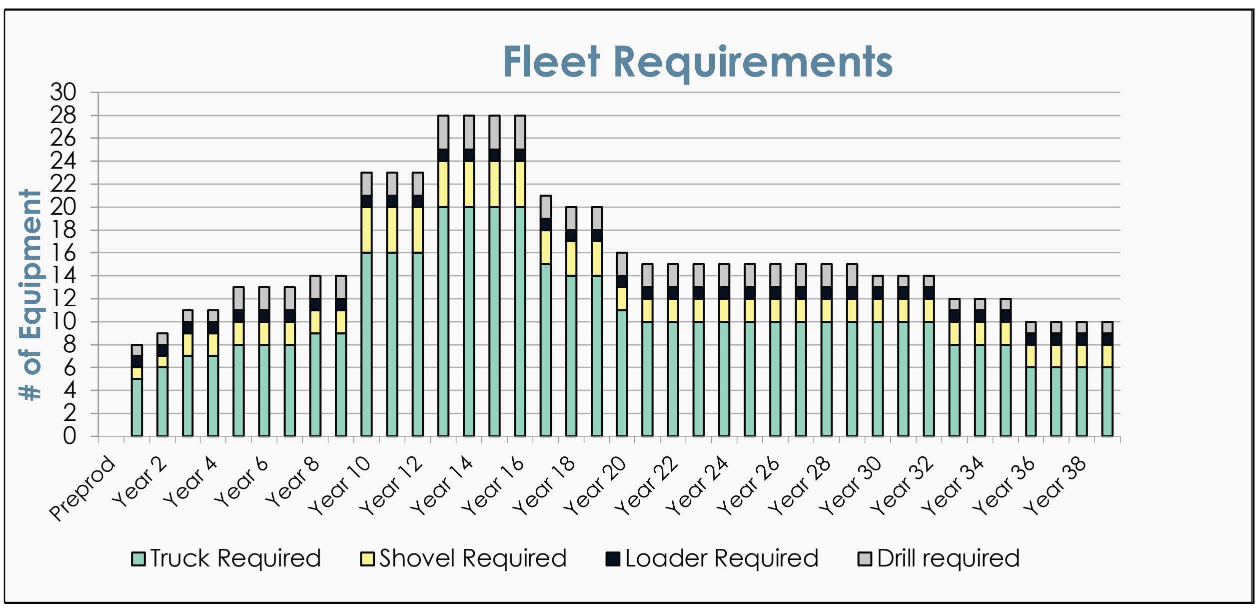


Figure 16-9: Annual Major Equipment Fleet Requirements

To complement the primary mining equipment fleet, a list of support and service equipment was developed by BBA. This list is based on similar operating mines and BBA experience. The requirements for support and service equipment were determined primarily based on the scale of the operation, the size and number of active waste rock piles and the length of haul roads to be maintained.

The support fleet requirement is as follows:

- 1 x Track Dozer;
- 1 x Wheel Dozer;
- 1 x Motor Graders;

Due to the small scale size of the project and to the availability of rental equipment in the vicinity of the mine operation, no back-up units have been envisaged for the support and auxiliary service fleet. An annual provision has been put in the mine operating costs for rental equipment.

The complete list of primary mining fleet and auxiliary support equipment requirements is listed in Table 16-5.

Table 16-5: Primary, support and auxiliary mine equipment requirements

Description	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13-14-15	Year 16-17-18	Year 19-20-21	Year 22-23-24	Year 25-26-27	Year 28-29-30	Year 31-32-33	Year 34-35-36	Year 37-38-39
MAJOR EQUIPMENT																						
Truck		5	6	7	7	8	8	8	9	9	16	16	16	20	20-15-14	14-11-10	10	10	10	10-10-8	8-8-6	6
Excavator		1	1	2	2	2	2	2	2	2	4	4	4	4	4-3-3	3-2-2	2	2	2	2	2	2
ROM Loader		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Production Drill		1	1	1	1	2	2	2	2	2	2	2	2	3	3-2-2	2	2	2	2-2-1	1	1	1
Pre-Split Drill		0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
SUPPORT EQUIPMENT																						
Track Dozer		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer		0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Road Grader		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
SERVICE EQUIPMENT																						
Service Truck		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
PM & Lube Truck		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Support Excavator		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Skid Steer		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Tower		2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Light Vehicle	4	4	6	7	7	7	7	7	7	7	8	8	8	8	8	8	8	8	8	8	8	8
Transport Bus		0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

NOTES :

For the years that are group per 3 years (Year 13 to 39) the equipment number per year is presented and not the sum of the 3 years.

16.3.4. Mine Personnel Requirements

The personnel requirements for the mine include the in-pit mine operations staff, the mine maintenance staff, and the mine technical services staff. The number of personnel reaches a peak of 210 during Year 16 of operations.

The number of operators required for the major mining equipment (haul trucks, shovels, and drills) was determined according to the number of operating units and number of rotations during which time the equipment is in operation. Most of the operators for the major mine equipment are based on a four-crew rotation. Hourly maintenance employee requirements were determined based on the number of machines to be maintained.

17. Recovery Methods

17.1 Mine and Beneficiation Plant

17.1.1 Process Flowsheet

The principal recovery testwork was performed at COREM in Québec City, and at SGS in Lakefield, Ontario to support the selection of a beneficiation process flowsheet for the recovery of a magnetite concentrate. Testwork was performed at a bench-scale level to determine proper particle sizes and to determine the impact of various grades on recovery. Pilot scale testwork was performed to confirm the findings from the bench-scale tests. Material used for the pilot scale testwork was chosen from the drill core selected within the proposed pit design at the time of testing.

The process flowsheets were designed by integrating laboratory data, pilot plant results, BBA's experience on similar projects and suppliers' input. The generalized process flowsheet (Figure 17-1) shows the process from crushing of run-of-mine ore to the dewatering of the final magnetite concentrate. Note that the generalized flowsheet is designed to depict the process in a simple fashion and does not necessarily show the proper number of pumps, conveyors and auxiliary systems.

The crushing circuit will consist of one stage of primary crushing with a jaw crusher. The crusher discharge will be fed to a covered stockpile where belt feeders will reclaim the crushed ore and convey it to the semi-autogenous grinding (SAG) mill. The SAG mill will be in closed circuit with a scalping screen and a horizontal classification screen to control the final grind size. The oversized material from the screens will be returned to the SAG mill for further grinding. The undersized material from the screens will be fed to a primary stage of magnetic separation (cobber).

The magnetic fraction recovered from the cobber will be fed to a ball mill and grounded in closed circuit with hydrocyclone classification. The hydrocyclone overflow will be fed to a secondary stage of magnetic separation (cleaner and re-cleaner). The magnetics from the second stage of magnetite separation will be filtered prior to being sent to the concentrate loadout.

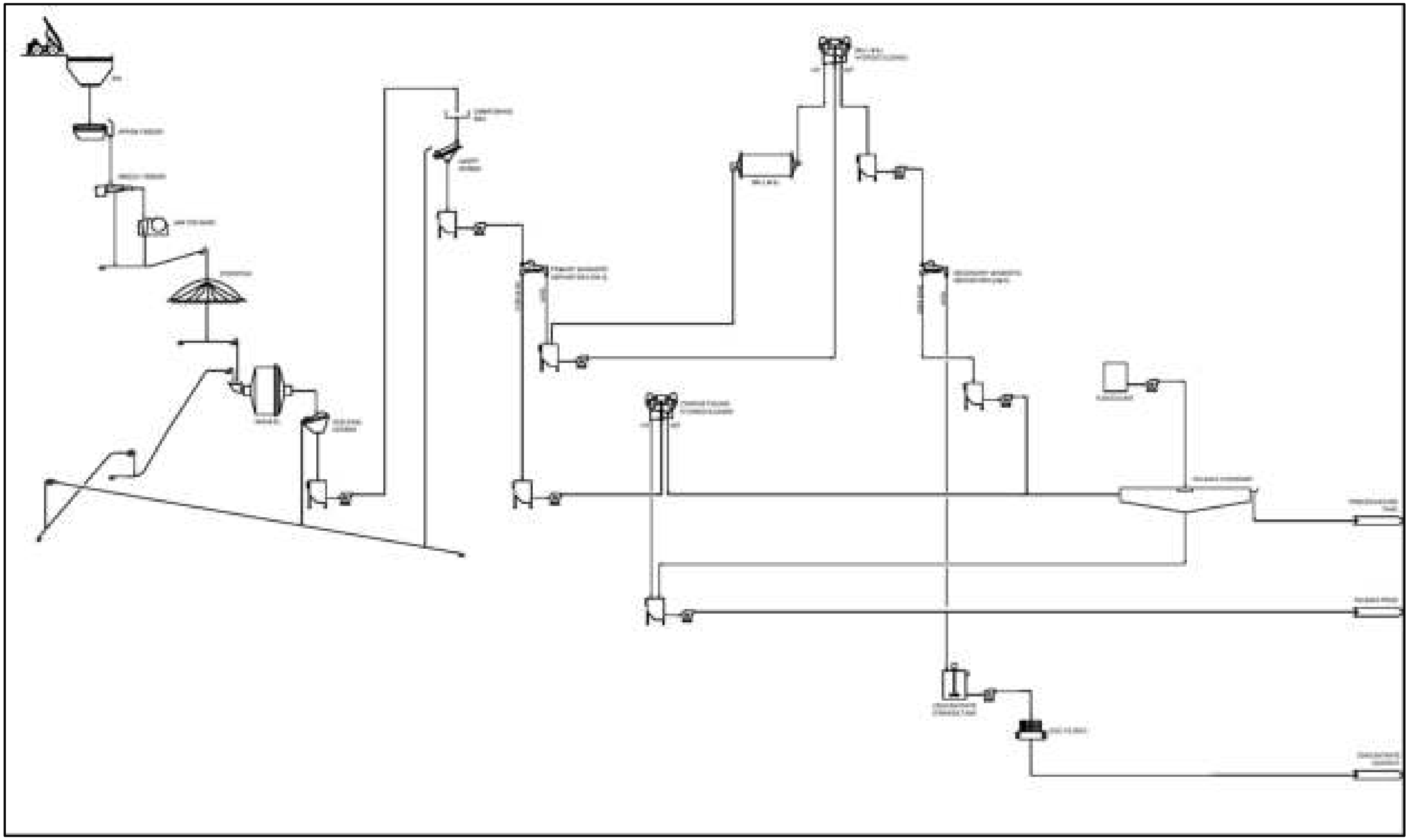


Figure 17-1: Blackrock generalized process flowsheet

17.1.1.1 Primary Crushing

Mining trucks will deliver ore from the open pit mine to a single dump point located above the crusher. A hopper with a capacity of approximately two truckloads will feed an apron feeder which will then transfer the ore to a grizzly feeder. The grizzly feeder will have an opening of 75 mm to 100 mm to allow for fine material to bypass the jaw crusher.

The oversized material from the grizzly feeder will be dumped into the jaw crusher. The crusher was sized with a utilization of 65% to allow for maintenance and for delays in mine truck hauling of ore. A hydraulic rock breaker will be installed adjacent to the crusher to break any rock bridging in the crusher mouth. The jaw crusher design will allow for a maximum rock size of 1.5 m and the mine blasting plan will take this limitation into account.

Crushed rock and the bypassed fines from the grizzly feeder will be combined on a sacrificial conveyor before being transferred to the primary conveyor. Belt skirting will further seal the dribble chute to prevent the spillage of fines and dust.

17.1.1.2 Crushed Ore Handling

The crushed ore conveyor from the primary crushing facility will be transported to a crushed ore stockpile. The stockpile will be covered to keep it dry and to prevent any dust from being picked up by the wind. The stockpile will have a storage capacity of approximately seven live hours (for a volume of 1791 m^3) to compensate for any delays in ore deliveries from the mine or major blockages and breakdowns in the crusher. In the event of longer delays, track or wheel dozers will manipulate the stockpile in order to utilize the full amount of crushed ore present (21 hours or 5,393 m^3).

Three belt feeders will be in a ventilated tunnel underneath the crushed ore stockpile. The belt feeders are designed so that two feeders can handle the full plant capacity under peak conditions. The placement of the belt feeders will allow for a control of the particle size being fed to the plant. The belt feeders will discharge onto a stockpile reclaim conveyor that will feed crushed ore to the concentrator.

17.1.1.3 Primary Grinding Circuit

The first stage of grinding will reduce the particle size (D_{80}) of the ore to 1 mm. In order to complete this task, a SAG mill in closed circuit with classification screens will be employed.

The crushed ore from the stockpile will feed a 30' x 15' FL-FL SAG mill with an installed motor power of 9,000 kW. The SAG will discharge onto a scalping screen. The scalping screen oversize will be sent to a recycle conveyor. The scalping screen undersize will be pumped from a slurry pump box to two static sieves in order to remove a portion of the water from the slurry, remove fines and provide steady, homogenous feed to the horizontal classification screen. The classification screen oversize will be recycled back to the SAG mill using the same conveying system as the scalping screen. The main recycling conveyor to the mill feed chute will recycle the ground ore from the oversize of the scalping screen and classification screen. Three conveyors will be needed to complete the recycling loop; two conventional belt conveyors and one steep-angle-vertical-conveyor. The classification screen undersize will report to a pump box where it will be pumped to low-intensity magnetic separation (LIMS) cobbing.

Forged steel balls, 5 in in diameter and at a total ball charge of 10-15% (v/v), will be used for grinding. A grinding ball storage and handling facility, including a loading funnel above the mill feed chute, will be provided to load the SAG mill, as required.

A variable-frequency drive (VFD) will be used for grinding control. The expected operating window of the normal speed is between 70-80% of the mill critical speed.

17.1.1.4 Primary Magnetic Separation (PMS)

The first stage of magnetic separation will be a bulk recovery of the magnetics from the ore, which has passed through both crushing and a first stage of grinding. By removing a portion of non-magnetic material at this stage a much smaller quantity of material will pass through the secondary stage of grinding.

A density metre will be provided to control the process water addition to the pump box feeding the magnetic separators. Slurry density is adjusted based on the incoming flow to ensure the LIMS are operated at their most efficient volumetric capacity. The feed will be diluted to a slurry density of approximately 30-35% (w/w). Two single-drum counter rotation wet low intensity magnetic separators will be required to handle the plant's throughput. Each unit will provide a magnetic field strength 1150 Gauss.

This LIMS non-magnetic tailings will be collected in a launder under the magnetic separators. The magnetics recovered from the primary magnetic separators will be fed by gravity to the ball mill discharge slurry pump box.

17.1.1.5 Secondary Grinding Circuit

In order to reach the final grind size of 75 µm, a secondary stage of grinding is required. A 15' x 26' FL-FL ball mill in closed circuit with classification hydrocyclones will perform the task of reducing the material down to the desired grind size.

The PMS magnetic fraction, along with the ball mill discharge, will be combined in a pump box where it will then be pumped to a hydrocyclone cluster. The cyclone overflow product, having an average P_{80} of 75, will continue to a second stage of magnetic separation. The cyclone underflow (U/F), with a pulp density of 75 (w/w) solids, will be returned to the ball mill feed chute by gravity for further size reduction. The ball mill and hydrocyclone circuit have been designed to be able to handle a circulating load range between 250% and 350%. Given the wide range of operating conditions, additional water will need to be added under low-flow conditions in order to maintain a minimum line velocity for the piping feeding the cyclones.

A ten-place cyclone cluster, with 500 mm diameter inclined cyclones will be required in order to handle the volumetric capacity of a potential 350% circulating load under design conditions. There will be six operating cyclones, and two spare cyclones, with two in blank position.

The ball mill will have an installed motor power of 4,500 kW as to match the SAG mill motors and simplify the maintenance. The ball mill drive will be equipped with a variable-frequency drive (VFD) for tonnage and grind control. The expected volumetric load of the ball mill is expected to be between 30% and 35%, and when operating the expected normal speed is to be between 60% and 75% of the mill critical speed.

17.1.1.6 Secondary Magnetic Separation (SMS)

A secondary magnetic separation stage will be required to obtain the target concentrate grade of over 62% Fe. The overflow from the ball mill circuit cyclones will flow by gravity to a pump box from where the slurry will be pumped to the secondary magnetic separation stage. Two double-drum, counter-current, wet LIMS with a 6 pole 850/1000 Gauss high gradient magnetic element will be required. The upgrading will occur in two stages: cleaning and re-cleaning, which takes place in the same unit. The feed will first be upgraded on the cleaner drum to produce an intermediate concentrate that will then re-pulp with process water. The magnetic concentrate will then be collected from the pulp by the re-cleaner drum. The secondary magnetic separation tailings will be collected in a pump box and pumped to the tailing's thickener.

17.1.1.7 Magnetite Concentrate Dewatering

The upgraded magnetite concentrate will then be collected in several launders before flowing by gravity to a filter feed agitated slurry tank. There will be a metallurgical sampling point present prior to the filter feed tank. In order to control the density, a density metre will be installed on the feed line to the filter slurry storage tank.

From the filter feed agitated slurry tank, the slurry will be pumped by two VFD peristaltic pumps, each feeding a disc filter with 106 m2 of filtration area. Two-disc filters will be required to be in operation to obtain a filter cake with 8-9% moisture. Each disc filter will have two associated filtrate tanks to better control cake formation and drying. As disc filters are operated under near-overflow conditions; it is expected that there will be up to 20% of overflow recirculation.

17.1.1.8 Concentrate Loadout

The magnetite concentrate will be discharged from the disc filters onto a conveyor. The conveyor will then discharge the concentrate directly into transport trucks with a 100-t capacity and the trucks will transport the material to a train loadout 25 km from the concentrator. The loading of the train trucks will be done in an uncovered area. In the case of truck delays and plant maintenance, the magnetite concentrate can be diverted from the conveyors to four emergency stockpiles, in an unheated building. Front end loaders will be used to load the trucks and keep the stockpiles organized. The emergency stockpiles will have a retention time of four days, which corresponds to 2,980 m^3 of concentrate.

17.1.1.9 Coarse Tailings Cyclone

The cobber non-magnetic tailings will be collected in a pump box prior to being pumped to a hydrocyclone cluster that can hold seven 500 mm cyclones. The purpose of the cyclones is to separate the material into +106 µm particles (underflow) and -106 µm particles (overflow). The coarse underflow will be fed to the final tailings pump box while the fine overflow will be pumped to the tailing's thickener.

17.1.1.10 Fine Tailings Dewatering

The fine tailings from primary and secondary magnetic separation will be combined in a thickener feed box along with various sump pump streams, drum filter filtrate and other water sources. The feed box discharge will feed a 22 m diameter high-rate thickener. Flocculant will be added to the thickener feed well in order to produce an overflow clarity of under 100 ppm solids. The thickener overflow will report to the process water tank, which has a retention time of 20 minutes. The thickened tailings will be pumped to the final tailings pump box where it is combined with the coarse tailings from the coarse tailings cyclone underflow. The solids density will be controlled at 50% solids (w/w) prior to the final tailings being pumped to the tailings pond.

17.1.1.11 Tailings Disposal and Management

The tailings from the concentrator are collected within the tailings pond located to the north of the Southwest Pit. A general overview of site water management can be found in Figure 17-2. The tailings pond allows for a 60-day settling time of the fine tailings. The tailings pond is designed to handle the capacity of the tailings from the Southwest Pit. Water from the tailings pond will be pumped to a polishing pond that has a 30-day retention time. The polishing pond will serve as a source of process make-up water and fresh water (filtered make-up water). Excess water from the ponds will be collected in a monitoring and treatment pond to ensure any water being sent to the environment meets all of the required quality specifications. The average effluent released to the environment per month is shown in Table 17-1 and is based upon average precipitation estimates for the region of Chibougamau, Québec.

Table 17-1: Average effluent discharge to environment

	Jan.	Feb.	March	April	May	June	July	August	Sept.	Oct.	Nov.	Dec.
m³	0	0	0	0	708,269	974,580	894,027	531,441	463,946	517,750	0	0

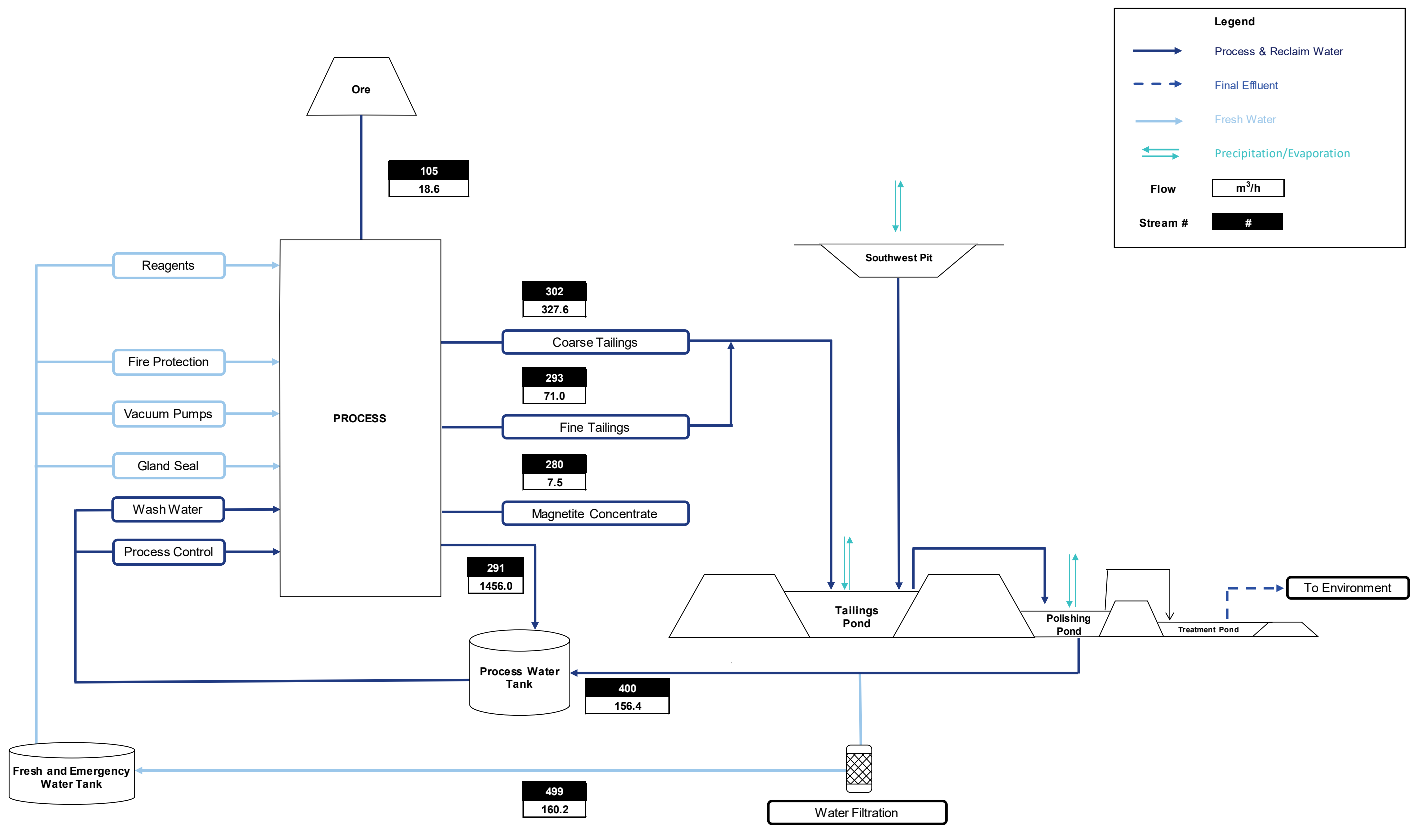


Figure 17-2: Site water flowsheet

17.1.2 Process Design Criteria

The process design criteria were built from the metallurgical testwork programs, mine plan, best practices, clients' and BBA's knowledge, and product specifications. The process design criteria were used to develop mass balances as well as size key process equipment. This plant criteria is detailed in Table 17-2.

Table 17-2: General process criteria

Criterion	Pit	Units
	Southwest	
Annual Throughput (average)	3.13	Mtpy
Design Factors (% increase on nominal throughput)		
▪ Crushing	17.5	%
▪ SAG Circuit	17.6	%
▪ Ball Mill Circuit	41.5	%
▪ Tailings Area	24.1	%
▪ Concentrate Area	33.6	%
▪ Tailings Thickener Feed	24.7	%
▪ Process Water	8.53	%
Head Satmagan Grade (average)	19.6	$\%Fe_3O_4$
Magnetite Concentrate Production Target(average) (2)	0.865	Mtpy
Plant Availability during Ramp-up	75	%
Plant Availability	90	%
Final Magnetite Concentrate Grade	62 (1)	% (Fe_T)
Mineral S.G	3.2-4	t/m^3
Estimated R.O.M Moisture	2-4	%w/w

Notes:

(1) A grade of 62% FeT concentrate is dependent on the residual TiO_2 and V_2O_5 levels as discussed in Chapter 13.

(2) Excluding first and last year of operations for the Southwest Pit.

17.1.3 Equipment Selection

A comprehensive list of mechanical equipment and sizing was prepared based on a mass balance developed using the process design criteria, the mine plan throughputs and proposed process flowsheet. This served to estimate the power requirements, as well as overall Capex values.

17.1.4 Air System

Process air is required for the vacuum drum filters. The process air generation is ensured by two operating process air compressor and one standby process air compressor, each providing 510 SCFM at a pressure of 45 psi. The plant and instrument air generation are supplied by two operating compressors and one standby compressor. These units each provide 589 CFM flow at a pressure of 125 psi.

17.1.5 Cooling System

The cooling system will consist of a closed-circuit water loop used to perform the majority of cooling required for the process equipment. The water in the equipment cooling loop will be chemically treated against corrosion (no glycol). The loop draws its energy from the various process heat exchangers and rejects this energy through a series of cooling heat exchangers into the tailings recirculation circuit. The cooling loop will be a constant volume flow. Heat rejection will be modulated at the level of the cooling heat exchangers by varying flow through the heat exchangers to supply a constant temperature to the process equipment. The design inlet water temperature to the process heat equipment will be a constant 30°C. The outlet temperature from the process equipment depends on the process, however, a heat gain of 5°C to 10°C or more is desirable in order to reduce the size of the cooling loop pumps, piping and heat exchangers. The cooling loop is designed so that all of the process equipment is connected in parallel. As such, all process equipment has the same inlet temperature, as well as a constant volume water flow throughout the year. In the event of a failure of the cooling loop heat exchanger controls, the lowest cooling water temperature delivered to the process equipment is expected to be about 5°C in winter. The process equipment served by the cooling loop consists mainly of heat exchangers integrated into equipment packages such as motor coolers, oil coolers, air compressor coolers, etc.

17.1.6 Energy, Water and Consumable Requirements

17.1.6.1 Energy

The principal energy source for the BlackRock project is electrical energy with the exception of the mine, which uses diesel powered equipment, and the heating of select buildings, which is to be performed by propane. The average power demand of the process plant will be approximately 14.0 MW (see Table 17-3). The grinding circuits represent 60% of the total operating power of the plant and thus the total yearly power demand will depend greatly on the power draw of the mills.

Table 17-3: Process plant power demand for key processing areas including network loss

Area	Power Demand (MW) (1) (2)
Crushing and Stockpiling	1.7
Grinding Circuits (SAG and Ball)	14.8
Others (pumps, agitators, auxiliaries)	6.7
Network Losses	0.5
Total	**23.7**

Note:

(1) The power demand was calculated using various efficiency, load and diversity factors.

(2) Values presented have been rounded for presentation purposes and thus the total might not equal the sum of the parts

17.1.6.2 Water

Water will be drawn solely from the polishing pond for both fresh and make-up water. A portion of the reclaim water will be filtered, used for equipment cooling, and subsequently sent to the freshwater tank. The freshwater tank has a dedicated portion of water that will be used in the case of an emergency (fire, etc.). Freshwater will be used for the gland seals on slurry pumps, sealing water for the liquid ring vacuum pumps as well as for reagent preparation.

Process water is used in various systems for wash water and process control of slurry densities. The majority of the process water requirements are fulfilled by the recycling of process water via the tailings thickener. The process water lost in the tailings and concentrate will be made up of reclaim water from the polishing pond.

17.1.6.3 Consumables

The BlackRock flowsheet requires the use of several consumables. The major consumable costs and consumption rates were estimated to build the process operating costs. Key consumables include; crusher liners, grinding mill liners, grinding media, screen media, filtration cloths, flocculant and fuel.

17.2 Metallurgical Plant

17.2.1 Plant Layout

The BlackRock Metals layout has been defined in cooperation with Tenova according to the following criteria:

- Logical utilization of the site geotechnical characteristics;
- Best process flow of materials inside the plant;
- Compact building and civil construction;
- Minimized piping, cabling, wiring to reduce installation time and cost;
- Safety and ergonomic considerations for plant workers.

The BRM plant site is located approximately 2.5 km from the dock at Port Saguenay. Material will be transported from the dock to the plant site and from the plant site to the dock by truck. The road to the dock is well maintained and is currently used to haul bulk materials to and from the dock. Most of the site location has already been cleared of trees, part of the plant area is levelled and filled with crushed rock.

The BRM site is characterized by a non-homogenous soil with bedrock at different elevations below the surface. Based on the data from the boreholes the positions of the heaviest equipment and parts of the plant (Reactor tower, OSBF etc.) have been located on the areas where bedrock is closest to the surface. The BRM layout is characterized by its compact design that utilizes a good part of the already levelled ground. Some additional cut fill will be required to level the site to 140 m above sea level.

17.2.2 Metallurgical Flowsheet

BlackRock intends to install a metallurgical plant to produce high purity merchant pig iron, titanium slag and vanadium slag. The BlackRock plant process will consist of pelletizing and pre-reduction of the VTM concentrate, followed by smelting, converting and refining of DRI to produce high purity nodular pig iron, commercial titanium slag, alloy metal strip and vanadium slag. The process is illustrated in Figure 17-3. The block flow diagram represents the process in more detail (see Figure 17-4).

Figure 17-3: Saguenay Metallurgical complex

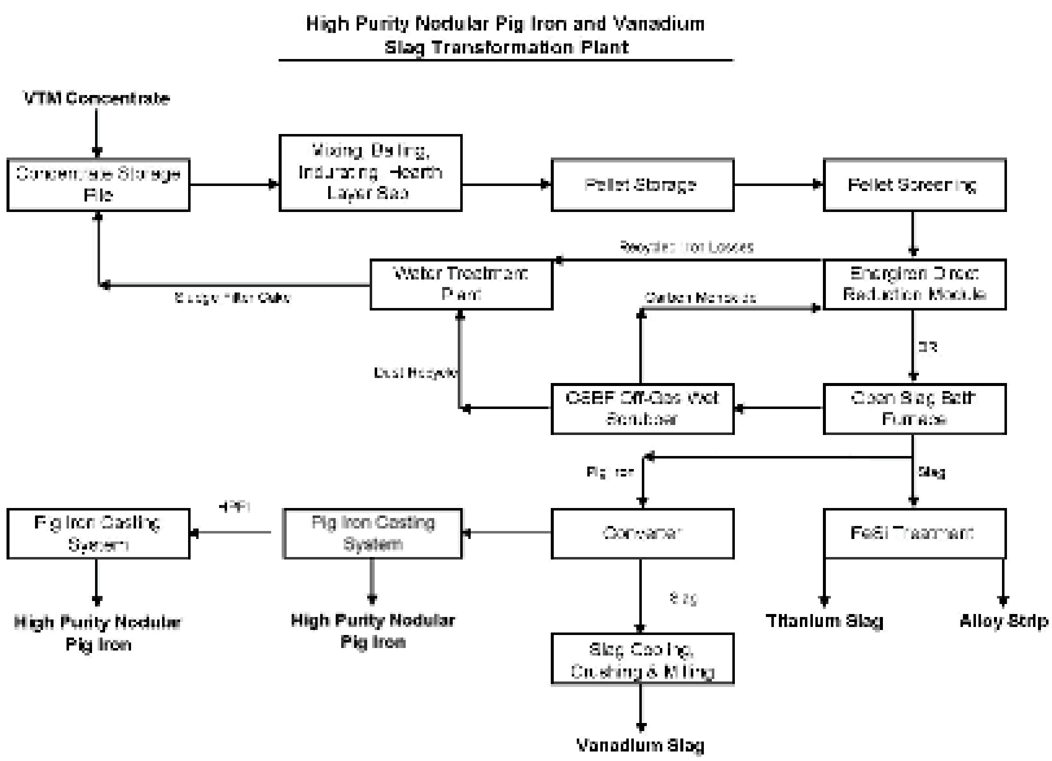


Figure 17-4: Process block flow diagram

17.2.2.1 Concentrate Receiving and Storage

The VTM concentrate will be transported by train to site. The storage area comprises a railway spur, a thaw shed, rotary car dumper, and material handling equipment. The storage area can hold 20,000 tons of concentrate, equivalent to a week of production. Two front-end loaders will act as the means of reclaiming the VTM concentrate from the storage area to a belt conveyor which will feed the pelletizing plant. The capacity of the conveyor 450 t/h; however, the average operating throughput will be 120 t/h.

17.2.2.2 Pelletizing Plant Technology

Tenova selected the Straight Grate Pelletizing Technology for the BRM plant due to its simplicity, flexibility, low operating and maintenance cost as well as its wide acceptance for production of DR grade pellets throughout the world. Currently, all pellets produced in Canada of both blast furnace grade and direct reduction grade utilize the straight grate process.

The pellet plant will be designed to produce 113 tph of screened and cement coated DR grade pellets, suitable as feed to a HYL DR module. Test work has confirmed several plant design criteria such as: particle size distribution, fired pellet chemistry, physical properties, metallurgical properties, and additive feed rates. Annual expected production of pellets is 906,712 tonnes prior to coating.

Raw materials required for pellet production include; blended iron ore to the required balling size distribution (nominally 100% passing 35 mesh (425 µm), 55% passing 325 mesh (45 µm), with a Blaine specific surface of 1290 cm^2/g.), limestone for fired pellet chemistry specification and bentonite as a binding agent for green pellet formation. Additives will be received at the size distribution required for pelletizing; typically, 100% passing 200 mesh (74 µm). Ground additives will be received in PD trucks or rail cars and pneumatically conveyed to storage bins in the mixing area. Additives such as limestone, mineral dolomite, and bentonite will be stored in 400 m^3 silos, if not in use. VTM concentrate will be stored in a 1000 m^3 day bin.

Discharge from the iron ore and additive bins will be proportioned to give the final pellet chemistry desired. The iron ore bin will be equipped with two variable speed discharge conveyors (with scale) and the ground additive bins will have loss-in-weight screw feeders.

Recycled dust from process gas electrostatic precipitators (ESPs) and housekeeping baghouses will be added to the iron ore and additives. The dust is collected at the end of the indurating process and its chemistry matches the desired fired pellet chemistry. Precise metering of the recycled dust and inclusion in the determination of the additive feed rates is not required.

Filter cake, ground additives and recycled dust will be fed into a high-intensity, horizontal paddle mixer. A small amount of water may be added in the mixer to produce a balling feed blend at or near the optimum balling moisture. The moisture content will be controlled with the aid of a moisture analyzer. The mixed balling feed will then be conveyed to two balling feed bins, which feed two 7.62m diameter balling discs. By controlling disc parameters such as rotational speed and final water addition with fine spray nozzles, green pellets with the correct size distribution and sufficient strength to travel to the indurating machine will be formed. Through a system of conveyors, the green pellets are then conveyed to a single deck roller screen that distributes the feed onto a 28 m long, 112 m^2 indurating machine for heat hardening.

The design for the Metso travelling grate iron ore pellet plant incorporates the Dravo Process, utilized for non-compacting feed. In this process the unfired green pellets are dried, preheated, indurated, and cooled on a continuous travelling grate, without intermediate transfers. Process air introduced for pellet cooling is recirculated from the cooling zone of the grate in a multi-pass manner to the other process zones to maximize thermal efficiency. Only relatively cool, moisture-laden gases are discharged to process gas cleaning and then to the atmosphere.

17.2.2.3 Screening and Coating

After separation from the hearth layer, the fired pellets will be transported directly to the screening station and coating system, before being fed to the direct reduction plant. If the direct reduction feed bins are full, the pellets will be diverted to an 8,000-ton storage pile, equivalent to three days of storage, for later use. The coating cement will be stored in a 70-ton storage bin, equivalent to a week’s worth.

The screened and coated pellets will be conveyed by a series of conveyors and a Flexowell vertical elevator to the Reactor Tower.

17.2.2.4 Direct Reduction Technology

When selecting the technology that would produce high purity merchant pig iron, BRM emphasized that environmental impact would be of huge importance to local communities, governments, investors and other stakeholders of the project. This required an alternative to the traditional coke-based blast furnace production route. In general, this would require pre-production of some kind followed by melting in an electric furnace. The selection of the iron ore reduction process has been a key decision, not only because of the impact that such decision has on the environmental footprint of the plant, but also because different routes of iron reduction have had varying degrees of technical and commercial success. The selection also has implications on different types of downstream equipment and on changes to capital expenditure and operational cost.

Some of the technologies for iron reduction that are available commercially include: Midrex direct reduction technology, HYL III process, HYL Energiron Zero Reformer (ZR) process, Fastmet/Inmetco Rotary Hearth Furnace process and Finmet technology.

The final process selection was carried out on comparisons relevant to energy requirements, environmental impact, plant size and infrastructure and reference plants in operation. The process route that was selected for the study is the direct reduction gas-based route through the Energiron technology, coupled with electric arc furnace melting and pig casting technology.

The Energiron process, based on the zero-reformer (ZR) scheme, is a major step in reducing the size and improving the efficiency of direct reduction plants (see Figure 17-5). Iron ore is converted into metallic iron by means of hot reducing gases that flow in opposite direction to the solid material inside the moving bed reactor, operated at a pressure of 6 bar on the top. In the ZR scheme, the reducing agent makeup is directly fed inside the reactor, where it's converted into reformed gas by exploiting the catalytic power of the DRI. Therefore, oxygen is removed from the iron ore by chemical reactions based on hydrogen and carbon monoxide, to produce highly metallized DRI. The CO_2 and water produced by reduction of iron ore are converted back to H_2 and CO by self-reforming reactions.

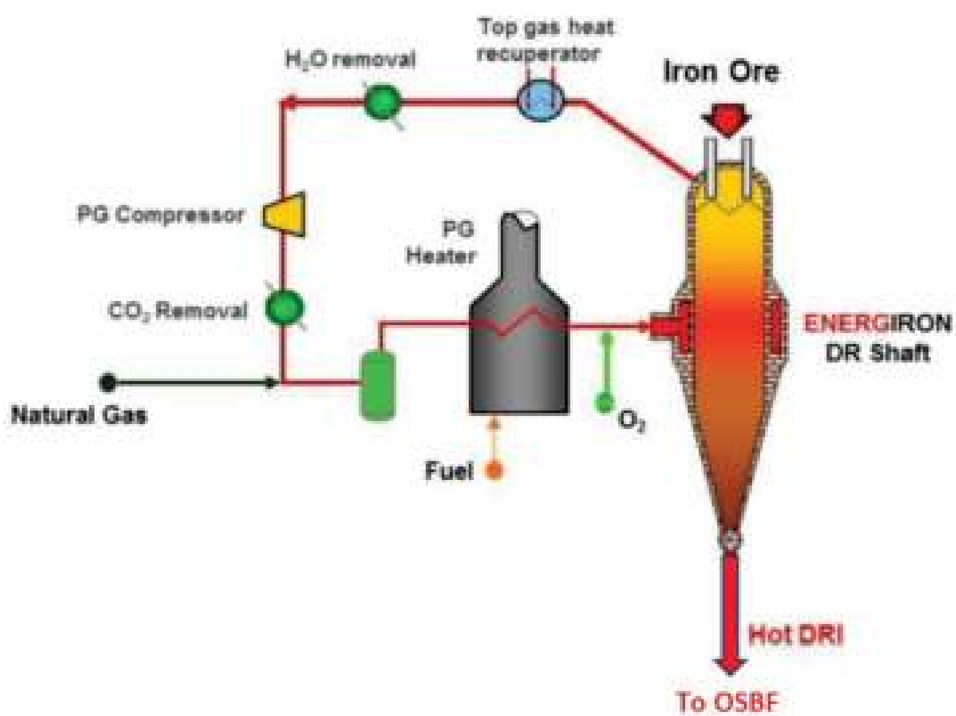


Figure 17-5: Energiron process, based on the zero-reformer (ZR) scheme

As the reducing gases are generated in the reduction section of the reactor, there's no need for an external reformer and the overall energy efficiency of the direct reduction is optimized. Most of the energy supplied to the process is taken by the product with minimum energy losses to the environment.

The average reactor capacity will produce 113 t/h of hot DRI which will feed the Open Slag Bath Furnace (OSBF). The hot DRI will be evenly distributed into the OSBF via a series of 12 feeding chutes.

Operating conditions of the ZR process are characterized by high-temperature (>1080˚C) and high-pressure (6-8 bar at top gas). The elevated pressure allows a higher productivity and low reducing gas velocities minimizing dust losses through top gas carry-over. This lowers overall iron ore consumption, which in turn lowers the overall operating costs. A distinct advantage of this process scheme without an integrated reformer is the wider flexibility for DRI carburization. DRI carbon levels up to 5% can be obtained, due to the prevailing conditions of high methane (CH_4) concentration within H_2-CO and the high temperature of the bed (>860˚C), which favours the diffusion of Carbon into the iron matrix and the precipitation of Iron Carbide (Fe_3C)

The DRI plant for BRM will have an annual capacity of 711,245 tpy hot DRI with a metallization of 94% (average), a carbon content of 4.3% at an hourly capacity of 89 t/h.

17.2.2.5 Smelting Technology

The processing plant consists of the smelter and the ladle furnace. The smelter consists of an electric furnace and its auxiliary system which include the feed system, gas treatment, liquid metal and slag handling and various cooling systems. The converter consists of a converter unit, metal and slag handling and gas treatment.

The smelter plant produces intermediate hot metal containing vanadium and a titanium rich slag. The titanium rich slag is treated with ferrosilicon (FeSi) to decrease the vanadium and other metals content to produce a commercial titanium slag and a commercial alloy metal strip containing mainly iron, vanadium and manganese. The Vanadium containing hot metal is fed into the converter for refinement to produce hot metal conforming to the desired pig iron grades and a vanadium rich slag (V-slag). The refined hot metal is fed to the pig caster to produce the final product of High Purity Pig Iron or Nodular pig iron (HPPI/NPI). The casted HPPI/NPI is stored, ready to be sold. The V-slag is cooled and then fed to a comminution and recovery circuit before the final product is bag and store before shipping to FeV conversion plant.

DRI produced from the HYL Energiron process is smelted in an AC electric furnace. The electric furnace is designed to operate in open arc mode; hence, the furnace is referred to as an open slag bath furnace (OSBF). This technology has been selected because the power input is less dependent on the slag chemistry and eliminates the need for using coke in the furnace. Open arc operation allows for the treatment of fine feed material in the furnace as there is no feed layer on top of the liquid bath and the charge permeability becomes irrelevant. Open arc mode enables higher electrical resistance in the OSBF, compared with furnaces operating with the electrodes immersed in the liquid slag or submerged in the furnace charge, results in higher operating efficiencies and better flexibility in operation.

OSBF uses DRI as the raw material, anthracite as the reductant and calcined dolomite/ limestone as the flux. The DRI is fed continuously from the Energiron process into the OSBF while it is hot (±600˚C) to minimize the electrical energy consumption in the furnace. Liquid metal and slag are tapped from the furnace intermittently (see Figure 17-6). The furnace off gas contains approximately 68% CO and is extracted at ±1900˚C and sent to a wet gas scrubber and the tapping fume is sent to a bag filter to clean the gas. The dust collected in the scrubber and the bag filter is recycled to the pellet plant; the off-gas tails are recycled back to the process gas heater.

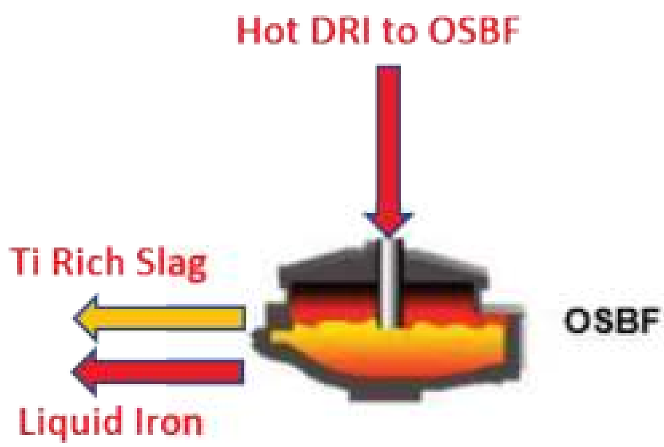


Figure 17-6: Tapping of liquid metal (hot metal) and slag

The hot metal tapped from the furnace, at ±1450˚C, is estimated to contain 3.4% C, 0.4% Ti, and 1.02% V together with other minor constituents and iron. Hot metal is tapped approximately every 2.5 hours into separate ladles and is transferred to the converter for selective oxidation of vanadium into vanadium slag (V-slag). The V-slag is removed from the furnace building, cooled, and fed to a circuit comprised of crushing, milling, metal recovery and slag bagging. The product is picked up by a buyer from the site.

The slag, tapped from the furnace, at ±1630˚C, contains approximately 61.13% TiO_2 and 0.95% V_2O_5 together with other oxides. The slag is removed from the furnace building, cooled, and then transported to another building that is divided into sections for stockpiling and further cooling, FeSi treatment and storage of the titanium slag and metal alloy strip.

Based on theoretical calculations and the information available to Tenova from similar projects, it will be possible to reduce and recover V-content of the slag by adding ferrosilicon (FeSi) after it is tapped out of the furnace in a separate tundish/launder arrangement. The added silicon in the FeSi reacts with vanadium oxide and the other oxides such as FeO, MnO, and Cr_2O_3 in the slag at varying degrees to form an alloy high in silicon, iron and manganese. Metal collected at the bottom of the tundish is tapped into a sand mould or ladle placed below the taphole under the tundish.

The OSBF furnace design was based on 7,600 hours availability (91.3%), which is in line with the DRI reactor availability of 8,000 hours plus some margin for design safety. The furnace load factor is estimated at 95% based on industry norms and Tenova's experience. The operating factor, availability multiplied by the load factor, is 86.8%. The furnace processes 711,245 tons of DRI per annum and produces 545,425 tons of hot metal, 117,728 tons of hot slag, 29,467 tons of hot gas containing 11,169 tons of dust per annum. Vanadium content of the OSBF metal is reduced in the converter from 1.02% to 0.04%. The balance of the vanadium is recovered to the V-slag.

17.2.2.6 Metal Refining

The main objective hot metal refining stage conducted in the converter is to oxidize V and to transfer the created V-oxides into the slag. As this reaction cannot proceed selectively at the presence of C, Si, Ti, Cr and Mn, these metalloids must be oxidized as well. The O_2 potential is higher for Si and Ti than for V. Therefore, all the present Ti and Si – in the normal hot metal – will be quicker oxidized than V. The O_2 affinity for Mn and V is comparative and slightly higher than for Cr.

The lines for the standard free energies of formation of various metal oxides run fairly parallel (see Richardson & Jeffes diagram); therefore, the above-mentioned O_2 – potentials are covering the full range of common operating temperatures.

In opposite to these lines is the trend of the standard free energy of formation of CO. Higher O_2 pressures do shift the line of the free energy up to lower O_2 potential so that the oxidation of V should take place even more preferentially. The above-mentioned O_2 potential, however, does not give any evidence about the speed of reaction which is of most importance for the process.

Objective of the converter:

- To oxidize the vanadium from the molten iron into a dry slag in the form of Vanadium Pentoxide;
- This vanadium slag to contain >20% Vanadium Pentoxide (V_2O_5);
- The efficiency of removal of Vanadium to slag to exceed >90%;
- To obtain an iron to blown metal yield more than 95%;
- To have a carbon content in the blown metal of >3.7%;
- To achieve a final temperature of between 1380-1410 ^{0}C in the blown metal.

Hot metal with high V content (approx. 1.02% V in average) is produced in the Open Slag Bath Furnace (OSBF). The Converter design is based on two ladles per OSBF tapping period intervals of 2.5 hours (150 min). The average hot metal weight in ladles from OSBF is approx. 86 t. The total hot metal production of OSBF metal input to converter will be approx. 550,000 tpy, or approximately 6395-6400 ladles per year. The ladles tapped from OSBF are transferred to and poured into the converter using overhead crane. In case of increased quantity of OSBF slag in the ladle (>500 kg slag) - depending on the hot metal composition from OSBF (contend of Si and Ti) - slag pre-skimming might be required before pouring into the converter in order to obtain a more valuable V rich slag.

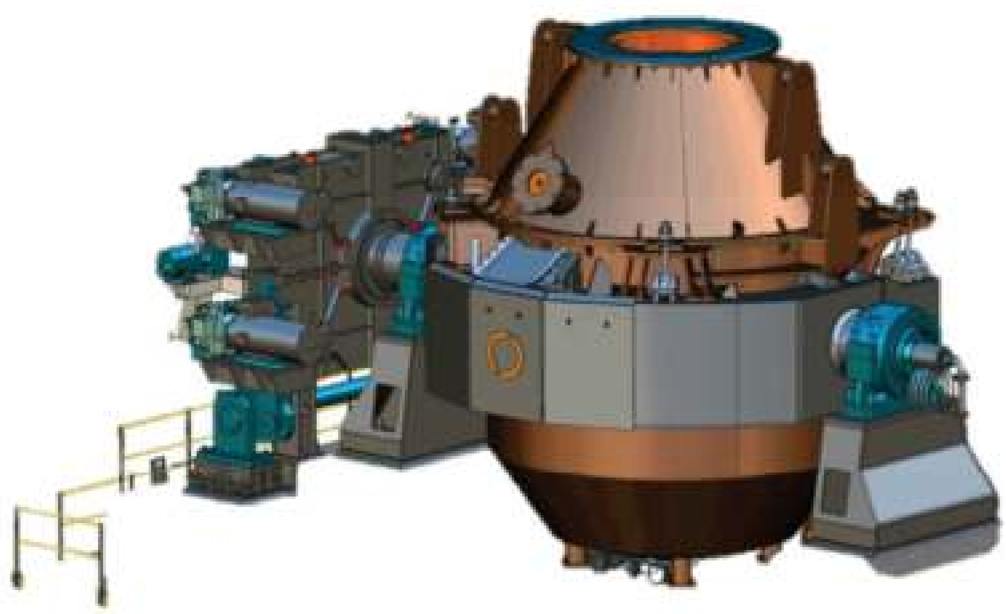


Figure 17-7: Typical deV converter vessel

From the converter, pig iron is transferred to the metal casting in a hot metal transfer ladle. The vanadium slag is tapped during the convertor sequence steps into a slag pot, the vanadium slag is cooled, crushed, magnetically separated and grind to the desired particle size where after it's loaded into bags for shipping to a third-party vanadium transformation plant to extract the vanadium from the vanadium slag and produce Ferrovanadium.

17.2.2.7 Metal Casting

The metal casting station will be sized to follow the 2.5 hours tapping sequence; therefore, the casting station will have two pig iron casting lines. The molten metal will be poured into the molds and cool as the molds move towards a storage area; the cooling will be enhanced by water sprays. The metal temperature will be critical; it should always be above 1310 °C. The pig iron casting system produces approximately 536,481 tonnes per annum. Each ingot cast has an average size (length x width x depth) of 380x150x75 mm. The ladle containing the hot HPPI is titled and poured into one common main runner and then into two feeding runners. The molten metal is poured into the molds and cools as the strand travels up the incline, with the assistance of water sprays. At the top of the incline, the solidified pigs drop off while the empty molds continue travelling down the incline, where they are coated and dried to prevent pigs sticking in the mold. The pigs are collected and organized on a storage pad.

17.2.3 Process Design Criteria

The process design criteria were built from the metallurgical testwork programs, mine plan, best practices, clients' and BBA's knowledge, and product specifications. The process design criteria were used to develop mass balances as well as size key process equipment.

17.2.3.1 Pelletizing Plant

The plant design and characteristics are based on the plant inputs. Table 17-4 shows the parameters established for the Pelletizing Plant.

Table 17-4: Pelletizing Plant parameters

Parameter	Units	Value Specification
Operating hours	h/y	8,000
Nominal production of fired screened pellets	t/y	904,000
Fired pellet basicity	-	0.6
VTM concentrate blaine surface area	cm^2/gm	1200-1400
Average VTM concentrate bulk density	t/m^3	1.8 to 2.2
Bentonite binder addition	%	1
Fresh VTM concentrate	t/t pellets	0.911
Iron ore concentrate	t/t pellets	0.972
Electrical Power	kWh/t pellets	69
Natural gas	Gcal/t pellets	0.1
Industrial makeup water	m^3/t pellets	0.02

17.2.3.2 Direct Reduction Plant

The plant design and characteristics are based on the plant inputs. The design basis for the Energiron DR Plant considers the pellet feed to the reactor and the natural gas requirements. Table 17-5 shows the parameters established for the DR Plant.

Table 17-5: DR Plant parameters

Parameter	Units	Value Specification
Product	-	Hot DRI
Metallization	%	94
Carbon content	%	4.3
Nominal hourly capacity	t/h	88.9
Nominal annual capacity	t/y	711,245

Parameter	Units	Value Specification
Operating hours	h/y	8,000
Number of reactors	-	1
Number of has heaters	-	1
Number of process gas compressors	-	1
Number of CO_2 removal units	-	1

17.2.3.3 Smelting Plant

The furnace was designed for optimal performance and in accordance with the parameters set for the direction plant. The furnace design is based on 95% utilization (use of available power) which corresponds to industry norms and PYR experience. Table 17-6 shows the parameters established for the smelting plant.

Table 17-6: Smelting Plant parameters

Parameter	Units	Value Specification
Operating hours	h/y	8,000
Availability	%	91.3
DRI feed temperature	°C	600
Utilization	%	95
Operating factor	%	86.8

17.2.4 Equipment Selection

A comprehensive list of mechanical equipment and sizing was prepared based on a mass balance developed using the process design criteria, the mine plan throughput and proposed process flowsheet. This served to estimate the power requirements, as well as overall Capex values.

17.2.5 Air System

17.2.5.1 Pelletizing Plant

The pellet plant utilizes dry, oil-free compressed air as plant air and instrument air during operation. In addition, compressed air is used for the maintenance of facilities. The compressed air will be supplied to the plant at 8.4 kg/cm^2. An air receiver will be in the pellet plant to provide surge capacity. Collected dust is transported to the dust recycle bin in the mixing area of the pellet plant via a pneumatic conveying air system. The air will be dried to a dew point compatible with local ambient conditions.

17.2.5.2 Direct Reduction Plant

Service and instrument air requirements will be satisfied through the compressed air distribution system.

17.2.5.3 Smelting Plant

The compressed air distribution system is purged of undesirable gases to enhance operating efficiency of the compressors, prior to sending service and instruments required to the smelting plant.

17.2.5.4 Cooling System

A closed-circuit system supplies 24 °C cooling water at 5 kg/cm^2 to the indurating furnace inlets. Cooling water will be supplied to the pellet plant for pneumatic conveying air compressors and self-contained process fan bearing cooling systems.

17.2.6 Energy and Water requirements

17.2.6.1 Energy

17.2.6.1.1 Pelletizing Plant

The main fuel source is natural gas, approximately 0.1 Gcal/t pellets are consumed. Electrical power is required for operation of the plant, approximately 69 kWh/t pellets are consumed. An emergency generator in the pelletizing plant is set up to provide emergency power, if required.

17.2.6.1.2 Direct Reduction Plant

Natural gas is the fuel source for the direct reduction plant. Natural gas is necessary to supply the reactant to the DRI plant; natural gas is mixed with recycled reducing gas and injected just before the process gas humidifier. The power consumption design value for the direct reduction area is 5.6 MW. Moreover, the area is equipped with a diesel generator to power the circuit in the case of a power failure.

17.2.6.1.3 Smelting Plant

Electrical Power is used by the furnace. Table 17-7 shows the electrical power design basis for the smelting plant.

Table 17-7: Smelting plant electrical power design basis

Description	Units	Design	Start-up
Total Transformer rating	MVQ	75	
Number of fumace transformers	Off	3	
Operating power (annual peak)	MW	51.3	36.4
Operating power (annual average)	MW	48.8	34.5
Operating resistance	mΩ	1.8 - 5.4	1.2 – 6*
Power Factor	cos θ (@ 3mΩ)	0.84	
	cos θ (@ 4mΩ)	0.90	

17.2.6.2 Water

The water treatment plant is important for supplying process, cooling, and service water to processing areas such as the pelletizing plant, coating station, DRI plant, converter, OSBF, and casting machine.

18. Project Infrastructure

18.1. Mine and Beneficiation Plant

18.1.1. General Site Works

General site preparation will consist of clearing, grubbing, topsoil removal, backfilling and surface levelling throughout the construction areas. Clearing, grubbing and topsoil removal needs were estimated based on borehole logs and BRM's knowledge of the ground cover. Topsoil removal and grubbing are considered to be a single construction activity for material take-off purposes. Clearing is done in and around all construction areas to provide easy access. Topsoil is removed to provide a stable subbase for roads and platforms and to provide stability below the overburden and waste rock stockpiles.

A general overview of the BlackRock site showing property limits, the Southwest pit, tailings management facilities, waste piles as well as site infrastructure can be found in Figure 18-1. A targeted view of the BlackRock concentrator can be found in Figure 18-2. A small portion of the waste rock pile presented may be in the Laugon Lake watershed. In the next stages of engineering design, the waste rock pile will be re-evaluated to ensure that the Laugon Lake watershed is not disturbed.

Site drainage will be achieved with the excavation of drainage ditches along the building platforms, roads and sedimentation ponds.

A frost depth of 2.7 m is also considered for building foundations not sitting on bedrock or underground piping.

18.1.2. Potable Water

The site will have skid-mounted potable water treatment plants, servicing both the process plant area and the mine garage. The treatment process consists of filtration, chlorination, and UV sterilization units to produce potable quality water. The potable water is then pumped in a designated tank and distributed to sinks, toilets and washing facilities.

18.1.3. Sewage

A skid mounted sewage treatment plant will be installed next to the mine garage and will be servicing both the concentrator and the garage areas. The sewage treatment plant utilizes a membrane process. It will produce a clear outflow which will be directed through an above-ground, insulated pipe with heat tracing in an existing draining ditch, leading to the tailings pond area. Remaining sludge will be collected, stored, and disposed of by vacuum trucks off site by an independent company.

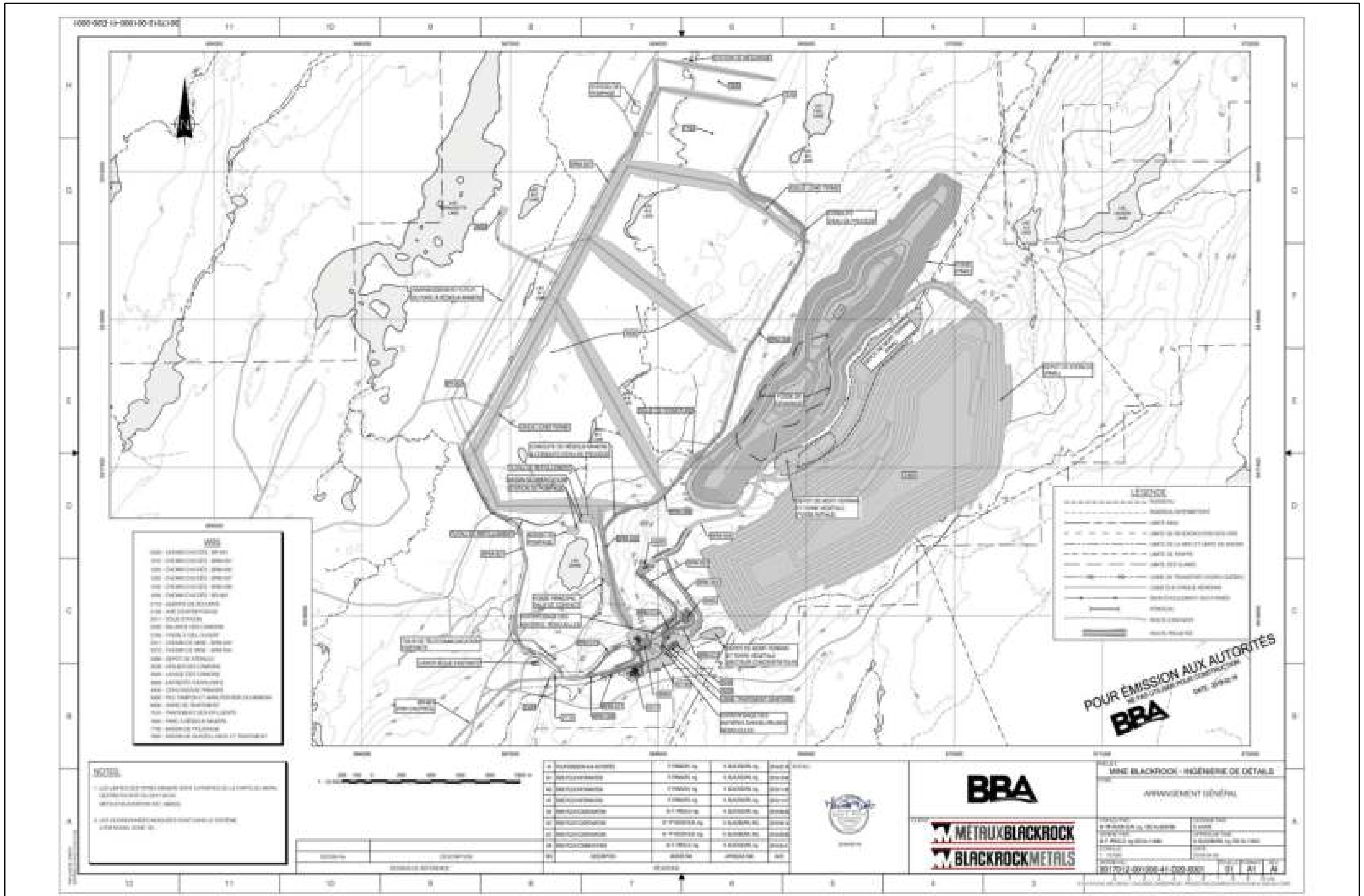


Figure 18-1: BlackRock site plan

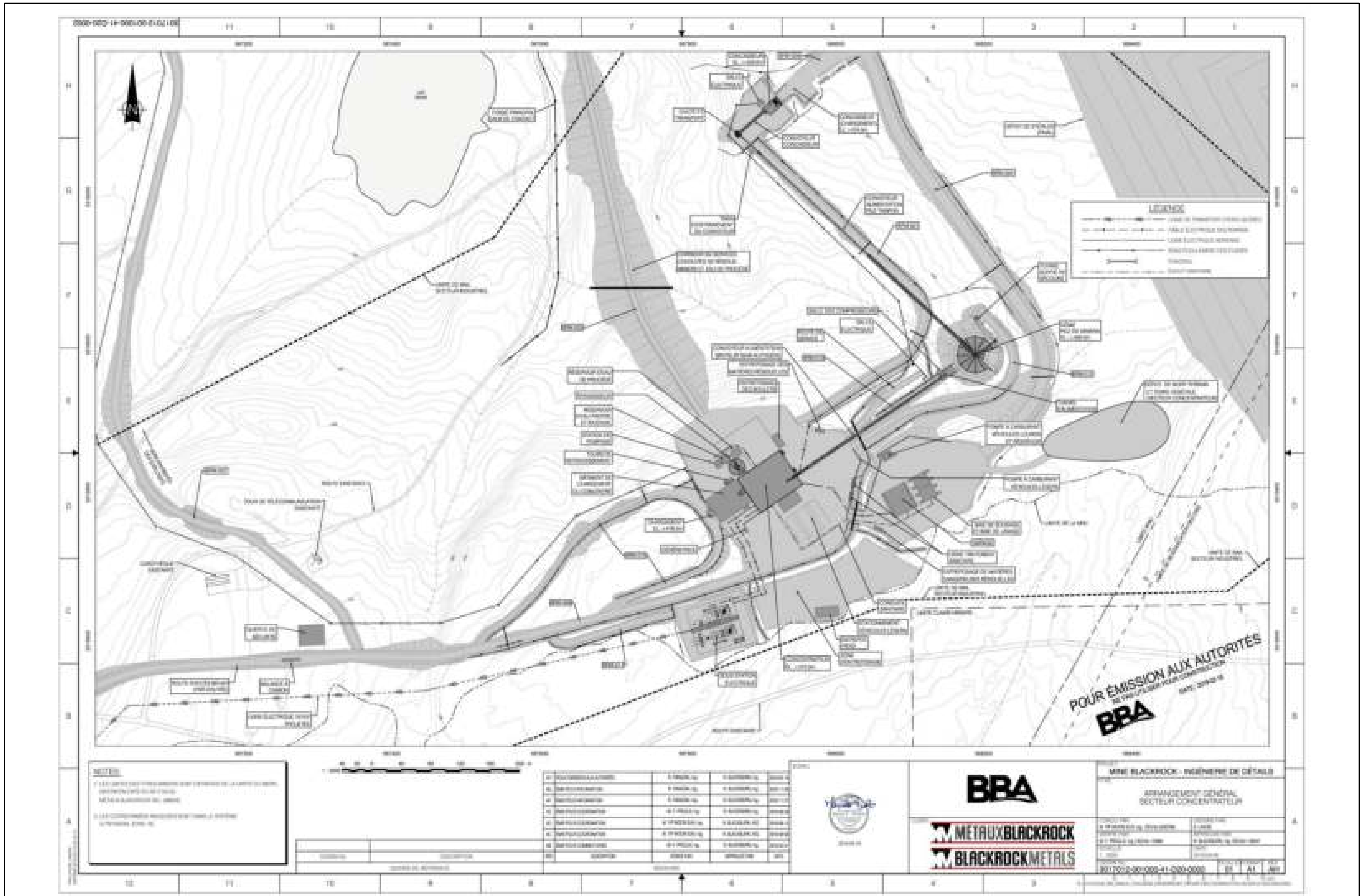


Figure 18-2: BlackRock process plant portion of site plan

18.1.4. Roads

18.1.4.1. Primary Site and Access Roads

The primary access route to the region is by paved Highway 167, running between St-Félicien and Chibougamau. At kilometre 200, Highway 167 is intersected by forestry road 210 (gravel road) that leads toward the project site. This intersection has been defined as “kilometre 0”. This 29 km road will be used to transport the magnetite concentrate produced at the BlackRock concentrator. Except for the last 2 km, the road will undergo upgrading and repair during the construction phase to allow for general traffic and later be fully built to allow for the concentrate transport as well as regular deliveries and transport to the site. The remaining 2 km close to the plant site entrance will be rebuilt. A future security gate will also be put in place along this road before entering the plant site to ensure access is limited to site personnel.

Within the site itself, a network of service and construction roads connect the processing plant, tailings pond, mine pit, waste rock stockpile, mine garage, explosives plant, powder warehouse and cap warehouse.

18.1.4.2. Ore Conveyor Maintenance Road

This service road allows access to the conveyor between the crusher and the stockpile for maintenance as well as the crusher pad lower level. The road was extended in its width at regular intervals for the construction of the conveyor supports and foundations.

18.1.4.3. Service Road to Monitoring Pond

An existing road reaching the tailings monitoring pond will be upgraded and extended close to the substation pad in order to bring power through an overhead electrical line at the northern end of the tailings disposal site. This same road will continue alongside the reclaim water pipe, which returns make-up water to the concentrator.

18.1.4.4. Service Road for Tailings Pipe

Except for a small portion at the concentrator area, the tailings pipe will be above-ground and will sit on a granular pad next to a service road between the concentrator and the tailings disposal dumping point. This road follows the same alignment than the Ore conveyor maintenance road from the stockpile to the lower crusher area. Between the crusher and the dumping point, it also acts as a physical barrier preventing run-off from entering the tailings disposal area next to Denis Lake.

18.1.4.5. Mine Haul Roads

The mine haul road from the Southwest Pit exit will be a 22 m wide double lane road. The road will lead from the mine exit to the jaw crusher along the way and will not have more than a 10% grade at any point.

To reach the mine garage for maintenance, trucks can use a 14 m wide single-lane road intersecting to the road from the Southwest pit to the crusher pad.

Mine roads will be constructed with a minimum of 2.15 m thick crushed waste rock having different layers with a gradual gradation to avoid particle segregation. Where more backfill is required due to topography, run of mine waste rock or competent excavated overburden is used.

18.1.5. Jaw Crusher

Mine trucks will use the mine haul road to bring run of mine ore to the jaw crusher. Mining trucks will dump the ore at a single drop point located on the top of a supporting reinforced earth wall 23 m high. Crushing building measures approximately 14 m by 29 m and houses the crushing unit, auxiliary equipment as well as a feed hopper which can contain two truckloads worth of run of mine material.

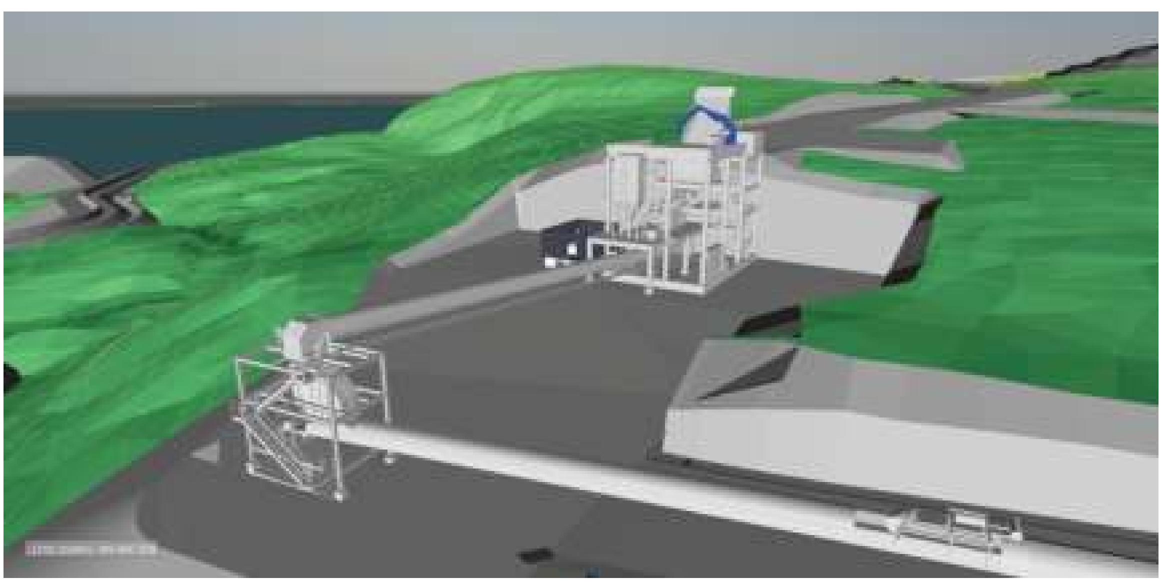


Figure 18-3: Jaw crusher building extracted from 3D Model

18.1.6. Crushed Ore Storage

Crushed ore from the jaw crusher will be conveyed to the crushed ore covered stockpile. The stockpile will be located approximately 230 m from the concentrator via a maintenance access road. The ore is reclaimed via underground belt feeders and a reclaim conveyor, which is supported by a tunnel allowing for maintenance workers to access the underground equipment.

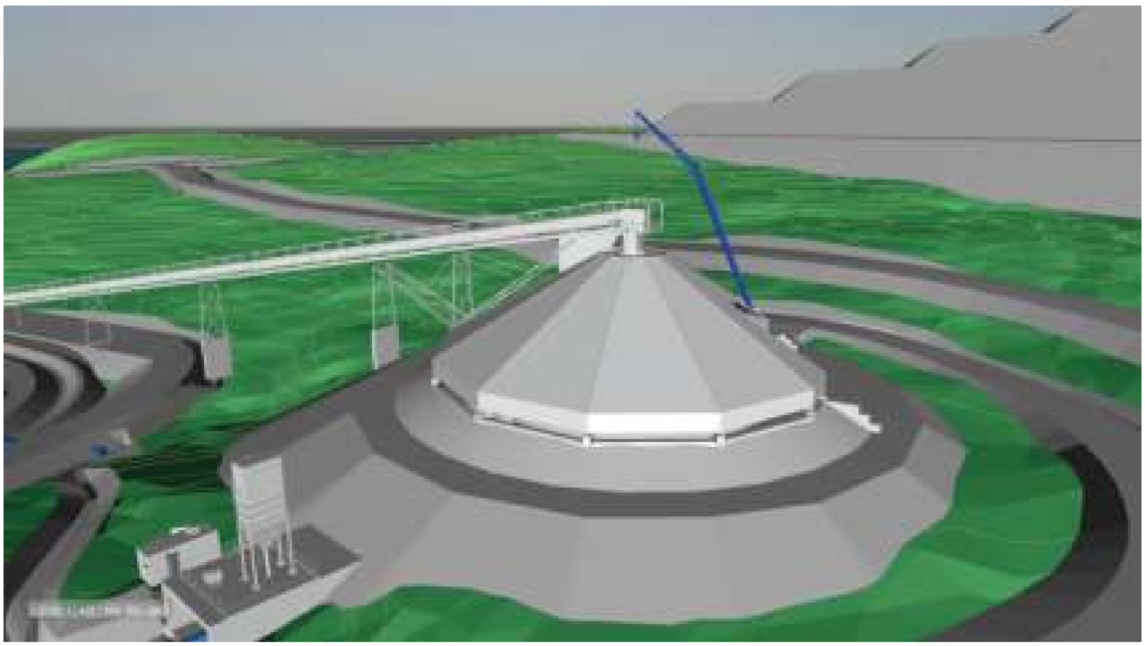


Figure 18-4: Stockpile extracted from 3D Model

18.1.7. Concentrator Pad

The substation, concentrator and mine garage have all been allocated areas on different levelled pads. The substation is located at the highest point of the three areas at an elevation of 480 m and is situated to the south-west of the concentrator area. The concentrator building, loadout facility, warehouse and office spaces are the furthest north and sit lower than the substation and the mine garage plateau, which is further east. The concentrator and mine garage plateaus are at elevations of 473 m and 475 m respectively.

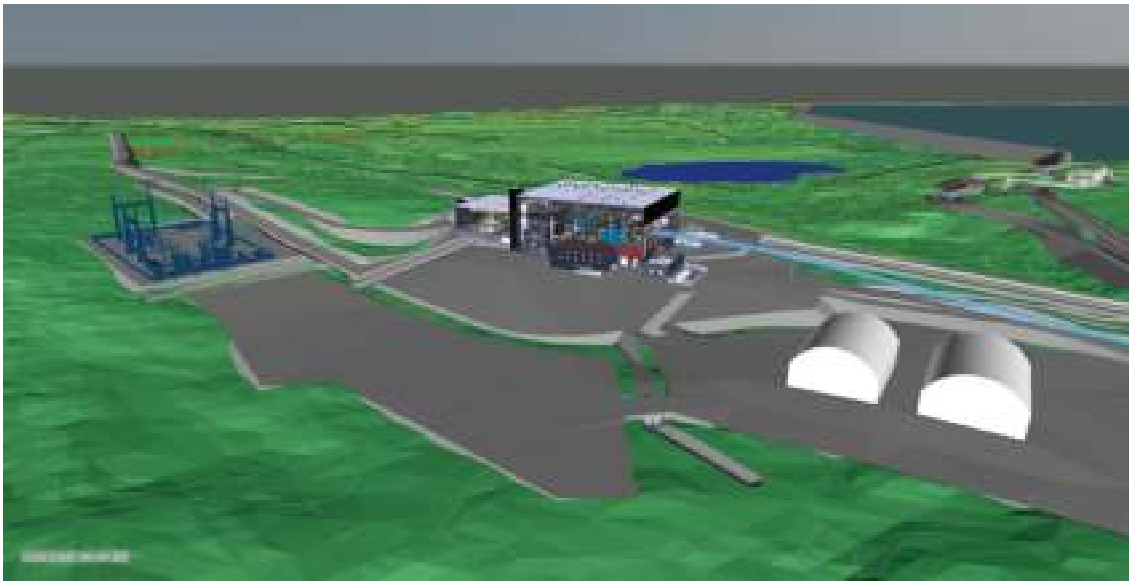


Figure 18-5: Pad levels extracted from 3D Model

The concentrator building measures 72 m x 59 m and is 27.5 m high, which includes the magnetite processing equipment as well as the services building. The concentrator houses the grinding, size classification, magnetic separation, pumping and filtration along with a mechanical workshop and laydown areas. The thickener, process water tank and fresh/emergency water tank are all located outside of the concentrator building.

The concentrate loadout facility is located directly next to the concentrator building. The building measures 38 m by 41 m with a height of 15 m. If the concentrate haul trucks are loaded directly with the conveyor, they will drive to the conveyor discharge point. The discharge point is protected by a shelter connected to the loadout facility. If the concentrate haul trucks are being loaded from the emergency stockpiles in the loadout facility, they will drive around the concentrator upon arrival and enter the loadout by the entrance next to the tailings thickener.

Attached to the concentrate loadout is the warehouse, which measures 18.6 m by 38 m and is 19.8 m high and accounts for two floors. The first floor will house the plant spares and equipment and the second floor will house wet and dry lockers, janitorial storage as well as the cafeteria.

The service building is located within the concentrator and consists of four floors. Within an area of 14 m by 17.5 m by 24.5 m high is found the HVAC rooms and electrical rooms. In an area of 7 m by 17.5 m by 24.5 m high are the offices, the metallurgical laboratory and the medical station.

Parking for employee vehicles and other service vehicles will be located in close proximity and west of the processing plant.

Figure 18-6: Concentrator extracted from 3D Model

18.1.8. Mine Services Facilites (Mine Garage and Truck Wash)

The mine garage and truck wash shop will both be constructed using a Megadome technology structure and corner cast base. The height is sufficient for maintenance of the largest mine truck, i.e., one equivalent to the CAT 777G, 32 ft at the roof peak.

18.1.9. Accommodation Complex

Due to the proximity of the site to the town of Chibougamau, no provisions have been made for the construction of a permanent nor a temporary camp. The project places an emphasis on utilizing local manpower. For non-local workers, lodging will be provided at a hotel complex in Chibougamau, which has enough capacity to house the peak quantity of workers during construction.

At site, services will include a potable water network, fire protection loop, and a sanitary sewage collection network complete with a treatment facility. Trailers will be used for temporary management complexes. Heating and electricity for the facilities will be provided by propane and temporary diesel power plants, respectively.

18.1.10. Fuel Storage and Dispensing

Storage tanks of #2 light fuel will be provided and located adjacent to the concentrator and at the mine garage. The mine equipment fuel will be stored in a tank farm consisting of five tanks at the mine garage, and one double-walled gas dispenser will be located on the mine garage pad. Four additional tanks will be placed at the concentrator and five propane tanks will also be located at the concentrator and will service the boilers. The #2 light fuel will be delivered by road tankers.

18.1.11. Explosive Plant and Storage

A platform for an explosive storage shed and maintenance facilities will be built at a safe distance (minimum 1 km from any building), northwest of the process plant area. These buildings will be supplied and constructed by the explosive's supplier.

18.1.12. Electrical, Communication and Automation

18.1.12.1. Tie-Point Switching Station, Power Line, Main Substation, and Site Electrical Distribution

The total power demand for the project, including a 10% tolerance factor for equipment design, is estimated at 21 MW based on the estimated connected load, running load and running power. Table 18-1 shows the power demand breakdown by sector. The power demand was calculated using an average efficiency factor, load factor and diversity factor. A new 161 kV power line will be built by Hydro-Québec and will connect to the existing 161 kV Hydro-Québec line #1627 (Obalski/Otabogamau), servicing Chibougamau.

Table 18-1: Process plant power demand by area

Area	Estimated Power Demand (MW)
Primary Crushing	0.7
Stockpile	0.3
Concentrator Plant	14.3
Tailings Pond	0.4
Other Services	3.2
Network Losses (2%) and Contingency (10%)	2.3
Total	**21.2**

The main 161 - 25 kV substation will be located near the concentrator building. Two main 161-25 kV, 24/32 MVA transformers will be used for a combined firm power of 24 MVA. The main substation (161-25 kV) supplies the power level required for site-wide voltage distribution. A 25 kV Gas Insulated Switchgear (GIS), complete with the usual electrical protection devices, will be included in the main electrical room near the substation.

At the concentrator, there will be several Induction Motors (SCIM) ranging from 0.5 to 900 HP, excluding the mills, which will require a 4.16 kV and 600 V levels. Therefore, two 25-4.16 kV power transformers and one 5 kV switchgear are required for the 4.16 kV loads and two 25000-600 V unit substations are included in the design. For loads at the crushing and stockpile facilities, a 4.16 kV feeder is planned for each facility. Both facilities will have a prefabricated electrical room housing a 4.16 kV and a 600 V switchgear, with the 600 V generated locally.

The open pit mine equipment will operate using diesel fuel and will not require an electrical feed from the main plant.

The electrical distribution network dedicated to the site infrastructure will consist of two 25 kV overhead lines comprised of ACSR (Aluminum Conductors Steel Reinforced) type conductors. This network will supply the following facilities:

- Tailings and Water Management;
- Polishing Pond (reclaim water);
- Telecom Tower;
- Explosives Storage;
- Effluent Treatment Plant;
- Potable Water Treatment.

Five emergency generators are planned for the following sectors:

- Concentrator (1750 kW @ 4.16 kV + 1200 kW @ 600 V);
- Crusher (800 kW @ 600 V);
- Stockpile (800 kW @ 600 V);
- Mine (1200 kW @ 600 V).

The 600 V generators will be recovered from the temporary installation during the construction.

18.1.12.2. Communication

18.1.1.1.1 Systems

A mobile radio system will be supplied for the construction and operation phase, providing coverage of the mine site, the construction area, and possibly the future BlackRock office located in Chibougamau. This system is provided by a telecom tower and a telecommunication shelter housing all of the communication equipment. An IP Phone System will be provided from the beginning of the construction phase.

Telecommunication systems will include the infrastructure equipment to provide networking services for the following applications:

- Corporate IT (Internet Access, File and Printer Sharing, PC Workstations, IP Telephony);
- Process Control;
- Instrumentation and Control of ancillary facilities (power substations, fuel farm, etc.);
- Security (access control and video surveillance);
- Fire Protection System;
- Mobile radio system (VHF/UHF);
- Satellite TV Service in camp bedrooms and public areas.

Mobile Radio and Corporate IT services will be made available in the engineering office and contractor field offices (contractor trailers) right from the start, at the beginning of construction. As new field offices arrive on site, they will be quickly hooked up to the mine campus network with prepackaged subscriber kits and wireless links, providing phones, networking and Internet access services. This quick deployment approach will keep communications flowing, avoiding unnecessary delays.

18.1.1.1.2 Services

The main telecom site is currently connected to an Internet Service Provider (ISP) via a microwave link to Mont-Bourbeau in Chibougamau, where a fibre-optic Internet access link is available. The microwave link equipment is supplied and maintained by the ISP.

The IP Phone System will be connected to an Internet Telephone Service Provider (ITSP) to supply VoIP (Voice over Internet Protocol).

TV services will be received via satellite and distributed via a mix of fibre optic and coaxial cabling and equipment.

18.1.1.1.3 Distribution

During the construction phase, all communication services, such as Internet and phone, will be distributed via Wi-Fi, WiMAX and Microwave radio links to reach all buildings, including the construction camp and contractor field office trailers.

A fibre-optic ring topology link will be extended from the main telecom site to the process plant, the administrative offices, the mine garage and all other facilities where it is required, following the 25 kV power distribution lines.

18.1.1.1.4 Corporate Network

Separate fibre-optic Ethernet ring topology backbones shall be deployed using separate fibres in shared cables around site for: A) Corporate IT, B) Process Control, C) Fire Protection System, and D) Security.

Core network equipment and servers will be hosted in dedicated server rooms located in the engineering office, telecom shelters, various office server rooms, and in the process control room.

18.1.12.3. Automation

18.1.1.1.5 Process Network

An automation Ethernet backbone at 100 Mbps in a ring-type topology will link all the main automation equipment such as SCADA, local HMI, PCS (processor only), and main electrical substation.

18.1.1.1.6 Process Control System

Each sector will have its own PCS connected to remote I/Os. Six main processors shall control the following sectors: Crushing, Grinding, Concentrate, Water Control and Utility, Tailing and Plant Emergency Power Control.

18.1.1.1.7 Supervisory Control and Data Acquisition System (SCADA)

The SCADA will be based on client/server technology:

- Two SCADA servers (for redundancy;
- Four main SCADA operator stations;
- One located in the Crusher control room;
- Three located in the Concentrator control room;
- Six local operation clients throughout the plant.

18.1.12.4. Process Analog Instruments

Process Analog Instruments will, whenever possible, support HART protocol and they will be wired to analog inputs/outputs of the Process Controller by means of traditional 4-20 mA loops.

18.1.13. Tailings Management

The fine and coarse tailings are combined at the concentrator and are pumped to the tailings park to be deposited. These combined tailings will from this point forward be referred to as "the tailings".

The design of the tailings park as well as the construction plan assumed an estimated 55.1 Mm^3 of tailings to be produced. The tailings pond consists of four distinct cells which will be sequentially constructed and filled with tailings. Thus, rehabilitation of the tailings park may begin as soon as a cell has been filled with tailings. The scheme of operation proposes the transfer of free water from the tailings pond to a polishing pond to allow for sedimentation of fine particles and other minerals. Water will then be transferred from the polishing pond to the plant process water tank, to be used as make-up water in the process. During the operation of the plant, excess water in the polishing pond will be measured and treated, as required, prior to being sent to a monitoring pond. If the water in the monitoring pond does not meet environmental standards, it will be returned to the polishing pond for further treatment. From the monitoring pond, once the water meets environmental standards, it will be released to the environment.

Tailings pond location and construction scenarios were based on topographic information of the site. The method of approach was that of raised embankments which are constructed in phases over the lifetime of the mine plan. Raised embankments begin with a low initial dyke with more height added in a subsequent phase to the embankment on the downstream side (downstream construction method as seen in Figure 18-7) as the volume of tailings increases in the impoundment. The tailings pond was placed on the west side of the proposed open pit mining zone and sized for 55.1 Mm^3 of tailings over a 43-year period based on a total of 95 Mt of tailings produced and tailings dry deposition density of 1.78 kg/m^3. The final tailings pond has a freeboard of 1.0 m between the tailings final surface and the spillway.

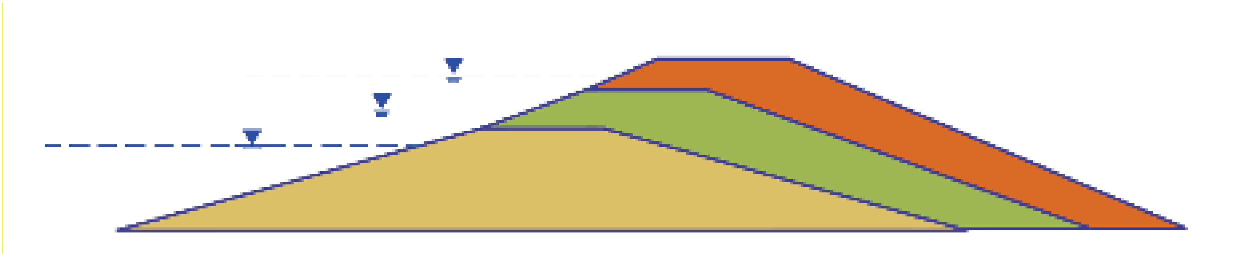


Figure 18-7: Downstream Construction Method used for Tailings Embankments

The tailings retention dams were designed with selected rock materials and an impervious core (see Figure 18-8) including the Polishing Pond and the Monitoring pond. Their crest width is typically 10 m, with slopes of 2.0:1.0 horizontal/vertical on both upstream and downstream sides. The crest of all impermeable dykes will be constructed at an elevation of 2 m above the tailings/water operating level to provide 1 m of freeboard between the operating level and the elevation of the emergency spillway and 1 m height for the emergency spillway above the freeboard.

In the interior of the tailings pond, the cells are separated with permeable rock dykes rising to the elevation of the tailings (i.e., 2 m below the crest elevation of the tailings retention dams (see Figure 18-9).

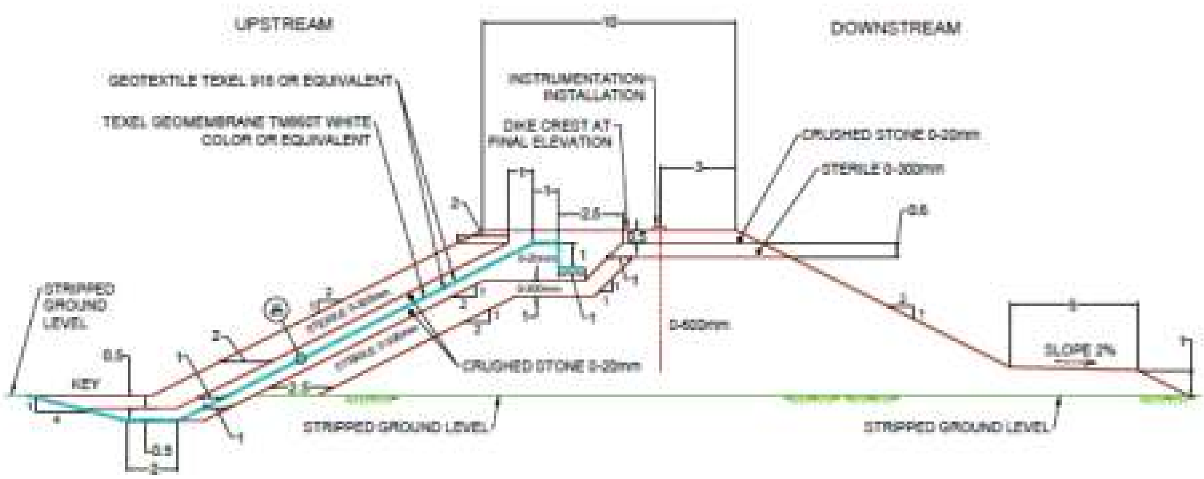


Figure 18-8: Tailings Park Impermeable Dike Typical Section

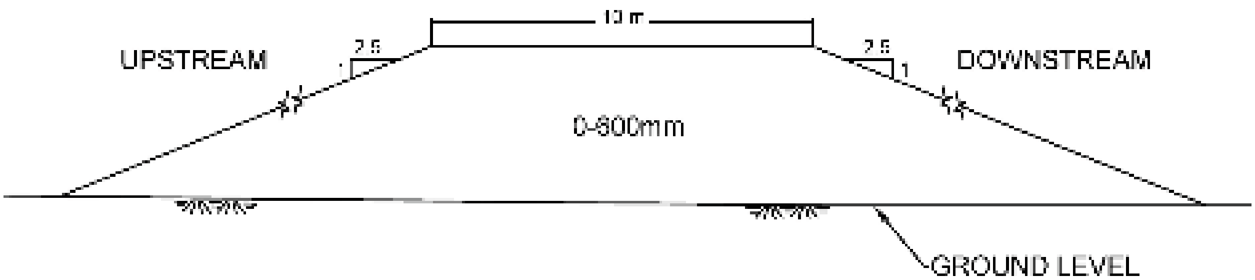


Figure 18-9: Tailings pond permeable dike typical section

18.1.13.1. Tailings Pond Construction - Pre-Operation

The tailings pond and polishing pond will need to be partially constructed prior to plant start-ups. It will be necessary to accumulate water inside these ponds to ensure that the reclaim water recirculation loop has sufficient time to reach steady state. Thus, a construction plan was put forward for the pre-operation period as well as sustaining construction during operation to match the rising level of tailings in the tailings pond.

The start-up tailings park in the pre-operation period will include the partial construction of the dykes surrounding Cell no. 1, the construction of the dykes to the final elevation for the polishing and water monitoring ponds as well as an access road (BRM-008), built with waste rock, on the east side of the tailings park to be used for the construction of the dykes.

Initially, partial impermeable dykes surrounding Cell no. 1 will be built at an elevation of 421 m (419 m for the permeable dyke separating the cells inside the park), the impermeable dykes surrounding the polishing basin will be built to their final elevation of 422 m as well as the impermeable monitoring basin dyke to its final elevation of 418 m.

The west-east dyke separating the tailings pond and the polishing pond will initially be constructed at an elevation of 422 m with a spillway at the elevation 421.3 m to transfer the tailings water to the polishing pond in the event of an extreme flood. The lower part of the south section of this dyke (dyke F – G) will also be constructed so as to avoid working in wet conditions when the dyke will be raised. The layout of the pre-operation construction is shown in Figure 18-10.

The capacity of Cell no. 1 of the start-up tailings ponds at elevation 419 m (impermeable dyke elevation to 421 m and permeable dyke elevation to 419 m) is 2.6 Mm^3 of tailings which is estimated to be a sufficient volume for the initial two years of operation.

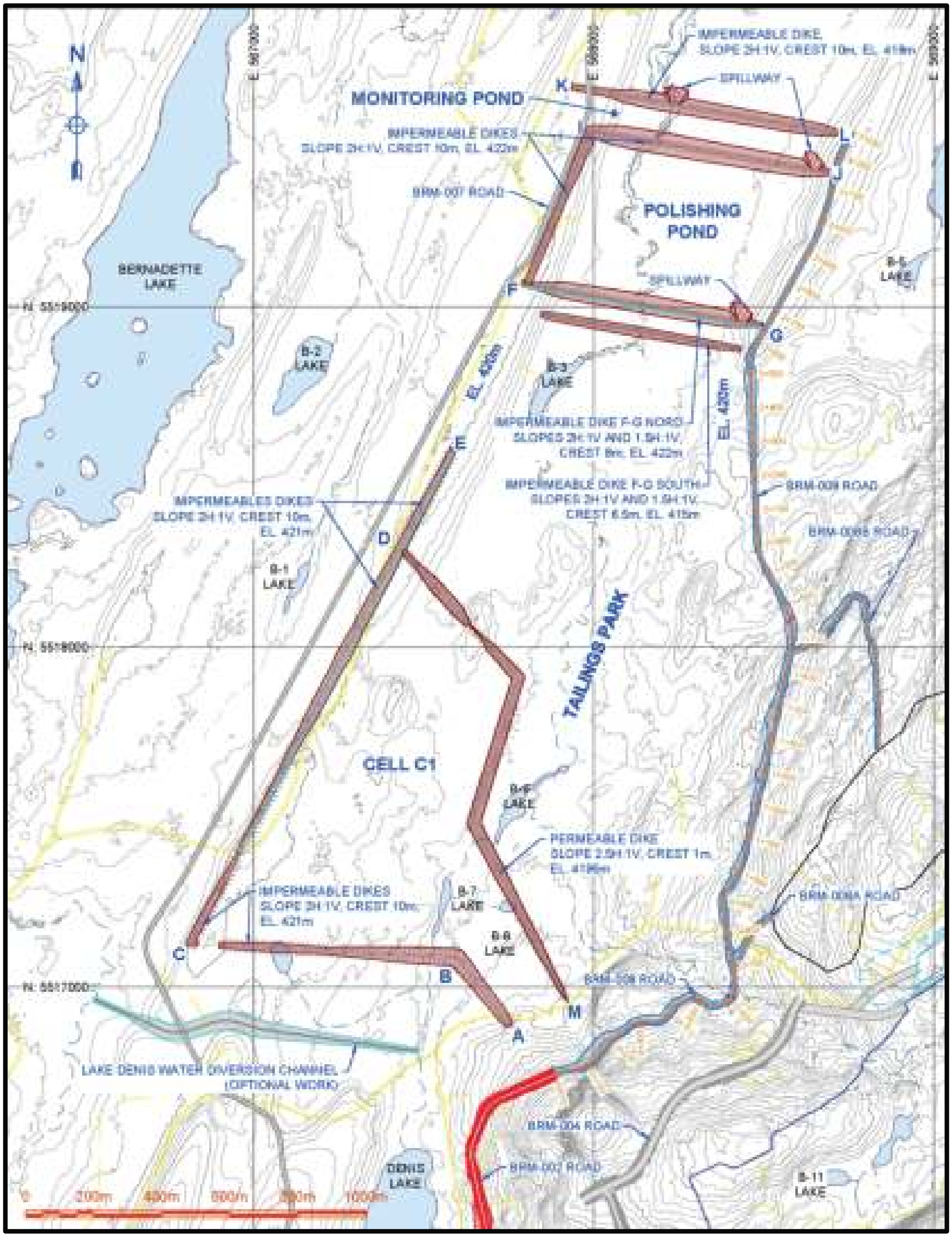


Figure 18-10: Starter dykes for the tailings, polishing and monitoring ponds as constructed at the end of the pre-operation phase

18.1.13.2. Polishing Pond Construction - Pre-Operation

The polishing pond will be located on the north side of the tailings pond (Figure 18-10). Two dykes, constructed with selected rock materials and an impermeable membrane core, are necessary to retain the precipitation and the free water transferred from the tailings pond. This water will still contain very fine particles which will settle in the polishing pond. The polishing pond is designed for a capacity of 2.6 Mm^3 to an elevation of 420 m. The polishing pond dykes will be constructed to their maximum height of 422 m in the pre-operation period, as it will be necessary to collect and retain water for the plant start-up.

18.1.13.3. Monitoring Pond Construction - Pre-Operation

The monitoring pond will be located on the north side of the polishing pond (Figure 18-10). One dyke, constructed with selected rock materials and an impermeable membrane core, is necessary to retain the precipitation and the excess water. During operation, run-off collected will be placed and monitored in the pond, where the water quality will be measured. If the water quality does not meet environmental standards, it will be returned to the polishing pond for further treatment otherwise it will be released into nature. It is designed for a capacity of 0.47 Mm^3 and its capacity may be increased, if required. The monitoring pond dyke will be constructed to its maximum height of 418 m in the pre-operation period.

18.1.13.4. Tailings Pond Construction during Operation

18.1.1.1.8 Construction between years of operation 1 and 10, completion of cell no.1

Between the first and tenth years of operation, the confinement dykes of Cell no. 1 (A - B - C, C - D - E and D – M) shall be raised to their final elevation providing an additional capacity of 12.6 Mm^3 of tailings for Cell no.1 which will reach a total capacity of 15.2 Mm^3 at the end of the eleventh year (total capacity of Cell no. 1 is eleven years of storage).

During the same period of time, the west dyke E - F will be built along its entire length to an elevation of approximately 429 m, which will represent approximately half the final height of this dyke, where its crest will be at an elevation higher than the spillway to the polishing pond (dyke F – G, see Figure 18-10 and Figure 18-10). At the same time, the access road leading to the tailings on the east side of the tailings park (road BRM-008) shall be moved over approximately 320 m at a higher elevation (422 m which is the elevation of the crest of the dyke between the tailings and the polishing ponds (dyke F – G) and 0.7 m above the spillway of the same dyke (see Figure 18-10); at this point, the spillway will be operational in case of an emergency.

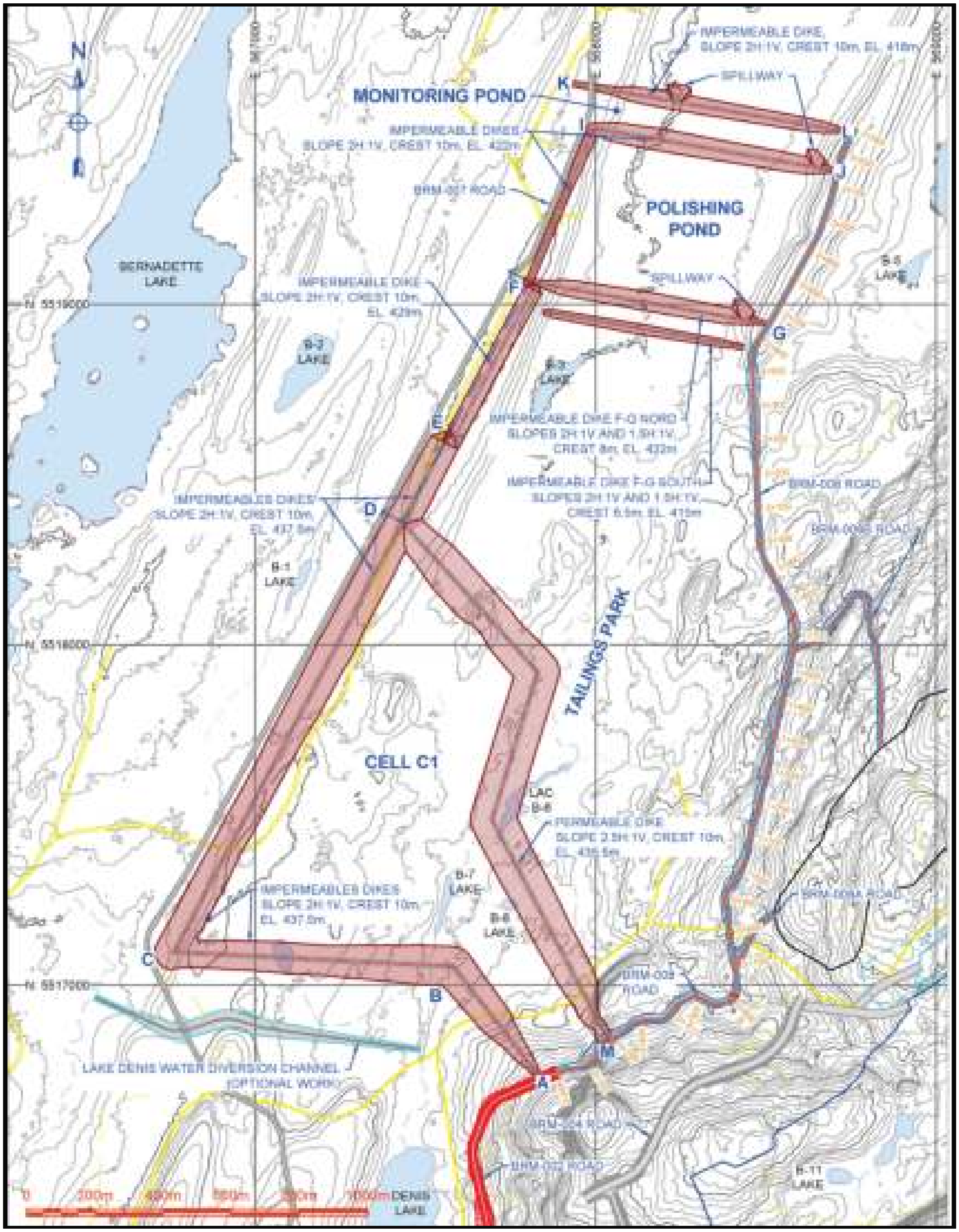


Figure 18-11: Tailings park at completion of Cell no. 1 (Year 10)

18.1.1.1.9 Construction between years of operation 11 and 20, completion of Cell no.2

Between the tenth and nineteenth years of operation, the minimum construction required is the construction of the permeable dyke up to final elevation 435.5 m (point N on the west side hill of the tailings park) to enclose Cell no. 2 with an ultimate capacity (at elevation 435.5 m) of approximately nine additional years of operation or 11.2 Mm^3 at the end of the twentieth year (total capacity of the tailings pond is 26.4 Mm^3 of tailings at this point). Access road BRM-008 will be rerouted to above the final tailings elevation of 435.5 m along the east side of the tailings park (see Figure 18-10).

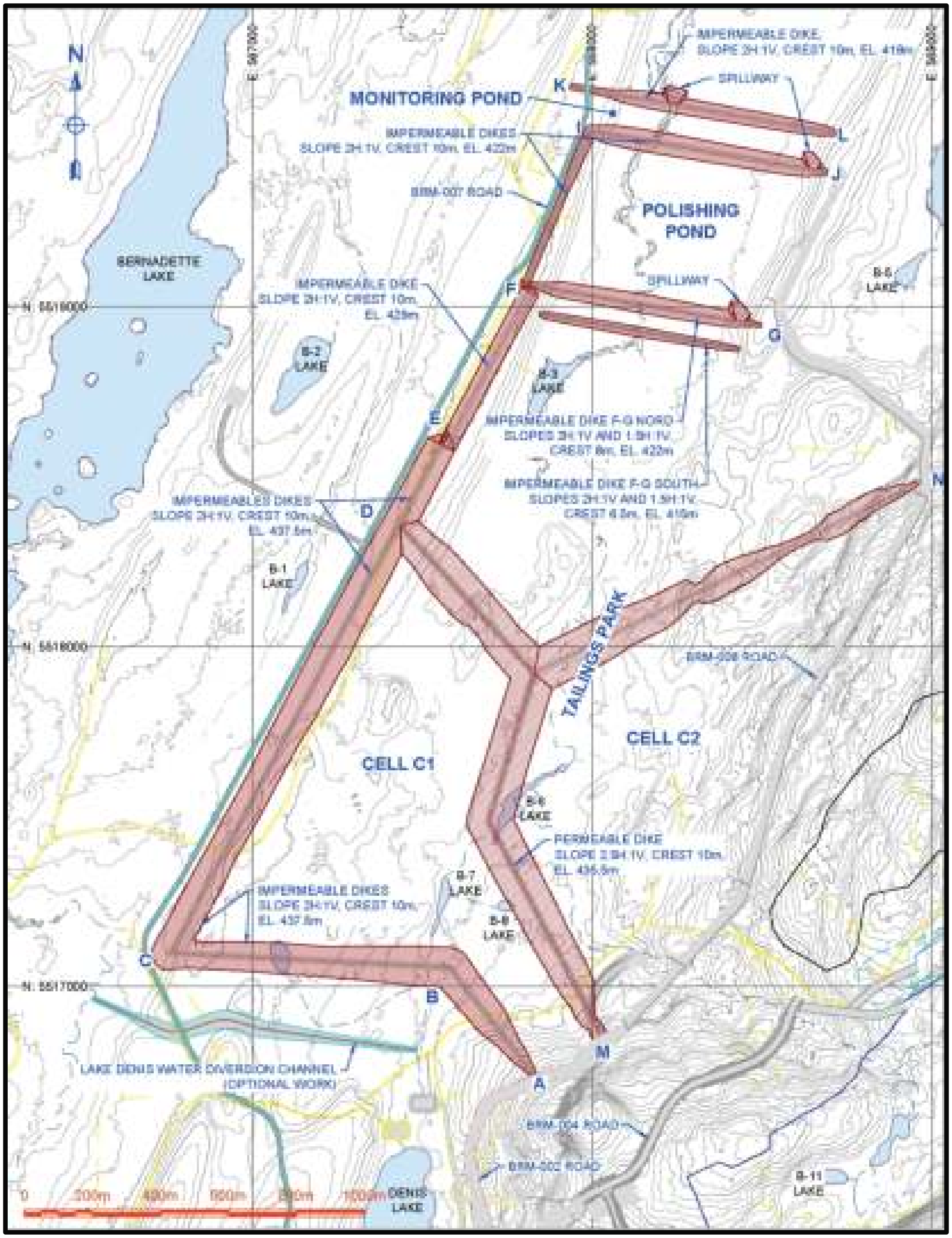


Figure 18-12: Tailings park at completion of Cell no. 2 (Year 20)

18.1.1.1.10 Construction between years of operation 21 and 27, completion of Cell no.3

Between the twentieth and twenty-seventh years of operation, the minimum construction required is the impermeable dykes of Cell no. 3 to final elevation 437.5 m (Figure 18-10, dykes C – O, O – P and P – D, in one or two stages including a spillway on the crest of the dyke between cells no.3 and no. 1 (dyke C – D) for water to transfer towards Cell no. 1 and eventually to the polishing pond. The ultimate capacity of Cell no. 3 (at elevation 435.5 m) is estimated to be 10.8 Mm^3 of tailings storage equivalent to an additional 8years of operation. Total capacity of the tailings pond is 37.2 Mm^3 of tailings after 27 years of operation.

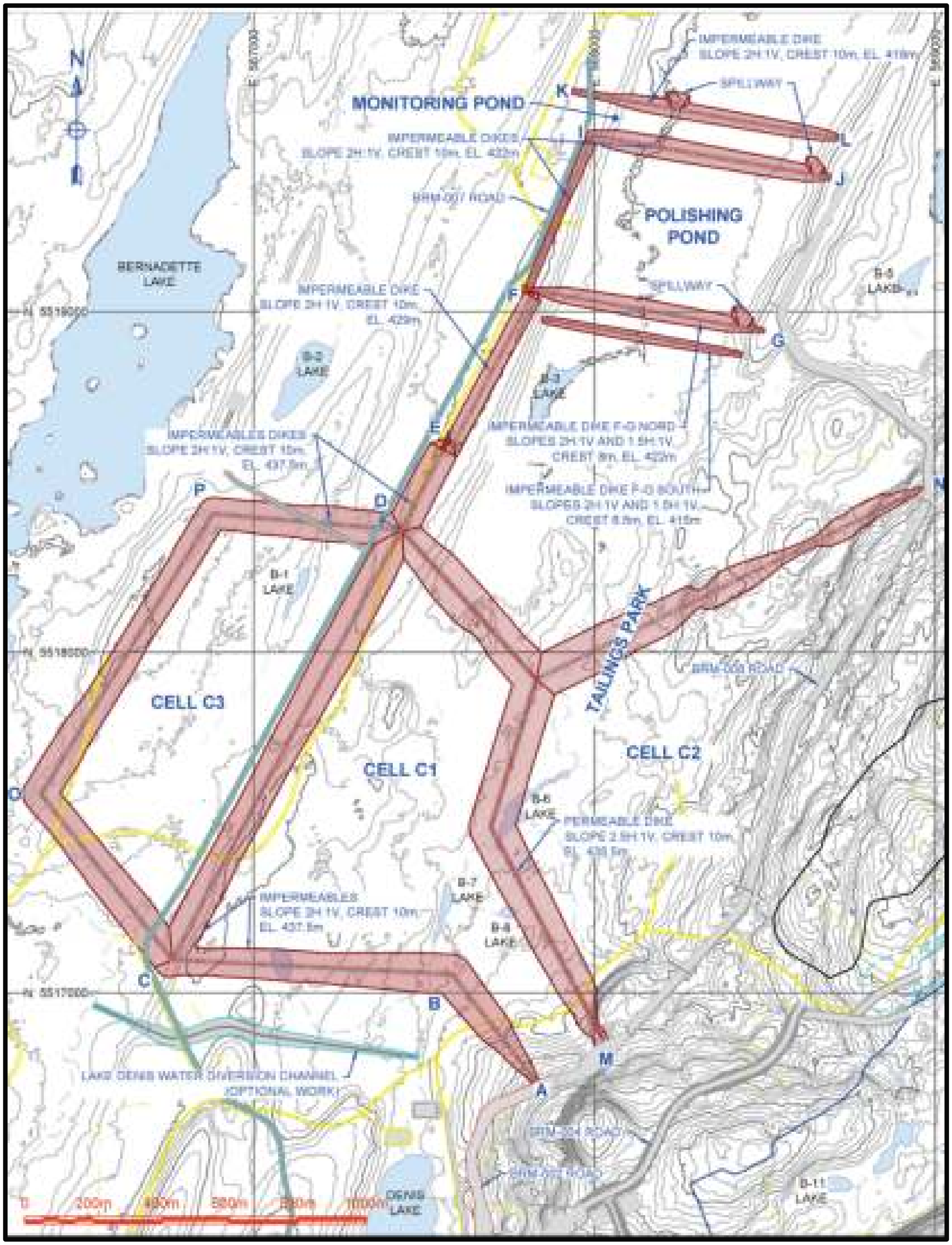


Figure 18-13: Tailings park at completion of Cell no. 3 (Year 27)

18.1.1.1.11 Construction between years of operation 28 and 42, completion of Cell no. 4

The construction of the tailings park will be completed with the construction of the impermeable tailings dykes on the west side of the park and between the tailings and the polishing ponds to the final elevation of 437.5 m thus completing the construction of Cell no. 4 (dykes D - F and F - G – H). A new final spillway shall be constructed on the dyke between the tailings and the polishing ponds (dyke F - G – H, Figure 18-10). The ultimate capacity of Cell 4 (at elevation 435.5 m) is estimated to be 17.9 Mm^3 equivalent to an additional 15 years of operation – end of operations (total capacity of the tailings park is 55.1 Mm^3 of tailings storage at the end of operations).

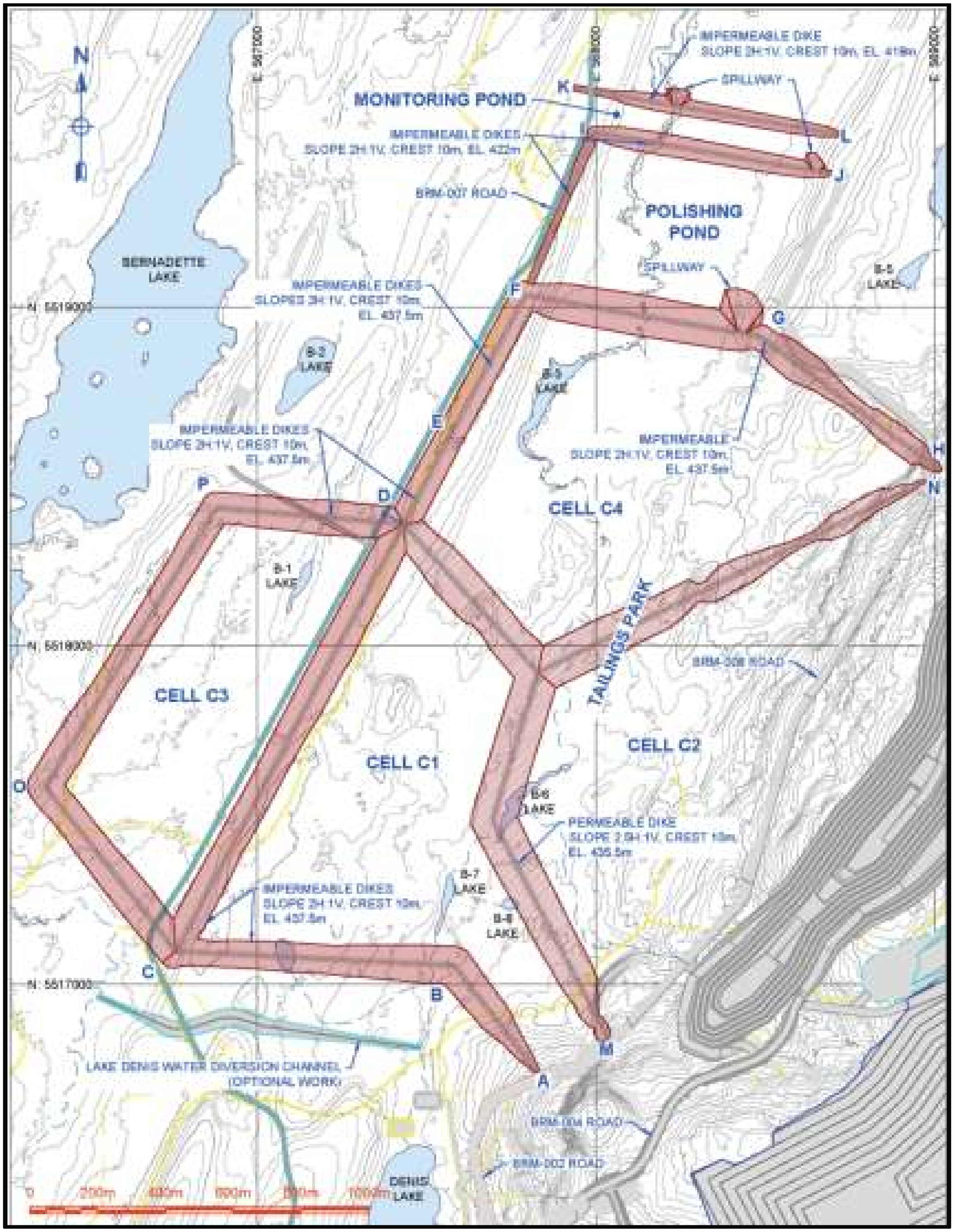


Figure 18-14: Tailings Park at completion of Cell no. 4 (Year 42)

18.1.1.1.12 Tailings Park Construction Quantities of Materials

As the level of tailings rises in the tailings pond, the level of the tailings dykes must be raised as well. A construction plan has been put forward wherein the year of construction and the quantity of materials required is described. The construction of the dykes must be completed by the year indicated in the table for the tailings pond to hold the estimated amount of tailings produced with a 2 m freeboard at all times. This plan, with the associated construction materials required, may be found in Table 18-2 and Table 18-3. Note that the estimated material quantities for the operation period do not include the pre-operation quantities. The tables do not include estimated material quantities for the tailings access road BRM-008.

Table 18-2: Pre-operation construction plan for tailings park dykes

Polishing Pond / Treatment Pond dyke elevations (m)	Impermeable Tailings dyke elevation (m)	Cumulative Tailings Volume (Mm³)	Rockfill 0-600 mm m³	Rockfill 0-300 mm m³	Rockfill 200-500 mm m³	Crushed stone 0-20 mm m³	LLDPE Membrane m³	Geotextile m³
422 / 418	421.0	---	839,690	185,510	600	142,690	164,900	352,200

Table 18-3: Yearly construction plan for tailings park dykes during operation

Construction period Year to year	Impermeable Tailings dykes elevation (m)	Cumulative Tailings Volume (M m³)	Rockfill 0-600 mm m³	Rockfill 0-300 mm m³	Rockfill 200-500 mm m³	Crushed stone 0-20 mm m³	LLDPE Membrane m³	Geotextile m³
1 – 10, Cell no. 1	437.5	15,2	5,068,670	257,640	0	132,750	148,140	332,830
11 – 20, Cell no.2	437.5	26,4	871,200	0	0	0	0	0
21 – 27, Cell no. 3	437.5	37.2	2,158,490	230,660	0	131,830	140,630	306,830
28 – 42, Cell no.4	437.5	55.1	1,477,860	126,760	400	66,310	61,180	140,710

(Estimated material quantities for the operation period do not include pre-operation construction quantities)

18.1.14. Site Water Management

The logistics of the water balance are defined by the concentrator mass balance and environmental/topographic data for the surrounding area provided by Journeaux Assoc.

The principal areas of interest in the tailings and water management strategy are:

- Concentrator (for ore processing);
- Tailings pond (for deposition of the coarse and fine tailings);
- Polishing pond (further decantation of water reclaimed from tailings and used as make-up water to the process);
- Monitoring Pond (to monitor effluents before release to the environment);
- Open pit mine.

18.1.14.1. Process Water Balance and Description

In order to minimize reclaim water requirements, the BlackRock water balance has been designed to maximize water recycling within the process, with shortfalls being made-up by water collected from precipitation and run-off in the surrounding watershed and from water reclaimed from the open pit mine.

The average total process make-up water and fresh water needs were determined to be approximately 317 m^3/h.

Major process water additions consist of the following process streams:

- SAG Mill feed;
- Ball Mill discharges;
- Scalping and Multi-slope screens spray bars;
- Cobbing and Cleaning/Recleaning magnetic separation (re-pulping and wash);
- Reagent dilution;
- General dilution, cleaning and make-up water.

Water enters the circuit in the form of moisture in the ROM, estimated at 4% and from reagents, and leaves the circuit in the tailings and final concentrates. All make-up and fresh water is taken from the polishing pond. A portion of the reclaim water is filtered and is used as a fresh water source. This water is used for reagent preparation and for gland seal water.

The majority of the water used in the concentrator is recycled within the process. The tailings thickener decants water from the tailings and returns it to the process water tank. These recycle overflow streams represent approximately 90% of the concentrator water needs. The remaining water consists of the make-up taken from the polishing pond.

The reclaim water is needed to compensate for water losses in the form of water lost in the final concentrates and the water pumped with the fine and coarse tailings to the tailings pond.

18.1.14.2. Site Water Management

In order to start production, water must first be collected in the tailings pond to feed the concentrator. Furthermore, water sent from the process plant to the coarse tailings stockpile and fine tailings impoundment will have a certain residence time before it can be returned to the process. Thus, there is a period of instability in the reclaim water cycle before steady-state conditions are achieved. Due to this phenomenon, excess water must be collected during the pre-production phase to ensure adequate pond levels and water accessibility.

Pre-Production

In the pre-production period, water will be collected in the tailings and polishing ponds once they have been constructed. The water shed that contributes to the water collection in the tailings and polishing ponds is denoted by the green highlighted area in Figure 18-15. In order to collect enough water for production, and for ease of construction, the polishing pond will be constructed to final elevation to hold 2.6 Mm^3 of water, while the tailings pond including cell no.1 could temporarily hold approximately 6 Mm^3 to elevation 419 m or approximately 5.3 Mm^3 excluding cell no.1 (in the zone between cell no. 1 and the polishing pond.

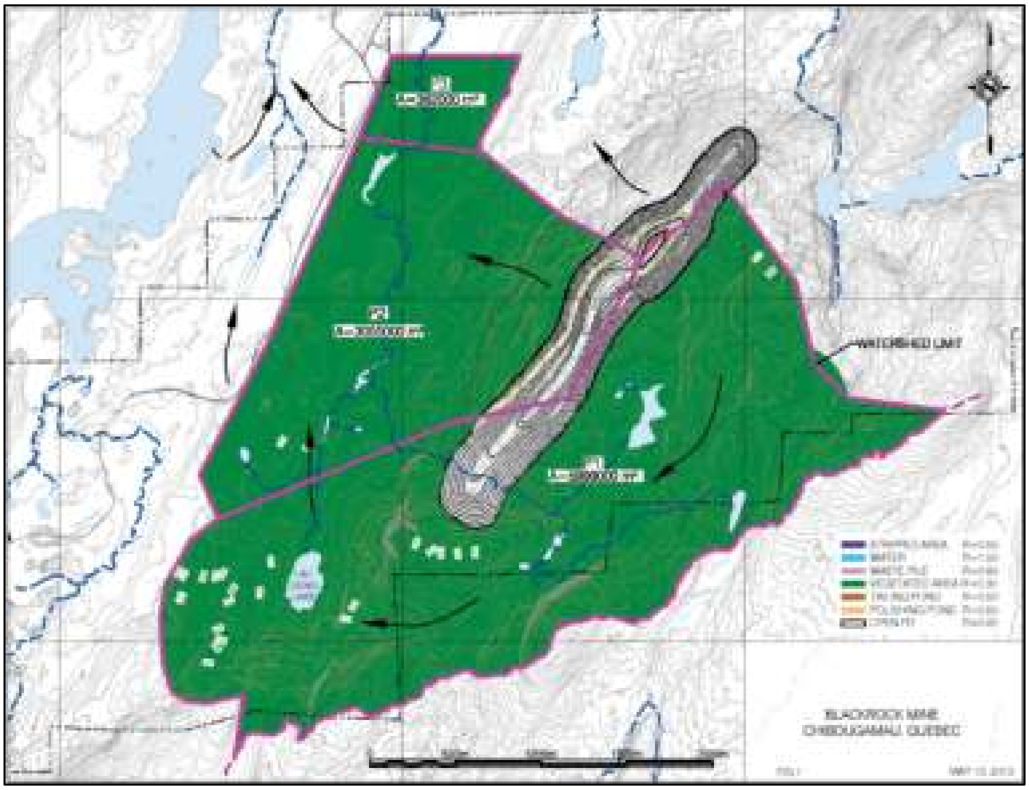


Figure 18-15: Pre-production water sheds (Journeaux Internal note 15132013)

Production

During production, tailings will be pumped from the concentrator to the tailings pond. Ditching is planned to direct the fine tailings into a steady stream that will reach the northern region of the pond. For water management calculations and pond volumes; the following hypotheses were taken into consideration:

- For start-up water calculation requirements, dry-year precipitation values were used;
- For yearly pond water levels and simulations average-year precipitation values were used;
- Pumping from the tailings pond to the polishing pond occurs from May to October exclusively;
- Effluent monitoring and discharge to occur from May to October exclusively;
- The minimum working volume of the tailings pond was taken to be 1,000,000 m^3;
- The minimum working volume of the polishing pond was taken to be 350,000 m^3.

As the mine is developed, rain or underground water sources will accumulate in the mine. This mine water will be pumped directly to the tailings pond.

Water falling on the pits will be collected via run-off and into the tailings pond. However, as the pits are mined, water will be collected within the mine itself. This water will be pumped directly to the tailings pond throughout the year.

18.1.15. Fire Protection

Fire protection for the processing plant and service buildings will be provided by a fire protection loop. The system comprises strategically placed hydrants and a plant header, all fed by the fire protection tank. The tank will be filled once with water retrieved from the fresh water (filtered reclaim water) and will be on standby to feed the fire protection loop should the need arise.

18.1.16. Train Loadout

The VTM concentrate will be transported to a train loadout facility around 25 km from the mine site. This facility has been designed by CIMA. BlackRock is planning to include the construction of this facility in a global transport agreement. Therefore, this loadout facility will be leased from a contractor who will also provide transport from the mine, and load the trains. These costs were included in the cash flow model.

18.2. Metallurgical Plant

18.2.1. General Site Works

BlackRock Metals Inc. (BRM) is developing a metallurgical plant 400 km from the vanadium-titanium-magnetite mine deposit site at the Port of Saguenay, Quebec, Canada. The metallurgical plant site is characterized by a non-homogenous soil with bedrock at different elevations below the surface. Development Port Saguenay (DPS) will provide the main infrastructure for the industrial zone of the metallurgical plant located 2.5 km from the wharf of Grande-Anse Maritime Terminal on the Saguenay River (see Figure 18-16).

Figure 18-16: Metallurgical plant location relative to the Port

The metallurgical plant will include a VTM concentrate receiving and storage area, a reclaiming and mechanical handling system, a pelletizing plant, direct reduction plant, and a smelting and refining plant. Other types of facilities on site will include administration buildings, a warehouse, and a laboratory. Access roads to the port, railway, and potable water are currently available. The city of Saguenay is to build a pipeline to transport potable water to the metallurgical plant. Hydro-Québec is to build an electrical power transmission line and a natural gas line. Figure 18-17 shows the overall layout of the BRM metallurgical plant.

Figure 18-17: Overall layout of the BRM Metallurgical Plant

18.2.2. Potable Water

The municipality of Saguenay will provide a back-up source of process water from the Chicoutimi pumping station. The back-up source of process water will have potable water quality. The potable water will be delivered via a process water pipeline.

The potable water will be supplied to the on-site ancillary buildings via a distribution network including isolation valves at intersections and a drain at its extremity. The potable water piping network will be installed underground. The main pipe forged of DR18 PVC will be at least 75 mm in diameter and the pipe connections to buildings forged of DR11 HDPE will be at least 25 mm in diameter.

18.2.3. Sewage

The municipality of Saguenay will take on the responsibility of sanitary wastewater treatment for the BlackRock metallurgical plant. The wastewater sanitary networks will be designed to handle peak flow rate conditions.

18.2.4. Roads

From Québec City, the beginning of the primary access route to the region is by paved Highway 175 North, running east of Lac Kénogami. Highway 175 North is intersected by Highway 170 East that leads to Chemin de la Grande-Anse. Chemin du Quai-Marcel-Dionne cuts on Chemin de la Grande-Anse and leads to the project site. A security gate will be put in place at the main road entrance to the metallurgical plant site to ensure access is limited to site personnel.

The roads on-site will respect the international road technical specifications. Three types of roads are recommended for the BlackRock project:

- Heavy-load vehicle roads (Road type – I);
- Light-load vehicle roads (Road type – II);
- Service vehicle roads (Road type – III).

On-site roads drainage will be possible due to a ditch drainage network. The depth of the ditches will need to allow for proper drainage of the road foundation. An access road to the pig iron storage area will facilitate the in/out material handling with heavy machinery.

18.2.5. VTM Concentrate Storage

The VTM storage area includes the VTM storage building and a material handling system (see Figure 18-8. The VTM concentrate storage building measures 32 m x 112.5 m which includes a material handling system for reclaiming. The capacity of the VTM storage building is 26,000 tons. The railcar unloading station and pellet plant are located directly next to the VTM concentrate storage building.

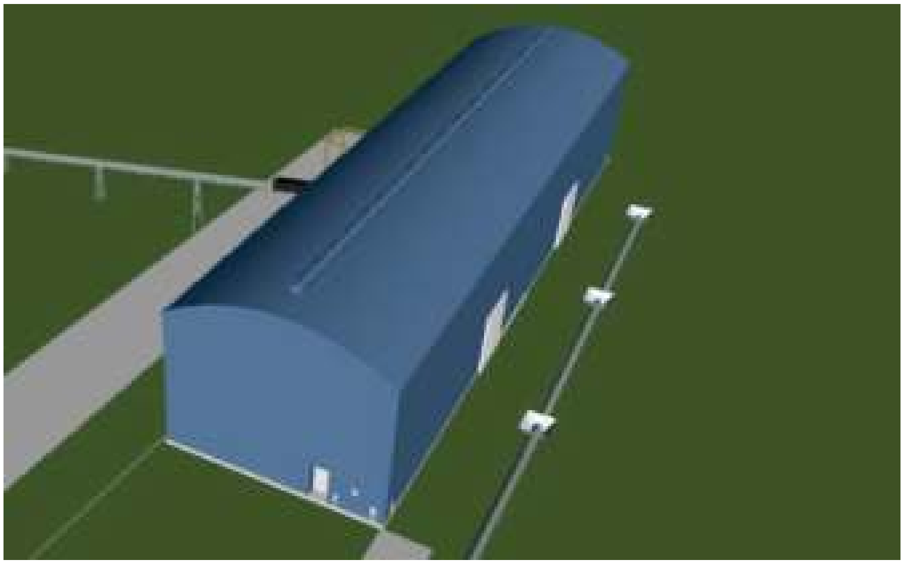


Figure 18-18: VTM concentrate storage area

Pellet Plant and Material Handling Area

The pellet plant area includes a material handling system, mixing, balling, indurating, hearth layer separation, pellet storage, screening, and coating (see Figure 18-9).

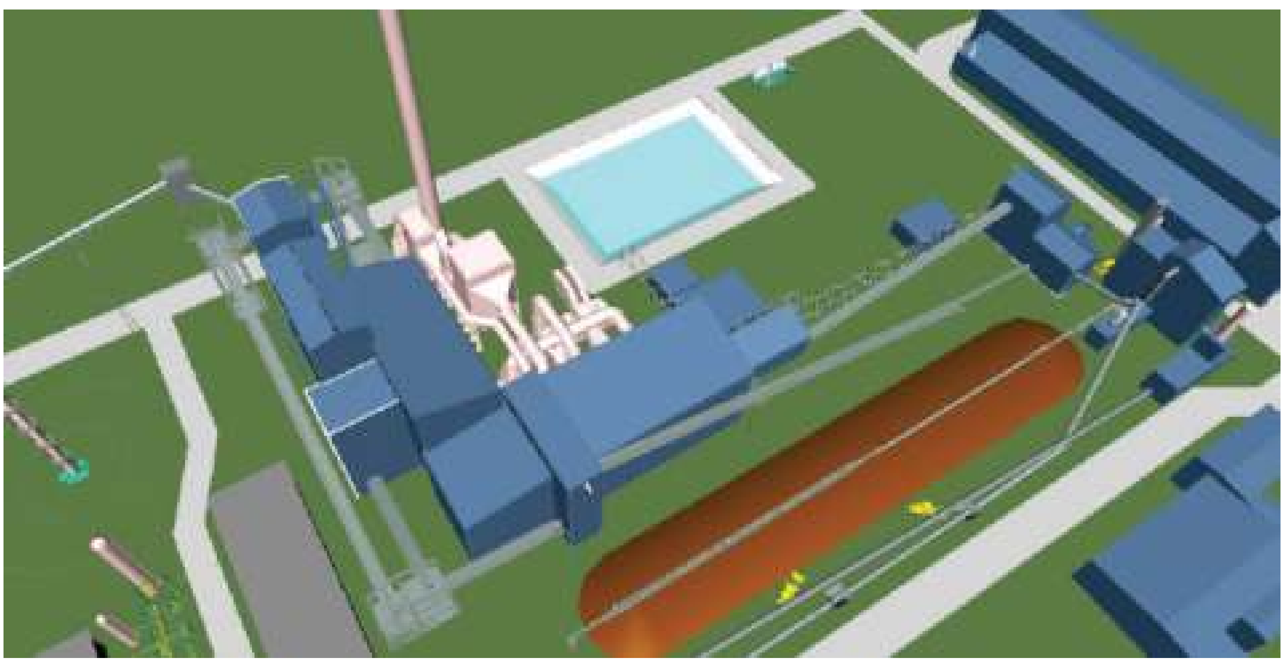


Figure 18-19: Pellet plant and material handling area

18.2.6. Direct Reduction Plant

The direct reduction plant includes curing bins, a process gas heater, DRI reactor, auxiliary equipment of the reduction area, CO_2 removal unit, and DMDS tank (see Figure 18-20).

Figure 18-20: Direct reduction plant

18.2.7. OSBF, Converter and Granulation

The smelting area includes the OSBF transformer, furnace shell refractory maintenance area, ladle refractory maintenance area, control room, pig iron storage, converter area, granulation area, and auxiliary equipment (see Figure 18-21). The pig iron storage area covers 6,240 m^2 with a capacity of 28,000 tons. In addition, the pig iron storage area will have a roof to cover the product from snow and rain. The titanium slag produced is transported from the OSBF building to another building divided into three sections:

- Titanium slag cooling section measuring 68 m x 49 m;
- Titanium treatment section measuring 68 m x 35 m;
- Titanium storage section measuring 68 m x 60 m.

A 100 tonne CLU converter will be used for metal refining and vanadium slag production. The CLU converter will process approximately one batch per hour. A metal granulation unit will be used to further process refined metal from the converter. For the granulation process, a GRANSHOT unit with a nominal capacity of 3 t/minute will be utilized.

Figure 18-21: Smelting plant

18.2.8. Accommodation Allowance

Due to the proximity of the site to the town of Saguenay, no provisions have been made for the construction of a permanent camp. The project places an emphasis on utilizing local manpower. However, travel and accommodation allowances will be given to employees based on the distance of residence to the construction site.

At site, services will include a potable water network, fire protection loop, and a sanitary sewage collection network complete with a treatment facility.

18.2.9. Electrical, Communication and Automation

18.2.9.1. Site Electrical Distribution

The 735-161 kV Saguenay power station, specifically circuits #1640-1641, will be the energy source for the metallurgical plant. The new 9 km high-voltage Hydro-Québec incoming line will be integrated with the existing electrical network and supply HV/MV transformers in the main BlackRock Metals substation (see Figure 18-22). The main substation shall be designed to allow future expansion and meet Hydro-Québec requirements stated in under "Technical Requirements for Customer Facilities Connected to the Hydro-Québec Transmission System".

Figure 18-22: New high-voltage Hydro-Québec line integrated with the existing electrical network

The metallurgical plant power demand is estimated to be approximately 120 MW. A power consumption of 5.6 MW, not including power consumption for the oxide handling and auxiliary systems, was used for pelletizing plant design purposes.

The smelter plant will receive power from the Hydro-Québec transmission network grid at 161 kV. The open slag bath furnace (OSBF) used for smelting purposes on site is a submerged electric arc furnace with a nominal power of 69.2 MVA at 0.85 PF supplied by a 161-34.5 kV transformer. The power consumption per metric ton of tapped metal will be 710 kWh/t; three main 75 MVA furnace transformers will be used. The service load of the metallurgical plant will be 25 MW. The service is made up of motors on VFD connected to the 161 kV network via a second 161-34.5 kV transformer. The 34.5 kV distribution voltage will be brought down to a medium voltage of 4.16 kV and 600 V for primary power distribution throughout the plant, using 34.5-4.16 kV and 34.5-0.6 kV outdoor oil-type transformers. A low-voltage cable, up to 1 kV, is used to connect electrical equipment and all low-voltage loads to MCC's in the main process and utilities areas.

To limit the impact of voltage and harmonics disturbance on other process and service equipment, the OSBF will be fed from an isolated HV/MV transformer. Two 34.5 kV harmonic filters of 20 Mvar each will be used to improve the metallurgical plant's overall power factor to 0.95 or above by reducing harmonic voltage distortion currents produced by the arc furnace from flowing in the power system. The ideal voltage range can be controlled by utilizing STATCOM technology.

The electrical distribution network dedicated to the site infrastructure will consist of multiple substation stations feeding power to all three areas of the metallurgical plant at different voltage intensities:

- Iron ore pelletizing area (Electrical building #1 @ 600 V + Electrical building #2 @ 4.16 kV and 600 V);
- DRI area (E-Room @ 600 V + Substation @ 4.16 kV + Gas compressor E-room @ 4.16 kV);
- Pyromet smelting area (Alumino-thermic plant @ 600 V + OSBF E-room @ 600 V).

The electrical rooms and buildings will strategically be located near the equipment they serve. To prevent fluid leakage/spillage, the electrical rooms will be fully enclosed with a roof and the walls will be coated with suitable material to prevent migration of wash-down material into the electrical rooms. For extra precaution, no piping or conveyor installations will be permitted above the electrical rooms. Moreover, all cables, conduits, and HVAC ducting will be inclined downward with one metre of the outside wall to direct any possible dust or fluid away from the room or building. Inside the electrical rooms, electrical components will be installed such as switchgears, MCCs, VFDs, PCS cabinets, fire alarm protection panels, telecommunication cabinets and lighting/distribution transformers and panels. The fire alarms will be connected to a photo-electric smoke detector which will alert operations locally and remotely at a central panel. The temperature inside the electrical rooms will be maintained between 15 °C and 30 °C.

The main distribution system consists of installed electrical components in the control room. Switchgears will supply feeder loads in a radial configuration. Each component of the radial circuit will have the capacity to provide 125% of the maximum power demand. The control voltage for 161 kV GIS, 34.5 kV high-voltage switchgears will be 125 VDC. The control voltage for both medium-voltage and low-voltage switchgears will be 120 VAC. The feeder loads will provide power to unitized motor control centres and non-essential electrical loads. Redundant process loads and service loads will each be grouped into two separate MCCs or panels boards. All switchgears and MCCs will be sized to not exceed 80% to 85% of the equipment rating, thereby leaving spare capacity available for future expansion. MCCs and switchgears will be integrated onto the Delta V network via IEC61850.

The metallurgical plant will contain emergency diesel generators which will operate in the event of an electric power failure. In this situation, a total plant shutdown will occur triggering a signal in the form of a no-volt relay sensing voltage level. If the voltage level is less than half the volts on the incoming supply for more than two seconds, then the emergency diesel generators will start automatically.

18.2.9.2. Communication

Communication technologies such as videoconferencing, internet-based protocols (Ethernet IP, WebEx, etc) will be leveraged as much as possible to reduce communication costs. A telephone system using Voice over Internet Protocol (VoIP) will be installed in each electrical room to facilitate plant equipment operation. Telephones will be available in administration buildings. A paging system will be installed through the plant as well. A closed-circuit television (CCTV) will be implemented in the iron ore coating system because there is a lack of camera surveillance in that area. Three CCTV will be connected via I/P and the recordings will be stored in a CCTV server.

A mobile radio system will be supplied to operators in the field to communicate with control room operators. A wireless tablet device will be an alternative communication tool for coordination between operators and the control room.

18.2.9.3. Automation

18.1.1.1.13 Process Network

The main automation equipment on site includes the SCADA system and instrumentation which will display measurements in SI units. All actuators and field instruments will be set at 120 VAC or 4-20 ma. The plant wide fibre-optic backbone network will link the main automation equipment such as SCADA, local human-machine interface (HMI), PCS (processor only), and main electrical substation.

18.1.1.1.14 Process Control System

Each process plant sector with their own programmable logic controller (PLC) will have a process & safety control system (PSCS) being used as a SCADA system to provide high-speed efficient and secure communications at all levels and monitor electrical parameters at each substation. The process control network (PCN) will link all PSCS devices.

The PSCS will have redundancy requirements for the DRI plant. The direct reduction plant utilizes 4-29 mA + hart signals for the Safety Instrumented System and Foundation Fieldbus with redundant power supplies including diagnostics for controlling and monitoring.

Various control components will be put in place to protect process circuits and for emergency uses. For instance, lead acid valve regulated batteries with gel cells and chargers will provide minimum one-hour back-up of DC power for control, protection, and tripping functions. In addition, local motor control stations located I the field will include start and stop pushbuttons. The stop pushbutton will be hard-wired to MCC intelligent relay which will be programmed to instantly trip the motor and send a signal to the PCS via Ethernet IP protocol, if the stop pushbutton is pressed. All rotating equipment will be equipped with zero-speed switches. Other control components such as pull cords and personal safety devices will be hard-wired to the MCC and linked to the PCS as well.

All controls and systems for the water treatment plant will be designed to have a fail-safe. The water treatment plant will have its own monitoring and safety interlock systems which will be hard-wired in the DCS.

18.2.9.4. Process Analog Instruments

All electrical equipment and instrumentation will be in compliance with the Canadian Standard Association (CSA). Analog inputs and outputs will have 24 V DC power supplies and distribution that will be circuit breaker protected. The 24 V DC power supplies for instrumentation will be filtered and regulated.

The balling process system contains a disc supplied by Metso. Specific electrical components are necessary for the proper functionality and safe operation of the disc in the balling system:

- Main drive motor (250 HP, 1.15 service factor, 600 V, 3 phase, 60 hertz, 1500 RPM variable frequency duty motor enclosed for taconite service);
- Disc variable-frequency drive (250 HP, 3 phase, 600 V, 60 hertz input with digital operator panel);
- The instruments receiving 60 hertz power will be protected by circuit breakers.

A conventional analog V/A/kW metering function will be used for local and remote indication to protect circuit relays.

18.2.10. Natural Gas Storage and Dispensing

Natural gas will have a dual purpose for secondary transformation. Natural gas will used as a reductant for the direct reduction process and a fuel for the process gas heater, pelletizing plant and heat production in the buildings during the winter. The company Energir will use its 20 km pipeline to locally distribute natural gas. However, the natural gas used is produced by suppliers in other Canadian regions and delivered via the Trans Canada Pipeline. The natural gas station measures 12.5 m x 4.5 m. The annual consumption of natural gas for the metallurgical plant is estimated to be 190,497,120 Nm^3.

18.2.11. Oxygen and Nitrogen Storage

A platform for an oxygen storage shed and nitrogen storage shed will be built at a safe distance (minimum 1 km from any building), west of the metallurgical plant area. Both the oxygen storage shed, and the nitrogen storage shed measure 60 m x 50 m.

Figure 18-23 represents the oxygen generation circuit for the metallurgical plant including the PRISM VSA O_2 Generator with a nominal capacity of 5,172 Nm^3/h. The minimum delivery pressure will 13.7 barg for the continuous flow to the DRI unit and the maximum delivery pressure will be 22 barg for the intermittent demands to the OSBF unit.

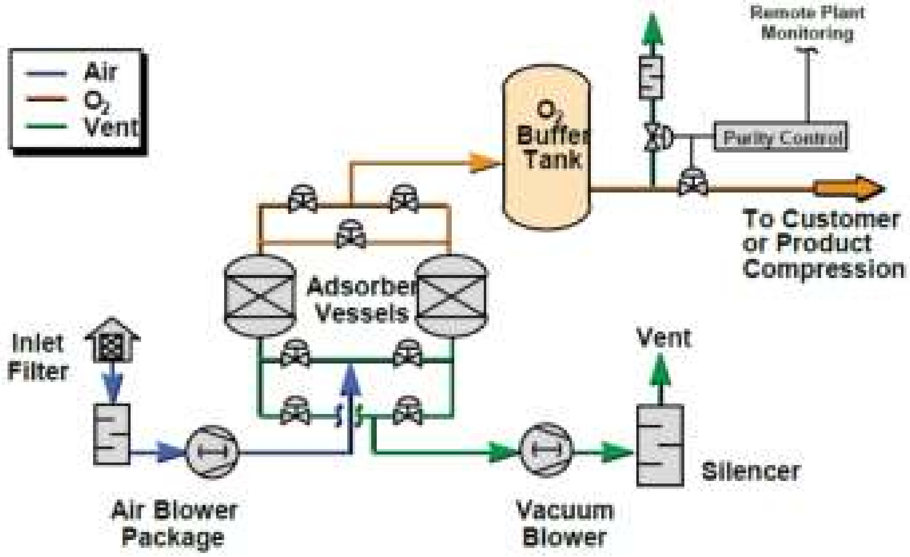


Figure 18-23: Oxygen generation circuit in the Metallurgical Plant

Figure 18-24 represents the nitrogen generation circuit for the metallurgical plant including the PRISM Nitrogen GN unit with a capacity of 2,620 Nm^3/h.

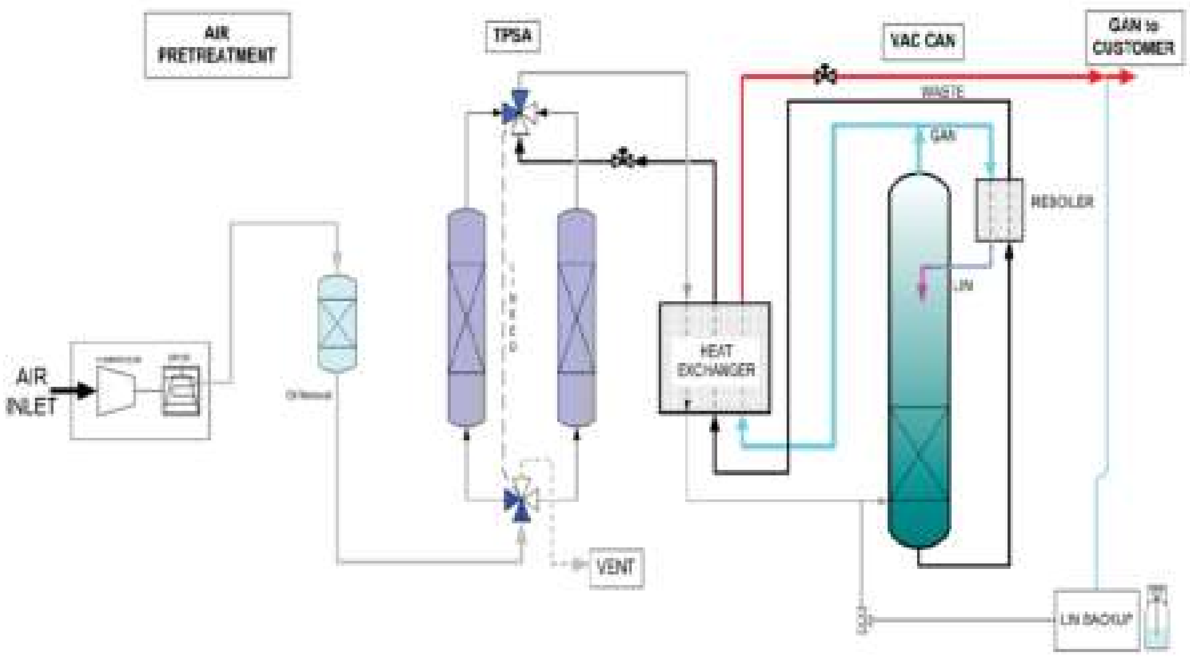


Figure 18-24: Nitrogen generation circuit in the Metallurgical Plant

18.2.12. Solid Waste and Off-gases Management

All air and solid emissions from the metallurgical plant will be accounted for. Waste slag and dust from the dust collectors will be stored in an engineering area. Access for monitoring and inspection will be provided in the waste disposal engineering areas. The risk of leaching from the waste slag is considered very low.

18.2.12.1. Pelletizing Plant

Gaseous emissions include products of combustion discharged from the indurating waste gas stack such CO_2 and SOX. Other gaseous compounds existing in low concentrations are nitrous oxides, carbon monoxide, water vapour, nitrogen, and oxygen. The indurating process off-gases will be cleaned with electrostatic precipitators (ESPs), hood exhaust and windbox exhaust. Dust and any other particulates collected will be pneumatically conveyed to a dust bin to be recycled back to the process. The ESPs in the plant will be designed to discharge gases to the environment with a concentration of particulates less than 30 mg/Nm^3.

Baghouses will be used to collect fugitive dust in certain sections of the pelletizing plant:

- Ground additive storage bins;
- Dust recycle bin;
- Mixer feed area;
- Indurating machine discharge and hearth layer bin area;
- Hearth layer separation area and pellet screening ahead of DRI furnace.

The baghouses will be designed to discharge gases to the environment with a concentration of particulates less than 10 mg/Nm^3.

18.2.12.2. Direct Reduction Plant

Reducing top gases produced will be recycled back to the process in a closed-loop circuit. Exhaust reducing top gases will leave the DRI reactor and pass through a top gas heat recovery system. A counter-current shell and tube heat will be used for heat recovery. Then, the exhaust gas will report to the quenching/scrubbing system. The exhaust gas will be quenched in a direct contact quench orifice. Other equipment included in the quenching/scrubbing system are a process gas venturi, process gas separator, process gas quench tower, and process KO drum. Scrubbed gas will pass through a process gas recycle compressor which comes with an electric motor, set of gas filters, a lube oil console, and process gas aftercooler. Finally, the recycled process gas will be sent to the carbon dioxide removal unit which will mainly consist of one CO_2 absorption column and one stripping column. Figure 18-25 shows the carbon dioxide removal system consisting of the two main columns and other main equipment.

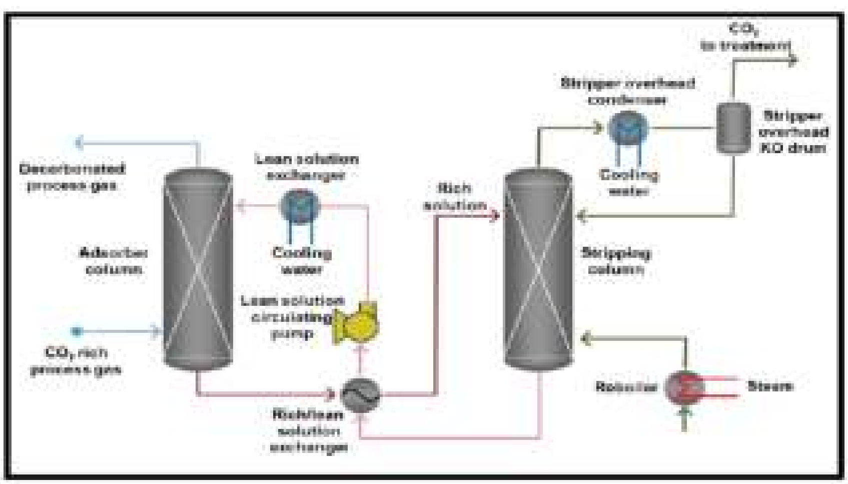


Figure 18-25: Carbon dioxide removal system in the Metallurgical Plant

18.2.12.3. Smelting Plant

The OSBF off-gases will be treated in a gas cleaning plant like the direct reduction off-gases. The OSBF off-gases will be subjected to scrubbing, then sedimentation and filtration to rid the scrubbing water of particulates to recycle the water.

18.2.13. Site Water Management

18.2.13.1. Process Water Balance and Description

Process water will be assumed to be of potable water quality, supplied from the raw water pumping station through the water supply line at a sufficient pressure to fill the raw water tank on site. The complete water balance will be finalized following the selection of water treatment plant technology and supplier.

To minimize rejects to the environment, the BlackRock water balance has been designed to recycle quenching cooling water, hydrocyclone reject water, and identify an excess of purge water which can be used as make-up water. A portion of the clarified water will be lost to the treated off-gas. The difference between the water entering the scrubbing plant and the slurry returned to the thickener will be made-up in the water treatment area.

Major process make-up water additions consist of the following process streams:

- Machine cooling water (MCW) systems;
- Equipment cooling water (ECW)/Quenching cooling water (QCW);
- Direct reduction plant (DRP) process cooling water clarifier;
- Open slag bath furnace (OSBF) process cooling water clarifier;
- Users.

The make-up water is needed to compensate for water losses in various cooling circuits. Based on design assumptions, the metallurgical plant will require at least 305 m^3/h of clean make-up water. The make-up water will be distributed via a network of underground carbon steel pipes.

18.2.13.2. Water Cooling and Treatment

A water treatment plant will be built on the metallurgical plant site to provide process, cooling, and service water required by the facilities. Water supplied from the Saguenay municipality will pass through a filtration system prior to distribution to remove any particulates. The water treatment plant will be designed to be a zero-discharge operation. Any water released to the environment will be monitored and taken into consideration. In addition, the water released will be treated to meet local, provincial, and federal regulations.

The cooling plant will contain equipment such as air coolers, cooling towers, and clarifiers. The cooling tower is used to cool the process stream from 80 °C to a discharge temperature of 30 °C into a single gravity return line.

The water treatment and cooling plant will include:

- Demineralization plant;
- Water cooling system for machine cooling water (MCW) for direct reduction plant and OSBF plant;
- Water cooling system for equipment/quenching cooling waters;
- Water treatment system of process cooling water (PCW) with two distinct clarification circuits;
- Water treatment system for water purge in the PCW circuit for ammonia removal.

Each water treatment and cooling circuit (ECW, QCW, MCW, PCW) will incorporate valves, process instrumentation, and tanks into a pumping system.

A demineralization system and a chemical dosing system will be installed to help control ECW and QCW quality. The type of chemical dosing will include scale inhibitor, biocide, and sulphuric acid. All cooled QCW shall be pumped back to the DR quenching system from the water treatment plant. Similarly, all cooled ECW shall be pumped back to the users from the water treatment plant.

All cooled MCW shall be pumped back to the DR and OSBF machine cooling system from the water treatment plant. A 250 m^3 storage will be used to store the DR MCW return. A heating system will help avoid ice formation during a shutdown in cold weather months. An automatic make-up water system and a chemical dosing system will be utilized to control water quality. The type of chemicals used for dosing will include corrosion inhibitor and biocide.

Two PCW treatment systems, one for DR and one for OSBF, will each have a suspended water clarifier with a carbon shell, motorized scraping bridge, and sludge discharging system. The sludge will be pumped to the dewatering system; a hydrocyclone and a filter press will be used to reduce moisture content below 10%. Dry sludge will be discharged on an indoor concrete pad or in bins. The filter water will be pumped back to the PCW clarifier. A skid will be used for chemical dosing to help control pH level.

Figure 18-26 is a 3D representation of the water treatment building:

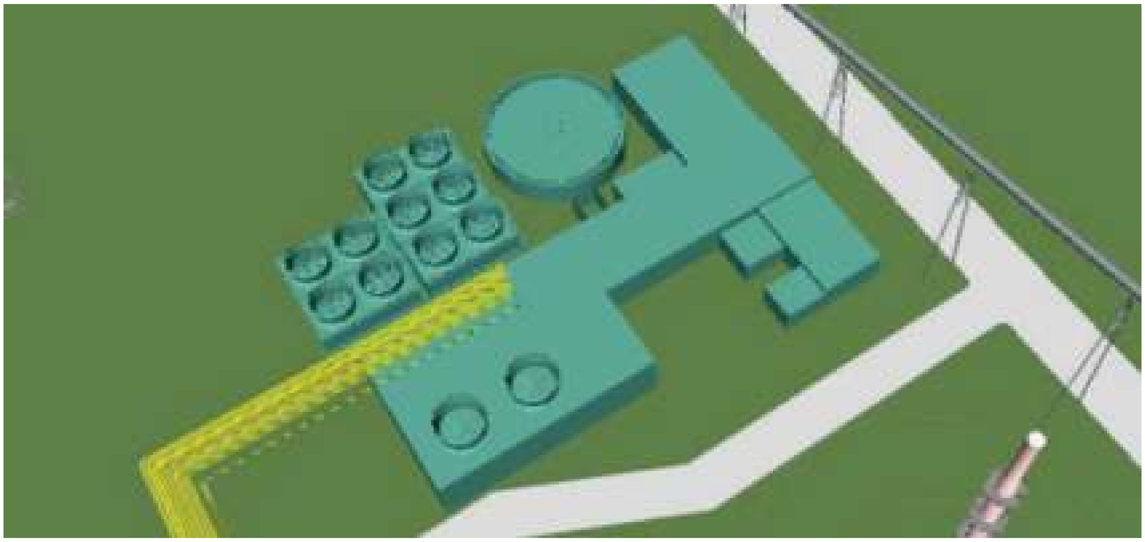


Figure 18-26: Water treatment plant 3D Model

Filtered clarified water will be supplied to the scrubber area at three different heights and to the OSBF building at two different heights. The scrubber area consists of the scrubber header tank, overpressure relief device, wet gas scrubbing plant manifold, emergency stack pressure relief device, off-gas duct water seal, wet gas scrubbing plant manifold, and gas plant slurry tank. The OSBF closed-loop cooling water system consists of a cooling water tundish, a high-pressure header and two low-pressure headers.

18.2.13.3. Drainage Network System

Drainage ditches across the entire site will be used to prevent the collection of water on roadways, around buildings and structures. The drainage ditches will collect rainfall and snowmelt. The purpose of the drainage network system is to prevent any damage or hazards associated with the pooling of water on roads and other site surfaces. Drainage routes/networks will be designed to accommodate the maximum flow rate resulting from either rainfall or firewater and other known contributing flows. The drainage system tributary surface areas will be slopes to ensure proper collection and storage in the detention basins. A rip-rap barrier around the entire basin area will be required to protect the basin area from erosion. An underground concrete chamber will be the location of the water flow rate controls. The Storm Water Management Model (SWMM) will be used to monitor continuous runoff flow rates and the flow rates of water reporting to retention ponds and treatment technologies.

18.2.14. Fire Protection

The fire protection system is designed based on the assumption that only one major will happen at a given time. To be consistent with National Fire Protection Association (NFPA) standards, the fire protection system will include the following equipment:

- Fire water tank;
- Fire water pumping station (including jockey pumps and main pumps with a pump in constant standby service with connection to the emergency generator);
- Piping distributor system;
- Control equipment;
- Monitoring equipment;
- Hydrants.

The fire water tank is an atmospheric, carbon steel tank providing about two hours of retention time. It is heated and insulated for protection against the cold during the winter months. The flow rate of the fire-water tank must be sufficient to fill the entire tank in less than eight hours. The main firewater pumping station is housed in an existing heated building adjacent to the fire-water tank. The pumping station measures 12 m x 8 m and includes an electric fire water pump and a standby diesel fire water pump designed to deliver 340 m^3/h for a pump head of 95 m. The electric jockey pump in the pumping station can deliver up to 30 m^3/h to maintain pressure. All the pumps in the pumping station are horizontal centrifugal pumps tied into the underground firewater loop which connects to the aboveground distribution piping system located throughout the metallurgical plant.

18.2.15. Train Loadout

The railway companies involved include Canadian National Railway and Roberval Saguenay. The rail cars will be leased by Blackrock. During regular operation, there will be five trains of VTM concentrate per week, each with 40-45 railcars, will arrive to be unloaded. A front-end unloader and conveyors will handle the concentrate to bring it from the railcar unloading station to the concentrate storage building.

19. Market Studies and Contracts

The qualified person has reviewed these studies and analyses and the results support the assumptions in this technical report.

19.1. Market Studies

19.1.1. Pig Iron Market

According to the International Iron Metallics Association (IIMA), Ore Based Metallics (OBMs) are iron feedstock materials used alongside scrap in the production of Steel. OBMs are produced mainly by reduction of iron ore in blast furnaces and direct reduction plants to produce Pig Iron, Direct Reduced Iron (DRI") or Hot Briquetted Iron (HBI). OBMs are characterized by consistent and predictable quality with a low content of metallic impurities and can be used to dilute the residual impurities in scrap.

Pig iron is a high iron content, low residual metallic material for producing high-quality iron and steel products. It is used as a source of clean iron units to supplement and enhance the scrap charge. Many Electric Arc Furnace (EAF) operators prefer using pig Iron to blend with scrap and other feedstock materials due to its high Fe content, low gangue, and chemical purity.

The vast majority of pig iron is produced and consumed within integrated steel mill complexes. Within integrated steel mills, blast furnace iron is transferred directly to the steel plant in liquid form, better known as "hot metal" or "blast furnace iron”.

Merchant Pig Iron (MPI) is cold pig iron, cast into ingots and sold to third parties as feedstock for the steel and ferrous casting industries. MPI is produced by dedicated merchant plants, who sell all of their production to external customers. MPI is also produced by integrated steel mills, with surplus iron that exceeds their internal requirements and is cast into ingots and sold to external customers.

There are three types of MPI, which differ in chemical and mechanical characteristics and have a variety of uses:

- **Basic Pig Iron**: Mainly used in electric arc furnace steelmaking;
- **Foundry Pig Iron**: Mainly used in grey iron castings, such as engine cylinders and flywheels;
- **High Purity Pig Iron (Nodular Pig Iron)**: Mainly used by foundries in the manufacture of ductile or nodular iron such as gears and suspension components, which require superior strength and hardness.

Wood Mackenzie provided an analysis of BlackRock`s High Purity Pig Iron / Nodular Pig Iron (HPPI / NPI) and benchmarked against other products available in the market. A detailed NPI price forecast was also prepared as part of the study. NPI benchmark assessment results indicated that BlackRock's HPPI product specifications met or exceeded industry standards as indicated below.

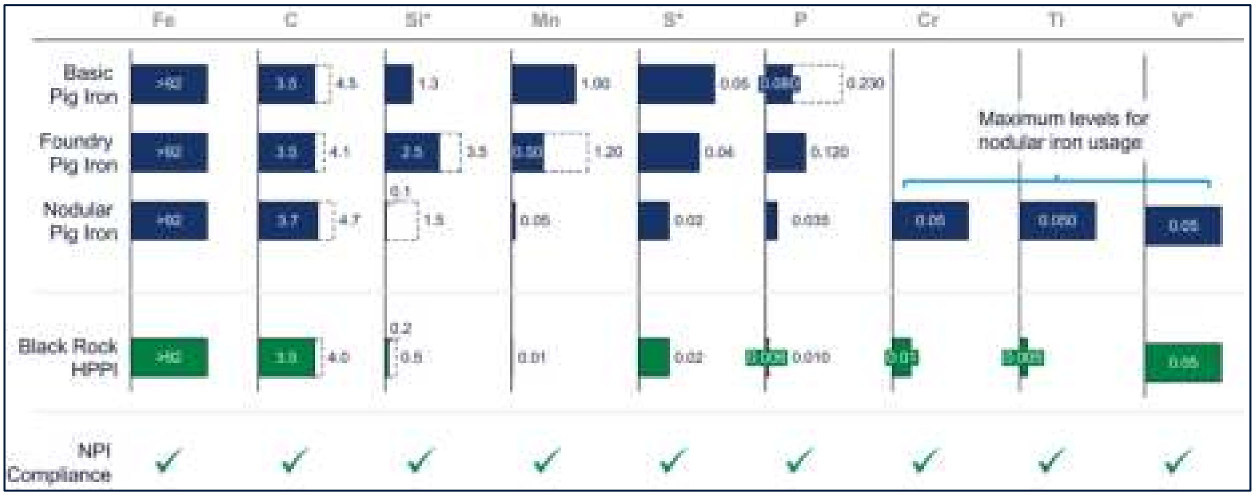


Figure 19-1: Black Rock HPPI Chemical content benchmarked vs. industry standards (%weight)
Source: Wood Mackenzie, BlackRock Metals, Ductile Iron Society; International Iron Metallics Association

NPI is further divided into three categories based on the level of P, S and Mn content:

- **T5 (Standard):** Contains up to 0.050% of phosphorous & manganese (and up to 0.02% sulphur);
- **T35 (High-Grade):** Contains 0.035% or less of phosphorus & manganese and receives a premium price to T5 grade;
- **T2 (Super-High-Grade):** Contains 0.020% or less of phosphorus & manganese and receives a further premium price to T35.

BlackRock's HPPI would be classified as T2 and command a premium price due to its low P, S and Mn content.

19.1.2. Pig Iron Pricing

Wood Mackenzie performed a predictive statistical analysis over selected related products in the market to review the pricing structure for NPI and the means to develop a method to forecast the future pricing trends of NPI. An analysis of historic pricing for different grades of pig iron was also conducted.

Based on this analysis and other work conducted by BlackRock, an average price over the forecasted period of USD786/t was used. Challenges on supply of high-quality prime scrap due to availability/chemical composition, could increase BlackRock's NPI premium, as higher purity products like NPI would be required to dilute the lower quality scrap.

19.1.3. Vanadium Market

The most common occurrence of vanadium is in vanadium and titanium magnetite (VTM) ore bodies. Vanadium feedstocks are produced via three main routes; co-production, primary production and secondary production.

Vanadium co-production comes from vanadium rich slags that are produced by steel plants processing VTM ores. This vanadium rich slag accounts for approximately 70% of vanadium production.

Primary production is the result of vanadium being produced as the primary product from mining and processing VTM ores. Primary production accounts for approximately 20% of vanadium production.

Secondary production of vanadium is the result of processing spent catalysts, residues and ashes largely from crude oil and oil sands in order to extract the vanadium. Secondary production accounts for approximately 10% of vanadium production.

Vanadium oxides are the most commonly produced vanadium compounds. Vanadium oxides can be used in the production of vanadium chemicals and in non-metallurgical applications. Vanadium pentoxide (V_2O_5) is the principal intermediate product produced from processing vanadium feedstocks. Most V_2O_5 is further processed into Ferrovanadium.

Ferrovanadium (FeV) is the main commercially produced alloy of vanadium and is produced in alloys grading 40% - 80%, with FeV_{80} being the most common alloy. FeV is formed by combining iron and vanadium. Ferrovanadium is a universal hardener, strengthener, and anti-corrosive additive for steels like high-strength low-alloy steel, tool steels, as well as other ferrous-based products.

Vanadium is also increasingly being used in vanadium redox flow batteries (VRFB). VRFBs are durable, have a long lifespan, low operating costs, safe operation and a low environmental impact in manufacturing and recycling. VRFB technology has advanced considerably in recent years and these batteries are ideal for storage applications requiring longer duration discharge and 20+ years of operation with minimal maintenance. The vanadium liquid electrolyte used in VRFBs can be recycled with minimal costs and reused in another battery application.

Market studies conducted by Guidehouse indicate that VRFB installation would require additional volumes of vanadium. The net effect on the demand of vanadium due to VRFB deployment is reflected in the graph below. The net effect requires new inputs to come online from 2025 to offset the demand. This demonstrates the necessity for the BRM project to produce the vanadium to support the anticipated vanadium demand.

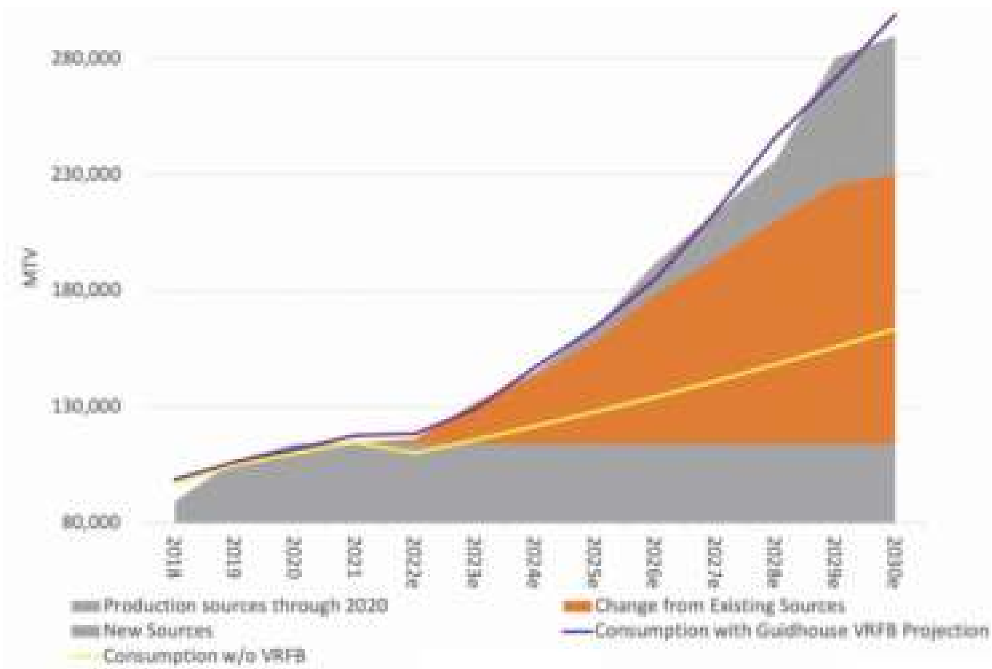


Figure 19-2: Vanadium production and demand
Source: TTP.

BlackRock will be producing a vanadium rich slag, via the co-production route, which will then be processed into FeV_{80} and other vanadium products for metallurgical and battery use by a vanadium processor.

19.1.4. Vanadium Pricing

Project Blue provided a market analysis and price forecast for FeV_{80}, by examining market drivers like the global steel sector, outlook for Chinese rebar and vanadium consumption in rebar, and the supply/demand and market balance for the vanadium market. Forecasted prices take into account expectations for the global economy, supply/demand balance and marginal cost of production.

Based on this analysis and other work conducted by BlackRock, an average price over the forecasted period of USD38.17/kg of FeV_{80} was used.

19.1.5. Titanium Market

Most titanium dioxide (TiO_2) products are derived from three naturally occurring minerals which are commonly referred to as heavy minerals or mineral sands: ilmenite, leucoxene and rutile. Ilmenite, rutile, leucoxene, as well as titanium slag and synthetic rutile which are processed from ilmenite bearing ores, are the primary feedstock materials for the production of TiO_2 pigment.

TiO_2 pigment is used in a wide range of products due to its ability to impart whiteness, brightness, and opacity. Approximately 90% of titanium dioxide globally is used as a pigment in the manufacture of paint, plastic, paper and fibre where, in addition to being a non-toxic whitener, it also provides UV and chemical resistance. The wide range of end applications for pigment includes house and car paints, laminates, plastic pipes and packaging, inks, clothing, sunscreen, toothpaste and make-up.

Titanium minerals are also used to produce titanium metal, which has the highest strength-to-weight ratio of all metals. Titanium metal is also chemically resistant, has a high melting point and low conductivity. It is used across a diverse range of applications including aeronautics, medical implants, defence, sporting goods and componentry in the offshore mining and petrochemical industries.

The U.S. Census Bureau provides import data on various titanium feedstocks and products into the U.S. The data in the table below was extracted from the USGS Mineral Industry Survey: titanium in the first quarter 2022 and provides the prices for titanium material for 2021 and Q1 2022. The data below indicates that the imported ilmenite price increased from Q1 2021 to 2022 by 38 to 48% and the titanium slag imported price increased by 49%.

Table 19-1: Prices Titanium Materials

		2021				2022
		1st quarter	2nd quarter	3rd quarter	4th quarter	1st quarter
Concentrate						
Ilmenite, CIR, China[1]	dollars per metric ton	260 - 270	280-290	290 - 310	360 - 400	360 - 400
Titanium slag, import[2]						
South Africa	do.	510	540	780	790	760

Note:
1. Source: Fastmarkets IM.
2. Unit value based on landed-duty-paid U.S. imports for consumption.

BlackRock mandated Symphony Trade, to evaluate the commercial viability of its TiO_2 Slag produced from the OSBF furnace. This report showed that the TiO_2 Slag price BlackRock can expect will be in the range of USD250/t - USD380/t. The report was produced when Ilmenite and slag prices were well below the current prices indicated in the table above. For the purposes of this report, BlackRock used a price of USD300/t.

Table 19-2: Expected Price from TiO_2 Slag

By-Product	Process Area	Market Price (USD)	FS Price used
TiO_2 Slag	OSBF Furnace after FeSi treatment	\$250 - \$380	\$300

19.2. Contracts and agreements

19.2.1. Off-Take and Agreements

BlackRock is in discussion with potential off takers for the pig iron, the vanadium slag and the titanium slag products. It is expected that these offtakes will be signed closer to construction financing.

19.2.2. Port Lease Agreement

BlackRock signed a Contract with the Port Authority (DPS) of Saguenay in 2018 to lease the industrial land for the Metallurgical Plant and to use the existing wharf.

19.3. Railway Transportation

The concentrate will be transported via truck to a rail yard located 25 km from the mine site. Once loaded onto train cars, the concentrate will then be transported on a CN rail segment (358 km), a Roberval Saguenay Segment (15 km) and a Saguenay Port Authority Segment (12 km). The CN Railway has a Common Carrier Obligation and the Tariffs are governed by Transport Canada. BlackRock received proposals from CN in 2018 for the Tariffs and operation mode. BlackRock signed a Letter of Intent with the Roberval Saguenay Railway (RSR) in 2018.

19.4. Electric Power Supply

BlackRock signed an agreement with Hydro-Québec in March 2012 for the engineering, construction and delivery of electrical power to its mine site. Another agreement was signed in 2019 with Hydro-Quebec for the construction of a power transmission line for its Metallurgical Facility in Saguenay.

Engineering was completed and construction of the transmission lines to provide power to the sites are on hold until BlackRock confirms the new construction schedule to Hydro-Québec. The commercial terms and rates for power, transmission and other infrastructure costs will be governed by Hydro-Québec's Rates Policy Framework.

19.5. Natural Gas Supply

BlackRock signed a service contract with Énergir in 2018 to supply Natural Gas to its metallurgical facility. The construction of a short connecting pipeline is the responsibility of Energir and the financial guarantees related to this pipeline are the responsibility of the Development of Port Saguenay (DPS). Engineering and permitting are completed to build the pipeline.

19.6. Process Water Supply

Development Port of Saguenay (DPS) and the town of Saguenay are responsible to develop the Process Water pipeline to supply the water to the industrial site leased by BlackRock. DPS and the town of Saguenay completed the engineering to release a request for bids in 2019. They received offers and they are waiting that BlackRock confirms the new construction schedule.

19.7. Other Equipment Supply Contracts

Between 2017 and 2019, BlackRock signed several contracts with Engineering firms and Equipment suppliers to progress detailed engineering. Major contracts were signed for most mechanical equipment at the mine site. Major contracts were also signed with process experts and equipment suppliers at the Metallurgical Plant. In addition, frame agreements were signed for the supply of Electrical Drives, Electrical Motors, Instrumentation and Control equipment.

19.8. Other Agreements

BlackRock signed an Impact and Benefits Agreement (IBA), called the BallyHusky agreement on June 20, 2013, with the Oujé-Bougoumou Cree Nation, the Grand Council of the Crees (Eeyou Istchee) and the Cree Regional Authority, for the future development of the BlackRock project in the Eeyou Istchee territory. This agreement was updated in March 2015 to include the Second Transformation Plant.

BRM also signed two Partnership and Development Agreements with the Innu First Nations of Pekuakamiulnuatsh (community of Mashteuiatsh) and Pessamit and Essipit in August 2020. A partnership agreement was also signed with the municipalities of Chapais and Chibougamau in May 2013. This partnership agreement was established to promote and develop a sustainable project, taking into account the social, economic and environmental aspects.

20. Environmental Studies, Permitting and Social or Community Impact

20.1 Mine and Beneficiation Plant

20.1.1 Study Area

20.1.1.1 Regional Study Area

The regional study area includes the municipality of Chibougamau, part of the Eeyou Ishtchee James Bay Regional Government (EIJB Regional Government) and the County Regional Municipality (CRM) of Domaine-du-Roy, as well as the municipality of Chapais and First Nations communities of Oujé-Bougoumou and Mistissini. The regional study area was used in the environmental impact assessment (EIA) to present the socio-economic aspects of the region's communities, human activities, transportation networks and significant components of the natural environment that could be impacted or, conversely, influence the mining project components.

20.1.1.2 Local Study Area

The local study area is approximately 700 km^2 in size and covers the area between Route 167 and northeast of Chibougamau Lake (Girard Bay). This area will be directly affected by mining activities and transportation activities. The local study area was used in the EIA of 2011 to characterize the environmental components that could be impacted and to establish the baseline conditions prior to mining operations.

20.1.2 Social Relations

BRM is committed to interacting and cooperating with the communities in the vicinity of the project. To this end, BRM undertook to set up discussion groups, open doors and information sessions in the communities of Oujé-Bougoumou, Chibougamau, Chapais and Mistissini. Table 20-1 summarizes the main issues or concern raised during the consultation and information process.

BRM has signed an IBA (impacts and benefits agreement) with the Oujé-Bougoumou Cree Nation, the Grand Council of the Crees and the Cree Regional Authority in June 2013 which was amended in 2015. The IBA foresees the establishment of an Implementation Committee composed of representatives of Oujé-Bougoumou, the Cree Nation Government and BRM. The Tallyman and members of the Impacted Family are also invited to participate. This committee has met regularly since 2014. Three subcommittees (Employment and Training, Environment and Culture, Business Opportunities and Lodging) have also been established and meet regularly.

BRM also signed an IBA with the First Nation of Pekuakamiulnuatsh (community of Mashteuiatsh) in 2020, A joint committee was established in 2020 and meetings have been held regularly. An employment and training subcommittee was created to focus on these matters and has met regularly.

These agreements primarily focus on training and employment, working conditions and operating policies, business opportunities, cultural, social and environmental considerations, particularly for the Impacted Family, and financial considerations.

In 2016, a partnership agreement was also signed with the Chibougamau and Chapais communities and the James Bay Regional government to establish good working relations and to maximize local economic spin-offs. Regular meetings of the Regional Committee (renamed as the Liaison Committee) were held but later suspended. Meetings will resume when financing is complete.

As well, a follow-up Committee was created to address the concerns of the four communities affected by the Project: Oujé-Bougoumou, Mistissini, Chibougamau and Chapais. Representatives include citizens, representatives from local governments, the business community and First Nations. The mandate is to exchange information, identify issues and concerns, find solutions and reach out to the population. Several meetings were held. Meetings will resume once financing is complete.

Table 20-1: Main issues of concern raised during the information & consultation processes

Category	Issues of Concern
Job Creation and Economic Benefits	▪ Portion of the construction and operating costs allocated in the region. ▪ Local hiring. ▪ Use of local businesses.
Training and Hiring Conditions	▪ Work schedule. ▪ Transportation offered by the company. ▪ Company internships. ▪ Mentoring. ▪ The most useful basic training for students. ▪ When the mine would open (timing of the start of operation). ▪ Training provided by the company, particularly for truck driving and mechanical maintenance. ▪ Training for drilling and blasting. ▪ The potential for adjusting work schedules. ▪ Minimum education needed to work at the mine (e.g.: Secondary V).

Category	Issues of Concern
Project Information	▪ Construction period. ▪ Project components. ▪ The sale of the iron and vanadium produced. ▪ Project profitability. ▪ The possible impact of lower iron prices. ▪ Secondary and tertiary transformation in China rather than in Québec.
Impact on Activities Practiced by Land Users	▪ Impact on wildlife populations that might be driven away from the area by the noise and the mining activities.
Practice of Traditional Cree Activities in the Area	▪ Portion of land exempt from any major development. ▪ Maintain adequate space on the trapline O-59 for Wapachee family members to continue their traditional practices (hunting, fishing, gathering, trapping). ▪ Raise awareness of people with regard to the culture and practices of the Cree way of life.
Migration of Local Manpower toward the Mine	▪ Migration of workers from local businesses to the mine. ▪ Worker shortage in Chibougamau.
Risk of Pollution	▪ Processing method. ▪ Amount of water used. ▪ Closure plan. ▪ Post-closure monitoring. ▪ Quality and strength of the dam membranes. ▪ Potential for contamination of the environment by mining activities. ▪ Measures taken to ensure the protection of the lakes around the mine site. ▪ Possibility of an accidental release and quality of the water that might be released.
Housing Prices	▪ Rise in housing prices in Chibougamau and Chapais due to higher demand.

20.1.3 Regulatory Context

20.1.3.1 Environmental Impact Assessment Procedure

20.1.3.1.1 Provincial Authorities

The EIA procedure in the province of Québec is divided into two regimes: Southern and Northern. The Black Rock Project location falls into the Northern regime, with the provisions applicable to the James Bay region located south of the 55th parallel (EQA, Title II, Chapter II). The Black Rock Project is located in the territory covered by the James Bay and Northern Québec Agreement (JBNQA). The projects listed in Schedule A of the Environment Quality Act (EQA) are automatically subject to the EIA and review procedure. Mining projects are listed in Paragraph (a) of Schedule A:

All mining developments, including the additions to, alterations or modifications of existing mining developments.

In 2011, an environmental impact (EI) statement was submitted for the exploitation of the Southwest pit. BRM obtained its Certificate of Authorization (CoA) from the provincial government in 2013 (EIA approval). In October 2014, the project was amended to change the wording and to extend the delay of certain conditions and a modified CoA was issued in 2015. Following the decision to build a secondary process plant for the transformation of magnetite, vanadium and titanium concentrate into high purity pig iron and ferrovanadium in Saguenay, another amendment of the CoA was requested in December 2017 to adjust the project to the production rate that could be achieved by the secondary process plant, for the new tailings management strategy, to include means of transporting the concentrate (road and railway) and for other minor modifications. A modified CoA was issued in 2019.

The EIA approval contains 40 conditions. One is no longer applicable (worker camp), nine are on mitigation measures or monitoring and 30 conditions require that additional information be submitted to the *Ministère de l'Environnement et de la Lutte contre les changements climatiques, de la Faune et des Parcs* (MELCCFP) for information, or approval, within 1 or 2 years of the issuance of the last amended CoA and/or prior to the construction phase. Of those 30 conditions, 11 conditions are completed and/or approved by the MELCCFP.

BRM has submitted its wetlands compensation plans in May 2020 for approval to the MELCCFP. Comments were received in January 2021, mostly concerning mining titles (*Ministère de l'Énergie et des Ressources naturelles* [MERN]) near or at the proposed locations and commitments. BRM is presently waiting for responses from the MERN as to the acceptability of the projects located near the sites under mining titles (borrow pits and old mine site).

20.1.3.1.2 Federal Authorities

In 2011, an EI statement was submitted for the exploitation of the Southwest pit. The environmental assessment was conducted under the previous *Canadian Environmental Assessment Act* (1992). In 2014, the Canadian Environment Minister, having considered the comprehensive study report and public comments, announced that the project is not likely to cause significant adverse environmental effects when the mitigation measures described in the comprehensive study report are taken into account and that the mitigation measures and monitoring program described in the comprehensive study report are appropriate for this project.

The Minister has referred the project back to the responsible authority, Fisheries and Oceans Canada (DFO), for appropriate action. The responsible authority must ensure the implementation of the mitigation measures identified in the comprehensive study report and to implement the monitoring program described in the comprehensive study report to determine the effectiveness of the measures taken to mitigate adverse environmental effects and to verify the accuracy of the environmental assessment of the project.

BRM has submitted its mitigation monitoring plan and their federal monitoring program in 2019 to the responsible authority for approval. A second version was submitted in May 2020 and the responsible authority issued comments in December 2020. The program is being reviewed.

The tailings management facilities (TMF) area uses a watercourse frequented by fish for mine waste disposal. The amendment to add the watercourse to Schedule 2 of the *Metal and Diamond Mining Effluent Regulations* (MDMER) was completed.

20.1.4 Permitting Requirements

Throughout all stages of the Project (construction, operations, closure), activities conducted by BRM will be required to comply with provincial and federal acts and regulations. The detailed engineering and operations will take into account the conditions, mitigation measures and monitoring requirements associated with the global CoA and the federal authorization. It shall also consider all applicable environmental standards included in other relevant provincial acts, regulations, guidelines and policies. The most relevant ones are listed below. This list is non-exhaustive and is based on information known so far. Their applicability will have to be reviewed as the Project components are defined.

20.1.4.1 Provincial Jurisdiction

- Mining Act (M-13.1):
  - Regulation respecting mineral substances other than petroleum, natural gas and brine (M 13.1, r. 2).
- Environmental Quality Act (Q-2):
  - Regulation respecting the regulatory scheme applying to activities on the basis of their environmental impact (Q-2, r.17.1);
  - Regulation respecting activities in wetlands, bodies of water and sensitive areas (Q-2, r.01);
  - Clean Air Regulation (Q-2, r. 4.1);
  - Regulation respecting industrial depollution attestations (Q-2, r. 5);
  - Regulation respecting pits and quarries (Q-2, r. 7.1);
  - Regulation respecting compensation for adverse effects on wetlands and bodies of water (Q-2, r. 9.1);
  - Regulation respecting the declaration of water withdrawals (Q-2, r. 14);

- Regulation respecting mandatory reporting of certain emissions of contaminants into the atmosphere (Q-2, r. 15);
- Regulation respecting the burial of contaminated soils (Q-2, r. 18);
- Regulation respecting the landfilling and incineration of residual materials (Q-2, r. 19);
- Regulation respecting waste water disposal systems for isolated dwellings (Q-2, r. 22);
- Regulation respecting halocarbons (Q-2, r. 29);
- Regulation respecting hazardous materials (Q-2, r. 32);
- Protection Policy for Lakeshores, Riverbanks, Littoral Zones and Floodplains (Q-2, r.35);
- Water Withdrawal and Protection Regulation (Q-2, r. 35.2);
- Land Protection and Rehabilitation Regulation (Q-2, r. 37);
- Regulation respecting the quality of the atmosphere (Q-2, r. 38);
- Regulation respecting the quality of drinking water (Q-2, r. 40);
- Regulation respecting the charges payable for the use of water (Q-2, r. 42.1);

▪ Threatened or Vulnerable Species Act (E-12.01):
  - Regulation respecting threatened or vulnerable wildlife species and their habitats (E 12.01, r.2);
  - Regulation respecting threatened or vulnerable plant species and their habitats (E-12.01, r.3).
▪ Watercourses Act (R-13):
  - Regulation respecting the water property in the domain of the State (R-13, r. 1).
▪ Sustainable Forest Development Act (A-18.1):
  - Regulation respecting the sustainable development of forests in the domain of the State (A-18.1, r. 0.01).
▪ Conservation and Development of Wildlife Act (C-61.1):
  - Regulation respecting wildlife habitats (C-61.1, r. 18).
▪ Lands in the Domain of the State Act (c. T-8.1).
▪ Building Act (c. B-1.1):
  - Construction Code (B-1.1, r. 2);
  - Safety Code (B-1.1, r. 3).
▪ Explosives Act (E-22):
  - Regulation under the Act respecting explosives (E-22, r. 1).
▪ Cultural Heritage Act (P-9.002).

- Highway Safety Code (C-24.2).
  - Transportation of Dangerous Substances Regulation (C-24.2, r. 43).
- Occupational Health and Safety Act (S-2.1):
  - Regulation respecting occupational health and safety in mines (S-2.1, r. 14).
- Dam Safety Act (S-3.1.01):
  - Dam Safety Regulation (S-3.1.01, r. 1).
- Directives and Guidelines:
  - Directive 019 sur l'industrie minière (2012);
  - Lignes directrices relatives à la valorisation des résidus miniers (2015);
  - Guidelines for preparing mine closure plans in Québec (2017);
  - Guide d'intervention – Protection des sols et réhabilitation des terrains contaminés (2019);
  - Guide de caractérisation des résidus miniers et du minerai (2020).

20.1.4.2 Federal Jurisdiction

- Fisheries Act (R.S.C., 1985, c. F-14):
  - Metal and Diamond Mining Effluent Regulations (SOR/2002-222).
- Canadian Environmental Protection Act (S.C. 1999, c. 33):
  - PCB Regulations (SOR/2008-273);
  - Environmental Emergency Regulations (SOR/2003-307);
  - Federal Halocarbon Regulations (SOR/2003-289);
  - National Pollutant Release Inventory.
- Species at Risk Act (S.C. 2002, c. 29).
- Canada Wildlife Act (R.S.C., 1985, c. W-9):
  - Wildlife Area Regulations (C.R.C., c. 1609).
- Migratory Birds Convention Act, 1994 (S.C. 1994, c. 22):
  - Migratory Birds Regulations (C.R.C., c. 1035).
- Nuclear Safety and Control Act (S.C. 1997, c. 9):
  - General Nuclear Safety and Control Regulations (SOR/2000-202).
  - Nuclear Substances and Radiation Devices Regulations (SOR/2000-207).
- Hazardous Products Act (R.S.C., 1985, c. H-3).
- Explosives Act (R.S.C., 1985, c. E-17).
- Transportation of Dangerous Goods Act (1992):

- Transportation of Dangerous Goods Regulations (SOR/2001-286).
- Directives and Guidelines:
  - Environment Canada Environmental Code of Practice for Metal Mines (2009);
  - Guidelines for the Assessment of Alternatives for Mine Waste Disposal (2016);
  - Strategic climate change assessment (2020).

Following receipt of the provincial EIA approval, BRM will require several approvals, permits and authorizations to initiate the construction phase, operate and close the project. In addition, BRM will be required to comply with any other terms and conditions associated by both provincial and federals global authorizations.

Table 20-2 presents a non-exhaustive list of required approvals, authorizations, permits or licenses and their application status.

Table 20-2: List of permitting requirements

Activities	Authority	Status
General Authorization		
Closure plan - Mine	MERN	Under review
Mining lease	MERN	Will be issued following closure plan approval
Mine waste management facilities location approval	MERN	Received May, 2019
Processing plant location approval	MERN	Received April, 2019
Permanent infrastructure implantation and mine waste management facilities on public land leases	MERN	Will be issued following closure plan approval
Harmful alteration, disruption or destruction of fish habitat	DFO	Received April, 2020
Use of water frequented by fish for mine waste disposal and fish habitat compensation plan	DFO	Will be issued following the deposit of the letter of credit
Implementation of the fish habitat compensation plan	MELCCFP and MRNF	Received July, 2020
Implementation of the wetlands compensation plan	MELCCFP	Submitted May, 2020

Activities	Authority	Status
Construction Phase		
Closure plan - Site Preparation	MERN	Received September, 2018
Site preparation - Hub Area, OVB pile and Waste rock pile	MELCCFP	Received July, 2018
Mobile crusher – Hub Area and Pit Area	MELCCFP	Received April, 2019
Temporary occupancy permit - Site Preparation Hub Area and OVB pile	MERN	Must be renewed
Site preparation - Hub Area and OVB pile	MERN	Must be renewed
Tree cutting (partially completed)	MRNF	Obtained in April 2019 but no longer valid
Exploitation Starter pit	MELCCFP	-
Construction of BR-001 Concentrator Access Road - Section # 3 (5 km)	MRNF	Must be renewed
Construction temporary access – Hub Area	MRNF	Must be renewed
TMF construction: pre-operation period (including effluent Lac Denis) (the activities must begin within two years of the date of issuance of the authorization)	MELCCFP	Received May, 2020
Ditch of contact water and its retention basin (including treatment for suspended solids)	MELCCFP	To submit
Civil works - sectors 2000, 3000, 4000, 5000	MELCCFP	To submit
Concrete works and structure - sectors 2000, 4000 and 5000	MELCCFP	Received June, 2019
Concrete works and structure - sector 6000	MELCCFP	To submit
Road construction	MRNF	To submit
General construction permit	City of Chibougamau	To submit
Domestic wastewater treatment system	MELCCFP	To submit
Septic systems license	City of Chibougamau	To submit

Activities	Authority	Status
Operation / Maintenance Phase		
Mobile surface water treatment system	MELCCFP	To submit
Mine water treatment system	MELCCFP	To submit
Withdrawal of water	MELCCFP	To submit
Potable water treatment system	MELCCFP	To submit
Mining and processing plant operation	MELCCFP and MRNF	To submit
TMF operation	MELCCFP and MRNF	To submit
Industrial depollution attestation	MELCCFP	To submit
Railway construction	MELCCFP and MRNF	To submit
Tree cutting – Railway	MRNF	To submit
Construction of a railway >2km	MTQ	To submit
Exploitation of a railway >2km	CTQ	To submit
Connection to the HQ electrical network	HQ	To submit
High-risk petroleum equipment	RBQ	To submit
Use of nuclear substances and radiation devices	CNSC	To submit
Explosives possession, magazine and transportation	SQ	To submit
Explosive transportation	NRC	To submit
Explosive manufacturing plant and magazine	NRC	To submit

20.1.5 Environmental Studies

During the period spanning from 2011 to 2020, several field inventories, environmental studies, analyses, and reports have been completed to support the EIA statement. Additional studies have been carried out between 2015 and 2020 to support the Global CoA amendment and to support the authorization request for the beginning of the site preparation and construction activities. The following subsections summarize the Mining and Beneficiation Plant Site's current biophysical environmental conditions.

20.1.5.1 Physical Environment

20.1.5.1.1 Hydrography

At a larger scale, the Mining and Beneficiation Plant Site is in the Nottaway River watershed, which is part of the Hannah and Rupert Bays watershed. At the regional level, the site is located in the Villefagnan Creek watershed, which is tributary of the Armitage River that flows in direction of the Chibougamau Lake. Locally, the infrastructure is mainly located in the Jean Lake watershed, which will be to most impacted by the project.

20.1.5.1.2 Surface Water and Sediment Quality

Surface water and sediment quality were characterized in 2011 and 2012 (Entraco, 2011, Lamont, 2012). Results for the lakes and streams characterized close to the site showed high conductivity, and concentrations of alkalinity, total inorganic carbon and calcium, magnesium and sodium concentrations that are generally seen in the Canadian Shield (elsewhere in Québec) (Entraco, 2011). In the 2012 study performed by Lamont, the results were not compared to water quality criteria since many of these criteria are dependent on the hardness of the environment, which was not analyzed.

In 2013, a new characterization was completed as part of condition 16 of the Global CoA (FaunENord, 2013). Samples were collected at five stations. The results show that aluminum concentrations exceeded federal guidelines at most stations. In addition, the waters of the Lac Jean tributary show a higher concentration of bicarbonate than the other stations. Finally, total coliforms and atypical colonies exceed the thresholds specified in the Regulation respecting the quality of drinking water. As for the sediment, they were compared to the provincial soil criteria instead of the sediment criteria. Metals concentrations exceeded background level for soils in the Superior Province at several stations. The two metals exceeding background level are mercury and copper (FaunENord, 2013). In 2014, the MDDELCC (now the MELCCFP) requested a complementary surface water and sediment characterization (MDDELCC, 2014).

This complementary characterization was completed in 2019 (FaunENord, 2020a). Samples were collected at six stations for surface water and 18 for sediments. The results for surface water show median arsenic concentration exceeded the prevention of contamination of aquatic organisms criteria at one station for Bernadette Lake and Jean lake, as well as at two stations of the Villefagnan effluent. As for the sediments, metals concentration exceeded one or many sediment quality criteria at one or several stations. The main metals exceeding criteria are cadmium, copper, mercury, and zinc.

20.1.5.1.3 Soil Quality

In 2013, a soil quality assessment was completed but deemed insufficient by the MDDELCC (now the MELCCFP) (MDDELCC, 2014). A new assessment was completed in 2018. The assessment of the soil initial quality and natural background was established with 49 test pits (Englobe, 2018). These were positioned in order to constitute a statistically representative set to establish the natural background grade. A statistical treatment of the analysis results was done according to each type of soil surface deposit and for all the analysis parameters. In general, for the majority of the parameters, the variation in results proved to be not statistically significant. However, some parameters showed larger deviations in measured concentrations, especially in the organic horizon. Nevertheless, almost all of the calculated upper vibrissae are below the criterion A of the MELCCFP's Intervention Guide – Soil Protection and Rehabilitation of Contaminated Sites, with the exception of the parameters cadmium, sulphur, total cyanide and C_{10}-C_{50} Petroleum Hydrocarbons, which slightly exceed the criterion A, but only in the organic horizon. Finally, for parameters for which no detection was found, the criterion A of the MELCCFP's Intervention Guide will be the reference in terms of environmental quality of these soils (Englobe, 2018).

20.1.5.1.4 Hydrogeology and Groundwater Quality

An hydrogeology study was carried out in 2011 to establish hydrogeological baseline conditions (Entraco, 2011) and the Mining and Beneficiation Plant Site and the transhipment site. Only two hydrostratigraphic units were identified at the investigated locations, a sandy till, considered as an aquitard due to its thinness, and the bedrock whose network of fissures constitutes a potential aquifer. The average hydraulic conductivity of the till is 1.29×10^{-4} cm/s and average hydraulic conductivity of the bedrock is 1.93×10^{-4} cm/s. The average hydraulic conductivity of all the measurement points, including the wells whose screens intersect till and rock, is

1.56×10^{-4} cm/s. The measured water level is generally within 1.5 m of ground level and the flow pattern is consistent with the local topography.

Groundwater pH is generally acidic in recharge areas and more alkaline in discharge areas. The low electrical conductivity indicates that the groundwater is generally very poorly mineralized. Iron was detected only at the transhipment site (area where truckloads of concentrate will be loaded into CN railcars), with concentrations ranging from 0.1 to 26 mg/L. Of the other metals analyzed, only arsenic, copper and zinc were detected in a few wells at concentrations that barely exceeded the detection limit (Entraco, 2011).

A new groundwater characterization was conducted in 2019 (EnviroCree, 2019) as part condition 17 of the Global CoA. Twenty observation wells were sampled. The analyzed parameters concentrations are, in general, below the detection limit or their respective alert thresholds. Of the metals monitored by the Directive 019, arsenic showed exceedance for the drinking water criteria in eight observation wells, and copper for the alert threshold for resurgence in surface water in three observation wells.

In 2019, report on the completion of TMF groundwater flow simulations was completed and demonstrates that the daily percolation rate at the bottom of the accumulation area meets the Directive 019 Richelieu Hydrogéologie, 2019a). An hydrogeological modeling of the transport of dissolved contaminants that would be emitted from the TMF was completed (Richelieu Hydrogéologie, 2019b). Simulations using conservative assumptions show that in the worst case scenario, dissolved metals (cadmium, copper and zinc) could migrate in groundwater downstream of the TMF to receiving environments (streams, lakes, etc). However, the contaminant plume would progress relatively slowly. Richelieu Hydrogeology therefore recommends to add observation wells downstream of the TMF and that groundwater quality and piezometry be monitored.

20.1.5.1.5 Ambient Air Quality

An air dispersion study of contaminants emitted during the site preparation and operation phases was prepared in response to condition 26 of the CoA. The modeling results demonstrate that based on available data and production projections, the project site will meet the particulate matter emission standard per Schedule C of the Clean Air Regulations. All other modeled parameters are 100 % below of the standard. (WSP, 2018). During mine operation, the direct greenhouse gas (GHG) emissions would average approximately 29.8 kt CO_2 eq/y (WSP, 2019). BRM will not be subject to Québec's regulation respecting a cap and trade system for greenhouse gas emission allowances.

20.1.5.1.6 Ambient Noise

In the initial impact study, 23 mobile equipment were planned for mining operations and 37 mobile support equipment (Entraco, 2011). In the updated project (WSP, 2018), 18 mobile mining operations and 25 mobile support equipment are planned. The expected noise emissions will be lower than the initial operating scenario. Considering the most restrictive noise criterion of Directive 019, i.e., 40 dBA, the noise estimated during the initial impact study of the mining activities was below this threshold from a distance of 2 km. It should be noted that there are no sensitive areas within a 10 km radius of the mine. Therefore, no exceedance of the noise standard is to be expected (WSP, 2018).

20.1.5.2 Biological Environment

20.1.5.2.1 Vegetation and Wetlands

The study area is in the Western Moss Spruce Forest bioclimatic domain. The main species found are black spruce, fir, trembling aspen, balsam poplar, tamarack, jack pine, paper birch and white spruce. Deciduous trees are often found as isolated patches and along roads, and tend to be trembling aspen and white birch. The vegetation cover of the region has been disturbed by logging over the years. Since the 1950's, the extent of logging has increased steadily and is now the main agent of disturbance (Entraco, 2011).

Vegetation and wetlands inventories were carried out in 2010, 2011, 2012 (Entraco 2011 and GENIVAR, 2012) for the EIA. In 2019, the MELCCFP request additional characterization (FaunENord, 2020b). No species at risk were found during the 2010, 2011, 2012 and 2019 inventories (Entraco, 2011, GENIVAR, 2012, and FaunENord, 2020b).

20.1.5.2.2 Fish and Fish Habitat

Fish inventories and habitat description was carried out in 2011 in the various streams and water bodies located within the project footprint. No species at risk were found. All streams and water bodies were considered to be fish habitat (GENIVAR, 2012). Watercourses and waterbodies frequented by fish will be impacted and will have to be compensated (WSP, 2018). Three compensation programs were approved by both the federal and provincial authorities. One compensation project is ongoing and is being monitored.

20.1.5.2.3 Herpetofauna

No specific inventory of herpetofauna has been carried out during the EIA, only opportunistic observations. An amphibian inventory was carried out in 2019, both by acoustic recording and active research (FauNEnord, 2019). Four amphibians species were detected by the acoustic inventory. One amphibian and one urodele species were identified by the active research. A complementary inventory targeting urodeles, in particular the green newt (*Notophthalmus viridescens*) was conducted in 2020 within the TMF footprint at the request of the MELCCFP (FaunENord, 2020c). No green newts or other urodele species were found during the surveys, either as adults or larvae. Two amphibians species and one snake species were identified during the field work. In both the 2019 and the 2020 inventories, no species at risk were found.

20.1.5.2.4 Wildlife and their Habitats

There were no specific inventories carried out during the EIA. General information was gathered through opportunistic observations, as well as the tallyman's observations. In 2014, the MRNF requested from BRM the confirmation of the presence of moose (*Alces alces*) winter habitat on its property, as indicated by the tallyman. A heliborne inventory was carried out, confirming the presence of moose winter habitat near the pit and a little further east of the center of the mine (FaunENord, 2014). At the time of the survey, only three moose were observed. The inventory also showed that the territory is used by wolves. The impacted moose winter habitat will be compensated.

The Mining and Beneficiation Plant Site is outside the area of application of the recovery plan for woodland caribou.

20.1.5.2.5 Micromammals

An inventory was carried out in 2019 (FauNEnord, 2019). A total of 46 specimens were captured during, all stations and all types of traps combined. One species at risk, the rock vole (*Microtus chrotorrhinus*), was captured near the entrance of the Mining and Beneficiation Plant Site.

20.1.5.2.6 Chiropterans

An inventory was carried out in 2019 to cover both migratory and reproduction periods (FauNEnord, 2019). A total of 43 sonograms were collected through the acoustic inventory. Only one sonogram contained chiropteran echolocation calls, across the two periods and all five stations. The only bat species identified in this sonogram was the silver bat (*Lasionycteris noctivagans*) which is a species at risk.

20.1.5.2.7 Birds

Bird observations were carried in 2011 (Entraco, 2011), followed by a specific inventory in 2012 and (GENIVAR, 2012). From the exhaustive inventory of 2012, 13 waterfowl species, 7 species of birds of prey and 55 land bird species were identified. Of those, five are species of birds at risk (GENIVAR, 2012). New nesting platforms to mitigate the impact on offsprey nesting sites will be installed.

20.1.5.2.8 Other Observations

A Yellow-banded Bumble Bee (*Bombus terricola*), a species of insect at risk, was observed by technical staff during chiropteran inventory on August 24, 2019 (FaunENord, 2019).

20.1.5.3 Human Environment

20.1.5.3.1 Land Planning and Development and Land Use

The Mining and Benefication Plant Site is in the Northern Québec administrative region (Region 10), on the territory of the EIJB Regional Government. The closest municipality is Chibougamau about 60 km north-west and Chapais is about 80 km west.

The Mining and Beneficiation Plant Site is located on the territory covered by the James Bay and Northern Québec Agreement (JBNQA) signed in 1975 between the Governments of Canada and Québec, the Grand Council of the Crees and the Association des Inuits du Nouveau-Québec. The land regime defined in the JBNQA is a determining factor in land use. It provides for the division of the James Bay territory into Category I, II and III lands. The site is located on Category III lands on which the First Nation of Oujé-Bougoumou has ancestral hunting, fishing and trapping rights.

No federal land is located within the Mining and Beneficiation Plant Site area. Therefore, no federal land will be used for the implementation of the infrastructure.

There are no permanent residences near the proposed Mining and Beneficiation Plant Site.

There are no current or planned protected areas within the Mining and Beneficiation Plant Site footprint. The closest protected areas are a designated biological refuge (21 km) and the exceptional forest ecosystem of Portage-Mackenzie (16 km) both located south-west of the site, south of Road 210.

20.1.5.3.2 Population and Economics

The population of the EIJB Regional Government was estimated at 18,347 people in 2020 (ISQ, 2021a). With 7,405 inhabitants (2020), Chibougamau has the largest population in the region, while Chapais has a population of 1 551 (2020) (ISQ, 2021b).

Median total income of individuals 18 years of age and over is $45,245 in Chibougamau and $40,253 in Chapais (ISQ, 2021c).

The economy revolves essentially around three resources: mines, forestry and tourism.

20.1.5.3.3 Archaeology Heritage

The archaeological potential was assessed in 2011 to identify areas where traces of human occupation might be found (Entraco, 2011). About 15 potential areas have been identified between Route 167 and Chibougamau Lake. All areas of archaeological potential likely to be affected by mining activities were inventoried in 2003 and subsequently in summer 2011 (Entraco, 2011). A total of 550 test pits were carried out. There have been no findings of archaeological interest (Entraco, 2011). The only known archaeological site in the study area is located on the eastern shore of Chibougamau Lake, south of Île des Commissaires (Entraco, 2011).

20.1.5.3.4 First Nations

The closest First Nations communities are Oujé-Bougoumou Cree Nation to the west and Cree Nation of Mistassini to the north.

The Mine and Beneficiation Plant Site lies on trapline O-59, which is managed by the Wapachee family from Oujé-Bougoumou, who has a hunting camp near Road 210. The family has concerns regarding the amount of land available to continue their traditional hunting, trapping, fishing and gathering activities. Discussions with the family are ongoing with respect to relocating the hunting camp. The northeastern end of the mining area (Laugon Lake sector) corresponds to trapline O-57/M-57.

The Innu of Lac-Saint-Jean (Pekuakamiulnuatsh) have ancestral rights (trapline 24) over part of the territory located in the St. Lawrence watershed (Saguenay and Saint-Jean Lake watershed). In 2011, updated information confirmed that trapline no. 24 has not been used for few years for traditional activities for (Entraco, 2011). No sites of interest have been identified on this territory and no particular land use designation (heritage site, Innu park, freehold land) has been identified as part of the land use negotiations (Entraco, 2011).

20.1.5.4 Environmental and Social Impacts

20.1.5.4.1 Anticipated Impacts

This section summarizes the main social and environmental anticipated impacts associated with the development of the project, as identified in the EIA statement submitted in 2011 (Entraco, 2011). Although this list is not exhaustive, it underlines topics that will require specific consideration. Some impacts were revaluated as part of the CoA amended process (WSP, 2018). Although the project site and the main components have the same overall footprint, mining equipment is smaller and less numerous. Some equipment has also been relocated, with no impact on the environment (WSP, 2018).

Only the anticipated impact and benefits for the most sensitive components (for which the impact or benefit is moderate or higher) are presented below. However, with the mitigation measures that will be implemented and the compensation plans, the residual anticipated impacts on the physical, biological and social environment are all considered minor or not significant.

Physical Environment:

- Surface water and soil quality alteration (suspended solids, accidental spills, effluents);
- Alteration, disruption and destruction of watercourses and water bodies;
- Deterioration of air quality through suspended dust and GHG emissions.

Biological Environment:

- Wetlands loss;
- Wildlife and bird habitats loss;
- Fish habitat loss;
- Increase in fishing pressure by the workers;
- Increase in risk of collisions with wildlife.

Social Environment:

- Increased noise levels in the area of the Mining and Beneficiation Plant Site, the transhipment station and the access road;
- Increased traffic of heavy vehicles and workers on local and regional roads and risk of road accidents;
- Loss of territory to perform traditional activities;
- Repercussions on land use by communities and other industries;
- Noise, dust and airborne contaminants;
- Introduction of perceptible anthropogenic structures in the Chibougamau Lake landscape;
- Loss of jobs and reduced purchasing in the region during at closure.

The anticipated residual benefits of moderate significance or higher relate to:

- Job creation;
- Economic benefits for local and regional suppliers.

The following enhancement measure will be taken:

- Implementation by the company and the community of personnel training programs to meet the needs of the project;
- Set up a team of Cree trainers with similar experience on mining projects;
- Establishment of interaction mechanisms between the employment centers of the various communities in the region and BRM;
- Preferential hiring of local workers and contractors;
- Concertation table to develop and follow up on the economic benefits optimization strategy.

20.1.5.4.2 Mitigation Measures

In conjunction with the 40 conditions from the CoA, BRM made the commitment to implement over 259 mitigation measures to attenuate the negative anticipated impacts mentioned in the previous section.

20.1.5.4.3 Cumulative Effects

Cumulative effects are changes to the environment that are caused by an action in combination with other past, present and future human actions. Consideration of cumulative environmental impacts is an essential component of any environmental assessment. Seven Valued Ecosystem Components (VEC) were selected for the cumulative effects' assessment (WSP, 2018):

- Lakes and streams;
- Vegetation, wetlands and species at risk;
- Birdlife and species at risk;
- Traditional use of the territory;
- Use of the environment for the exploitation of other resources;
- Economy and employment;
- Transport of workers and concentrate.

To assess the cumulative effects of the project, the following activities were considered to be additional to those at the BRM project site: mining activities, forestry activities and other major projects (e.g., Wind farm projects, power lines, industrial site development, intermodal transhipment, etc.).

Cumulative effects were re-evaluated taking into consideration the project's modifications of the amended CoA (WSP, 2018, 2019). Table 20-3 summarizes the cumulative effects on the VECs selected for the Mining and Beneficiation Plant project. Considering that the Project will not result in significant cumulative effects, no additional mitigation measures are deemed necessary to reduce cumulative effects.

Table 20-3: Summary of the cumulative effects

Valued Ecosystem Component	Global Effect
Lakes and streams	Not significant
Vegetation, wetlands and species at risk	Not significant
Birdlife and species at risk	Not significant
Traditional use of the territory	Significant, but mitigated
Use of the environment for the exploitation of other resources	Not significant
Economy and employment	Positive
Transport of workers and concentrate	Not significant

20.1.6 Closure Plan

Under the *Mining Act*, a person who performs prescribed exploration or mining work must submit a closure plan for the land affected by their operations, subject to approval by the MERN and is conditional upon receipt of a favourable decision from the MELCCFP. This approval is required for the release of the mining lease and the mining operations to begin (including the construction phase).

The main objective of a mining closure plan is to return the site to an acceptable condition. Protection, reclamation and closure measures that will be presented will aim to return the site to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring public safety;
- Limiting the production and spread of contaminants that could damage the receiving environment and, in the long term, aiming to eliminate all forms of maintenance and monitoring;
- Returning the site to a condition in which it is visually acceptable (reclamation);
- Returning the infrastructure areas (excluding the tailings impoundment and waste rock piles) to a state that is compatible with future use (rehabilitation).

An amendment to Section 111 of the *Regulation respecting Mineral Substances other than Petroleum, Natural Gas and Brine* was made in 2013 (Decree 838-2013). Thus, mining companies must now provide a financial guarantee. This financial guarantee ensures that funds will be available to carry out the work provided for in the closure plan in the event of default by the proponent. It covers the entire cost of land rehabilitation and reclamation work for the entire mine site as provided for in the closure plan.

Moreover, in November 2017, the MERN published the Guidelines for the preparing mine closure plans in Québec. A detailed breakdown of the dismantling cost for all infrastructure built on-site must now be provided and the engineering and supervision fees (indirect costs) have been fixed to a minimum of 30% of the direct cost (conceptual stage), which include the post-restoration monitoring. A mandatory contingency of 15% must be added to the estimated cost. The proponent who engages in mining operations must pay the financial guarantee according to the following terms:

- The guarantee must be paid in three installments;
- The first payment must be made within 90 days of receiving the plan's approval;
- Each subsequent payment must be made on the anniversary of the plan's approval;
- The first payment represents 50% of the total amount of the guarantee, and the second and third payments represent 25% each.

Total guarantees for the Mine and Beneficiation plant in Chibougamau is estimated at $60.15 M, including the direct and indirect costs (30% of direct cost), and a 15% contingency. This cost includes site rehabilitation and restoration as well as the post-restoration monitoring. The guarantee must remain in effect until the certificate of release provided for in Section 232.10 of the Mining Act has been issued.

All mine buildings and infrastructure will be dismantled, including water management infrastructure, a railway section of 1.7 km, electrical and support infrastructure, unless it is shown that they are necessary to achieve and maintain a satisfactory condition, or to support the area's socio-economic development.

The access to the open pit will be permanently closed by the construction of a 2 m high embankment made of waste rock and with a ditch in front. The open pit will be filled gradually by groundwater and precipitation. A spillway will be constructed to control any overflow by safely directing it to the environment.

All areas affected by mining operations will be revegetated to control erosion and to return the site to a natural appearance integrated in the surrounding topography and landscape.

Before the revegetation of the affected areas, a characterization study certified by an expert authorized under section 31.65 of the EQA must be submitted to the regional branch of the MELCCFP. If the study reveals the presence of contaminants in a concentration exceeding the regulatory limit values, a land rehabilitation plan must be submitted for approval.

The accumulation areas (tailings and waste rock) will be reclaimed to ensure geotechnical stability and to prevent acid mine drainage (AMD) and contaminated neutral drainage (CND).

The reclamation objective of the Tailing Management Facility (TMF) is to minimize water infiltration into the tailings to reduce the volume of leachate produced, and therefore the amount of CND that could flow out. The preliminary design calls for a monolayer cover with low permeability materials that would increase surface runoff and reduce water infiltration into the tailings. As a preliminary concept, a compacted 300 mm thick layer of 0-20 mm aggregate has been planned. The aggregate will be crushed waste rock. This low permeability cover will be capped with a 150 mm thick layer of soil suitable for revegetation followed by hydroseeding to ensure better integration with the landscape and to provide erosion protection. The overall slope of the tailings surface cover will be greater than 1% to maximize surface runoff. The slope and crest of the dykes will also be revegetated. Progressive reclamation of the TMF will be done during operation of the mine.

For the waste rock pile, as they do not produce AMD or CND, the reclamation method will consist of a 150 mm thick layer of soil suitable for revegetation will be placed at the top of the benches and by a 300 mm thick layer on the slopes. Conventional revegetation by hydroseeding will be carried out to ensure better integration with the landscape and to provide erosion protection.

The overburden stored will be used to restore the waste rock pile, the TMF and all of the other affected areas. The rest of the overburden pile will be levelled to blend with the surrounding topography.

A post-closure monitoring and maintenance program will have to be carried out to ensure the physical stability of all infrastructure and the effectiveness of any remedial measures applied at the site. The post-closure monitoring and maintenance program will include:

- A physical stability monitoring and maintenance program;
- An environmental monitoring program;
- An agronomical monitoring program.

Closure work must begin within three years of the cessation of operations. A certificate of release may be issued when:

- The MERN is satisfied that the closure work has been completed in accordance with the closure plan approved by the MERN, and no sum of money is due to the MERN with respect to the performance of the work;
- The MERN is satisfied that the condition of the land affected by the mining operations no longer poses a risk for the environment or for human health and safety;
- The MERN receives a favourable decision from the MELCCFP;
- The certificate of release relates only to the obligations under the Mining Act and does not release a person from the obligations under the EQA and its regulations.

20.1.7 Site Water Management

20.1.7.1 Site Water Management

During the pre-construction phase, runoff will be directed to a sedimentation basin.

20.1.7.2 Perimeter Ditch Network

The water from the tailings management facilities and waste rock piles or its resurgence will flow into the network of ditches surrounding the property. All the water will end up flowing to a monitoring point downstream from the property.

20.1.7.3 Domestic Wastewater

Domestic wastewater treatment units will be installed as required at the concentrator. These treatment plants will be located close to the target facility. The waste water will be treated using a membrane bioreactor, and the treatment sludge will be collected on a regular basis by a specialized supplier.

20.1.7.4 Pit Water

The water pumped from the pits will consist of groundwater and precipitation that falls on the footprint of the pit, which will be directed to the tailings pond. The amount of groundwater to be pumped will vary depending on pits depth.

20.1.7.5 Process Water

BlackRock optimized water recirculation in an effort to minimize the quantity of fresh water pumped from the environment. Process water will mainly consist of water from the thickener overflows and the tailings pond. Aside from the recirculated water in the process water tank, the water required to meet processing needs will come from the polishing pond. This water will be pumped to a reservoir that is located next to the processing plant.

20.1.7.6 Polishing Pond Water Treatment Unit

The treatment unit will be located at the polishing pond and is sized to treat a flow of 20,000 m^3/d, thus treating water from peak flows during the spring thaw or heavy rains. The water will be transferred from the tailings pond to the polishing pond by pumping. The tailings pond will also be linked to the polishing pond through an emergency spillway.

The polishing pond will be equipped with an emergency spillway allowing the water to flow into the monitoring pond. The treatment unit is designed to precipitate the suspended solids through the addition of polymers and coagulants. Sludge from the treatment unit will be pumped as needed and sent to the tailings pond. Given that this sludge consists of agglomerated fine particles and not metallic precipitates, no impact from the re-dissolution of metals or other parameters is anticipated.

20.1.7.7 Mining Effluent

Before being released into the environment, groundwater from the pit and runoff from mine site accumulation areas will be treated to meet the water quality criteria of Directive 019 and, to the extent possible, the effluent discharge objectives (EDO) defined by the MDDEFP. The treated water will be discharged into the stream bed upstream from Lac Jean. The flow rate will vary depending on the time of year, with less discharge in winter and during low flow periods. During the construction years, this stream will be dry because the water from the polishing pond will be pumped to the tailings basin in preparation for plant start-up. There will therefore be no effluent during the construction years, and Lac Jean will be fed by other streams.

20.1.7.8 Drinking Water Supply

The mine site drinking water will be supplied by wells. The treatment process includes filtration, chlorination and UV sterilization. Drinking water from wells will be stored in a tank and distributed for needs in all required units and service areas.

20.1.8 Tailings and Waste Rock Disposal

20.1.8.1 Tailings

20.1.8.1.1 Geochemistry of the Tailings

Short-term static testing methods were used to assess the chemical composition of the tailings, the acid rock drainage (ARD) potential and metal leaching (ML) potential. The long-term weathering characteristics of the tailings have been evaluated through kinetic weathering tests.

An initial Tailing geochemical characterization study was completed in 2013 (Lamont Inc., 2013). Static and kinetic tests were conducted on two samples (fine and coarse), collected by SGS from the metallurgical process, were initiated in 2012. Kinetic tests were done by using two humidity cells over a period of 20 weeks (SGS). The potential of the tailings to generate ARD was evaluated through Modified Acid Base Accounting (MABA) method. The static tests were interpreted by using both the Directive 019 and MEND (2009) guidelines to classify the ARD potential. Using those guidelines, the samples are classified as potentially acid generating (PAG). As for the ML potential, based on the Toxicity Characteristic Leaching Procedure (TCLP) method, the samples are leachable. None of the samples are classified as high risk. Kinetic tests were then performed to obtain more representative results. The results indicate that the samples can be qualified as non-acid generating and non-leachable.

Since the geochemical characterization was based on only two samples, the MELCCFP and MERN has requested a more complete geochemical characterization (ARD potential, total metals content and leachability potential (TCLP, SPLP, CTEU-9) and kinetic tests.

In 2019, a geochemical characterization was completed by WSP (2019) on 15 tailings samples from a bulk operation carried out in 2015 for a demonstration plant project. Following BRM decision to manage the two types of tailings together, the samples in this geochemical characterization represent a mix of fine and coarse tailings prepared for rheology testing. The samples were selected, collected and sent by BRM employees to the laboratories in December 2018. The samples were analyzed by Techni-Lab S.G.B. Abitibi Inc./Actlabs and AGAT. The potential of tailings to generate ARD was evaluated through the Modified Sobek method. The static tests were interpreted by using both the Directive 019 and MEND (2009) guidelines to classify the ARD potential of the tailings. Using the MEND guidelines, all samples are classified as PAG, and using the Directive 019, one of the 15 samples is classified as PAG. As for the ML potential, the 3 of the 15 samples are leachable. None of the samples are classified as high risk.

In 2020, three column kinetic tests were carried out by the *Unité de recherche et de service en technologie minérale* (URSTM) on the samples from 2019 to confirmed the ARD and leachability potential from the static tests. The choice of composites was based on the results of the static tests (WSP, 2019), mainly on the sulfur-sulphide contents. The extrapolations of the oxidation and neutralization reaction data suggest that the three composites would not produce long-term ARD under the conditions of the column tests. The tailings are leachable, mainly to copper (Cu) and zinc (Zn), thus contaminated neutral drainage (CND) generating, which will have to be monitored.

URSTM suggest that a mineralogical characterization by X-ray diffraction and optical and electron microscopy would allow the identification of the minerals carrying the elements that were mobilized during the kinetic tests (Cu, Ni, Zn), as well as the sources of Ca, Mg and Mn, and thus the nature of the neutralizing minerals. These data would allow to better target the risks of CND and to suggest strategies to control the risk of water contamination. In addition, intermediate scale field tests (barrels, test plots) at the mine site would allow the mobility of metals to be assessed under more realistic scale conditions as well as under site weather conditions.

Tailings could be considered "acid generators" if the new MELCCFP requirements are used (to be confirmed by geochemistry specialist).

20.1.8.1.2 Tailings Management

At the tailings management facility, the water in the pond will consist of a mixture of process water, mine water and rainwater falling on the surface of the pond. This water will be transferred by pumping into the polishing pond for a second round of settling. Water will be pumped from the polishing pond to the concentrator to be reused.

The tailings management facility will be located west of the pit where dams will be built around most of the perimeter. The dams will have a maximum height of under 30 m as to hold about 55 Mm^3 of wet tailings (water and solids) after 43 years of operation. The tailings deposited in the pond will be flooded to eliminate a source of dust emissions. The aqueous phase on top of the tailings will be transferred to the polishing pond and then recycled to the concentrator or released into the environment through the monitoring pond, which will allow the standards for mining effluents to be met.

20.1.8.2 Waste Rock Management

20.1.8.2.1 Geochemistry of the Waste Rock

Short-term static testing methods were used to assess the chemical composition of the Waste Rock, the acid rock drainage (ARD) and metal leaching (ML) potential. The long-term weathering characteristics of Waste Rock have been evaluated through kinetic weathering tests.

An initial Waste Rock geochemical characterization study was completed in 2013 (Lamont Inc., 2013). Static tests were conducted on 101 samples. These samples were selected proportionally from all lithologies. They were identified from 26 geological sections covering 2.6 km from the Southwest Pit. The potential of the Waste Rock to generate ARD was evaluated through Modified Acid Base Accounting (MABA) method. The static tests were interpreted by using both the Directive 019 and MEND (2009) guidelines to classify the ARD potential. Using those guidelines, the samples are classified as non-PAG. As for the ML potential, based on the TCLP method, 32 % the samples are leachable. None of the samples are classified as high risk. Since Waste Rock is non-PAG, Lamont assumed the material had a low leaching potential under atmospheric exposure conditions, thus a low risk of CND.

In 2016, a humid cell kinetic test was conducted as part of a university research project (Lévesque Michaud, 2016). The objective of the thesis was to develop a test protocol to better predict CND by comparing the effectiveness of different methods to force metals to leach under certain conditions. The Waste Rock, not being PAG, was well suited to this type of testing. Therefore, static tests and kinetic tests under different conditions were carried out in order to compare different protocols on the quality of leachate water. In 2019, Lamont prepared a technical note based on this research project to present additional interpretation of the Waste Rock ML potential. Lamont concludes that since the metal concentrations obtained during rinses with complexing agent solutions all show a decreasing trend and stabilize rapidly after a few weeks, indications are that the initial metal concentrations of the waste rock samples are too low to generate DNC. This technical note was presented to the MERN in 2020 and no further questions concerning the leachability of Waste Rock was asked.

20.1.8.2.2 Waste Rock Storage – from the Southwest Pit

Waste rock will be stored on a pile to be built east of the Southwest pit. The overall slope of the pile will be 22 degrees for a maximum elevation of 655 m, which means a maximum thickness of about 155 m of waste rock. The Armitage pit approximately 150 ha for a maximum volume of approximately 117.4 Mm^3 of stored waste rock.

20.1.8.2.3 Recovery, Recycling and Disposal Methods

Debris generated during the construction, operation and closure phases will be disposed of at the Chibougamau municipal landfill, a MDDEFP-approved site. Recycling and reuse of materials will be encouraged.

20.1.9 Tailings Pond Construction Plan

The tailings pond will be composed of four cells and it will be built in stages, over the years. In preproduction, the cell no.1 dykes will be constructed to an elevation of 421.0 m, allowing sufficient space for roughly 2.6 Mm^3 of capacity. The cell no.1 dykes will be progressively raised to the final elevation of 437.5 m followed by the progressive construction of the remaining three cells such that it meets or exceeds the schedule shown in Table 20-4.

Table 20-4: Dyke construction schedule

Construction Completed by Year	Level (m)	Cumulative Tailings Capacity (Mm^3)
Cell no.1, Start-up	421.0	2.6
Cell no. 1, Year 10	437.5	15.2
Cell no.2, Year 19	437.5	26.4
Cell no. 3, Year 27	437.5	37.2
Cell no.4, Year 42	437.5	55.1

20.2 Metallurgical Plant

BlackRock will have one of the lowest environmental footprints in the industry. This will be achieved by minimizing environmental impacts, using renewable electrical energy and natural gas instead of coal, by having a zero waste strategy, by making products that can be recycled and will significantly decrease the Green House Gas footprint of the steel industry, from the product environmental lifecycle assessment.

20.2.1 Study Areas

20.2.1.1 Restricted Study Area

The restricted study area, showed in map 1.1, covers an area of some 73.3 ha (0.733 km²) and includes all the proposed BlackRock infrastructure. This area for the processing plant and all of its components is part of the industrial site of the port of Grand-Anse, in the town of Saguenay.

Bordered to the north by the Quai Marcel-Dionne road, it targets Lots 5 646 311, 4 012 436, 4 012 439 and part of lots 4 242 207 and 4 242 269. With a width of 600 m, the site stretches to the south on a length varying from 940 m to 1460 m.

20.2.1.2 Local Study Area

The local study area, shown on map 1.1, intersects the immediate area surrounding the proposed plant site and includes the scope for the proximity impacts, such as community nuisance, that can reasonably be expected. The choice of the zone allows a good comprehension of the Project insertion in the human component. The local study area covers an area of 5,351 ha (53.51 km²). Its Northern limit follows the south shore of the Saguenay River from the eastern end of Grande Anse to a small cove east to the 'Pointe à Gonie'. Its southern limit is based on the wooded summits of the Cap-à-l'Ouest Peninsula and joins to the west the intersection of Chemin de la Grande-Anse and Boulevard de la Grande-Baie Nord (Route 372).

20.2.1.3 Regional Study Area

The regional study area situates the project in its socio-economic and geographical context it covers the territory of the Municipality of Saguenay. This territory is considered because the majority of the socio-economic benefits of the project are likely to be regional in scope.

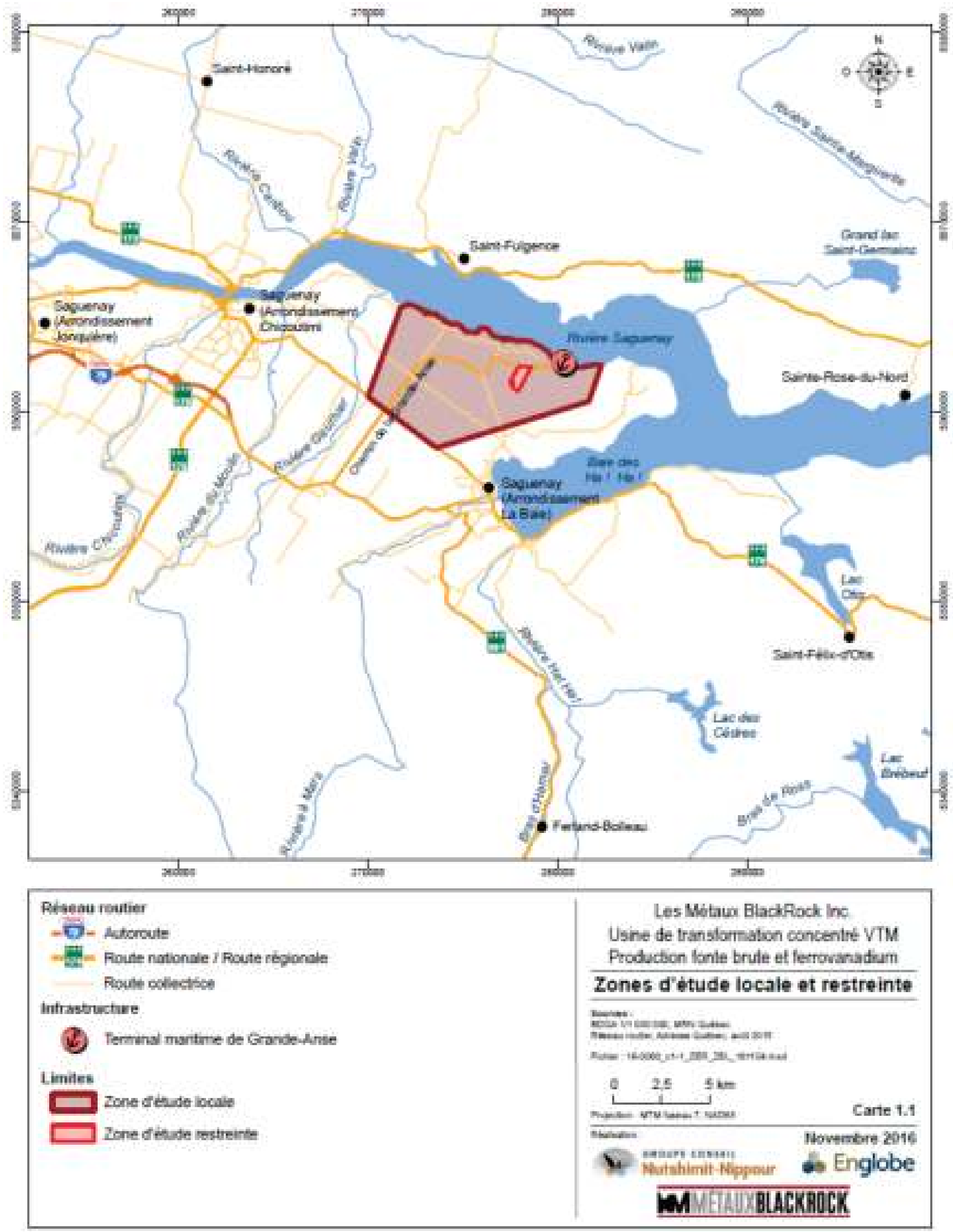


Figure 20-1: Study area

20.2.2 Social Relations

BlackRock committed to interacting and cooperating with the communities in the vicinity of its project. To this end, BlackRock met with several groups and has had discussions since December 2016 to include the concerns of the local stakeholders in the Project design. This also meant for BlackRock to have mitigation measures focused on local concerns and to have a better control over the impacts of the Projects.

A number of issues and concerns emerged through stakeholder consultation activities. In general, environmental concerns are distinguished into two categories: impacts affecting the neighborhood more directly (traffic, dust, respiratory health, ambient noise, quality of life, market value of homes) and impacts associated with issues of greater concern on a regional scale (landscape, greenhouse gas emissions, jobs, closure phase).

Several questions were asked about the plant's production process and its possible effects on the environment. Responses to these questions were generally well received because they provided stakeholders with a better understanding of the plant's environmental impacts.

With respect to First Nations, the general concerns were about rail and marine transportation as well as jobs. BlackRock is committed to keeping communities informed about these activities and supporting them in their efforts, where possible. BlackRock will also promote partnerships, including training programs, with First Nations in collaboration with the Innu communities of Mashteuiatsh, Essipit and Pessamit.

Many discussions are ongoing in the regions with various groups of interest: municipal, economical, environmental and social. BlackRock signed in 2018 an agreement with CMAX (Regional Economic Development Group) to maximize economic benefits to the local economy.

BlackRock signed an Impact and Benefits Agreement (IBA) in August 2020 with three Innus Nations, First Nation of Pekuakamiulnuatsh (community of Mashteuiatsh), First Nation of Pessamit, First Nation of Essipiunnuat (Essipit). Regular meetings of the Joint Committee and the Employment and Training Committee have been held since the fall of 2020.

20.2.3 Permitting

The project falls under the southern Québec regime. The purpose of the Environmental Assessment is to allow the relevant regulators to properly assess the impact of the project and to seek input from local stakeholders on the proposed development.

In this context, BlackRock submitted a Project Notice and received the Guideline issued by the BAPE via the MDDELCC.

An EIA (Environmental Impact Assessment) was filed with the MELCCFP under the southern regime, since the selected site is in Saguenay. Public Hearings were held in 2018 and a Decree (Certificate of Authorization) was received in April 2019 for the construction of the metallurgical facility. Once the provincial administrators have issued authorizations for project development, final permits will be sought from the MELCCFP, the MERN, and all relevant municipal authorities.

20.2.3.1 Provincial permits (Preliminary list)

Certificates of authorization are required from the MELCCFP under section 22 of the EQA for the following elements of the BlackRock Project.

- Certificate of authorization for construction of required infrastructure;
- Certificate of authorization for construction of access roads;
- Certificate of authorization for production;
- Certificate of authorization for the scrubbers and dust collector;
- Certificate of authorization for pit and quarry operation (if required);
- Certificate of authorization for a mobile concrete plant (if required).

Authorizations are required from the MELCCFP under section 32 of the EQA.

- Authorization for effluent treatment facilities;
- Authorization for water/oil separation system.

20.2.3.2 Ministère des Ressources Naturelles et des Forêts (MRNF) (Ministry of Natural Resources and Forests)

- Forest management permits are required under section 20 of the Forest Act (R.S.Q., c.F-4.1) for deforestation of the required sites.This permit was obtained and work is completed.

20.2.3.3 Municipal requirements

- A certificate of conformity to regulation from the selected site Municipality is required in order to submit authorization requests under section 32 of the Environment Quality Act (R.S.Q., c. Q-2) and certificates of authorization requests under section 22 of the Environment Quality Act (EQA);
- Construction permits are required from the Municipality before starting construction work.

20.2.3.4 Permits

Here is a preliminary list of planned permits and permits obtained.

Table 20-5: Permit list

Item	Permit Description	Site	Authority concerned	In charge of inputs	Filing request date	Expected date to obtain permit	Status
U-0	Environmental Authorization (EIA) Decree	Met. Plant	MELCCFP	BRM Project		2019-04-03	obtained
U-1	Application for a building permit and related buildings to the City of Saguenay	Met. Plant	City of Saguenay	BRM Project	2023-02-01	2023-05-01	
U-2	Certificate of authorization for tree cutting	Met. Plant	City of Saguenay	BRM Project	2019-03-01	2019-05-08	obtained
U-3	Application for a grid connection license - water supply	Met. Plant	City of Saguenay	BRM Project	2024-06-01	2024-07-01	
U-4	Application for a septic installation permit	Met. Plant	City of Saguenay	BRM Project	2023-06-15	2023-09-01	
U-5	CA application for tree cutting of the site	Met. Plant	MELCCFP	WSP / BRM	2019-03-06	2019-05-08	obtained
U-6	CA application for Excavation, Backfill and Excavation	Met. Plant	MELCCFP	WSP / BRM	2019-04-27	2023-05-01	suspended for payment of financial contribution
U-7	CA application for crushing and sieving	Met. Plant	MELCCFP	WSP / BRM	2019-04-27	2023-05-01	suspended for payment of financial contribution
U-8	CA application for the preparation of concrete for the building of the OSBF and temporary building installations of the processing plant	Met. Plant	MELCCFP	WSP / BRM	2023-02-15	2023-05-01	
U-15	Underground services and temporary sanitation facilities	Met. Plant	MELCCFP	WSP / BRM	2023-02-15	2023-05-01	
U-9	CA application of all the foundations and structures of the plant	Met. Plant	MELCCFP	WSP / BRM	2023-06-15	2023-09-01	
U-10	CA application for the installation of the permanent sedimentation basin	Met. Plant	MELCCFP	WSP / BRM	2023-04-01	2023-07-01	
U-11	CA application - Construction of roads and drainage ditches	Met. Plant	MELCCFP	WSP / BRM	2023-09-15	2023-12-01	
U-12	CA Application - Septic Tank Installation	Met. Plant	MELCCFP	WSP / BRM	2023-06-15	2023-09-01	
U-13	CA application - Exploitation - Authorization for the construction of the EQA (may include equipment, electricity, pipelines, product rental, residual materials, dust collectors, scrubbers, etc.)	Met. Plant	MELCCFP	WSP / BRM	2023-07-01	2024-07-01	

20.2.4 Environmental Studies

During the period spanning from 2016 to 2018, several field inventories, environmental studies, analyses, and reports have been completed to support the EIA statement. Additional studies have been carried out between 2018 and 2021 to support the authorization request for the beginning of the site preparation and construction activities. The following subsections summarize the Metallurgical Plant current biophysical environmental conditions.

20.2.4.1 Physical Environment

20.2.4.1.1 Hydrography

The site is located approximately 1,300 m south of the Saguenay River. An intermittent stream characterized by small watersheds flows just south of the study site towards the Saguenay and will potentially be affected by the project.

20.2.4.1.2 Surface Water and Sediment Quality

Surface water and sediment chemistry analyses were performed at three stations located in the intermittent streams flowing through the project site. The environmental quality criteria were generally met at all three sites.

20.2.4.1.3 Soil Quality

Environmental site assessment studies (Phase I, II and III) were completed from 2017 to 2020 by Englobe to establish the baseline state of soils and past uses for the study area. The northwestern portion of the site was used as agricultural land until the 1980s and the rest of the site remained vacant with the exception of the presence of a building and a small area used as a road salt storage area. Phase II and III characterisation revealed presence of about 2 700 m^3 of soil with conductivity values (salt parameter) above the MELCCFP maximum acceptable limit for an industrial site and is not compatible with the current and intended use of the property. It is planned to carry out the excavation of the soil at the site under study as part of the construction of the plant. The soil that is elevated in conductivity values will be removed in accordance with a rehabilitation plan approved by MELCCFP.

20.2.4.1.4 Hydrogeology and Groundwater Quality

The hydrostratigraphic units present on the site have a low aquifer potential since their permeability does not allow the exploitation of groundwater from an economic point of view. They are considered Class III according to the Québec Groundwater Classification Guide.

An hydrogeology study was carried out in 2020 by Englobe to establish hydrogeological baseline conditions. The results of the chemical analyses carried out on the groundwater samples taken from three observation wells demonstrated exceedance of the MELCCFP criteria for metals (Ag, Ba, Cd, Cu, Mn and Zn) and for chlorides. As the criteria for chloride parameter is directly related to road salt storage activities, Englobe recommends groundwater monitoring, three times per years (spring, summer and fall) for the observation wells installed in this mandate.

20.2.4.1.5 Ambient Air Quality

Three ambient air monitoring stations are part of the MELCCFP ambient air quality monitoring network in the vicinity of the project. These stations are located at a distance of 6 to 22 km from the project site and allow various parameters to be monitored continuously (O_3, $PM_{2.5}$ and SO_2) or sequentially (PST and PM_{10}). Air quality statistics in Saguenay indicate that there are on average only six days per year when the air quality index (AQI) is considered poor (MELCCFP, 2016).

Emissions and atmospheric dispersion from plant operation were modelled to ensure compliance with MELCCFP air quality standards of the Clean Air Regulations. The contaminants selected for atmospheric dispersion modelling are suspended particulate matter, total particulate matter (TMP) and fine particulate matter (PM2.5), as well as the following gases: carbon monoxide (CO), nitrogen dioxide (NO^2) and sulphur dioxide (SO^2). The results of the modelled operating scenario indicate that all MELCCFP standards will be met (WSP 2018).

20.2.4.1.6 Ambient Noise

The residual noise levels currently existing in the study area around the future site have been surveyed in order to determine the baseline state of the site. Four measurement locations were selected inside the inventoried residences in order to characterize the existing residual sound climate. Noise modelling studies completed for construction and operation (WSP 2018) confirmed compliance with MELCCFP noise standards.

20.2.4.2 Biological Environment

20.2.4.2.1 Vegetation and Wetlands

The project is carried out in the bioclimatic domain of the yellow birch fir, subdomain of the East. This area is a transition zone between the deciduous forest and the boreal forest. The territory of the restricted study area is 61% (44.9 ha) covered by wooded areas and 39% (28.5 ha) by non-wooded areas. The restricted study area is characterized by the presence of eight different plant associations of the terrestrial environment totaling 40 ha, the most widespread of which are the mature gray pine forest with black spruce on dry station.

The restricted study area also includes 16 wetlands corresponding to five different plant associations. Together they cover 13.8 ha, or 19% of the restricted study area, and almost all have hydrological connectivity with the area's streams. The wetlands inventoried have all experienced disturbances of natural or anthropogenic origin. The ecological value of all these wetlands is considered low (Groupe Conseil Nutshimit-Nippour and Englobe, 2017).

20.2.4.2.2 Fish and Fish Habitat

Five streams were characterized in the study area. There is no upstream water link with a body of water that would shelter fish populations. These streams flow towards the Saguenay, but the topography of the area with steep slopes prevents fish from going upstream to the project site. The composition of the benthic fauna was determined by sampling two streams in the restricted study area. Benthos samples collected confirmed low diversity and abundance of organisms, reflecting low environmental productivity.

20.2.4.2.3 Herpetofauna

No specific inventory of herpetofauna has been carried out during the EIA. Searches in public database confirmed the presence of seven species of amphibians and reptiles within a 5 km radius of the study area: the Green Newt, the Grey Salamander, the American Toad, the Cruciferous Tree Frog, the Green Frog, the Wood Frog and the Red-bellied Snake. Each of these species could potentially be found in the study area, if suitable habitat is found there. No record of species with protected status belonging to this wildlife group is reported for the local study area. (Groupe Conseil Nutshimit-Nippour and Englobe, 2017).

20.2.4.2.4 Wildlife and their Habitats

Three species of large fauna frequent the project area: moose, white-tailed deer and black bear. With regard to small fauna and fur-bearing animals, 19 species could potentially frequent the restricted study area and the presence of some of them was confirmed during the fieldwork. Among these, the beaver is the species that has been most visible during the various inventories. Other species observed or for which some evidence has been identified are raccoons, red foxes, red squirrels, snowshoe hares and striped chipmunks (Groupe Conseil Nutshimit-Nippour and Englobe, 2017).

20.2.4.2.5 Chiropterans

The sonograms collected revealed the confirmed presence of the following species in the study area: Grey Myotis, The Silver Bat, the NOAE Complex (Silver Bat and Great Brown Bat) and the Myotis sp Complex. (Little Brown Myotis and Northern Myotis). Among these species, the Northern Myotis, Little Brown Myotis, Silver Bat and Grey Myotis are protected species (FaunENord, 2018).

20.2.4.2.6 Birds

Public databases report the presence of more than 250 species of birds that may frequent the Bay area at any time of the year. However, only a small proportion of these species are likely to use the restricted study area significantly, particularly for their reproduction. It should also be noted that eight species of birds were observed in the restricted study area during the various inventories carried out in 2016: the American kestrel, the American woodcock, the grey-cheeked warbler, the ruffed grouse, the St. Martin's golden bear, the minor woodpecker, the little buzzard and the flamboyant woodpecker. In terms of special status species, seven species could potentially frequent the restricted study area: the fir crossbill, the common nighthawk, the olive-sided flycatcher, the Canada warbler, the short-time owl, the rusty blackbird and the peregrine falcon. However, only the Canada Warbler has been confirmed in the vicinity of the project site. (Groupe Conseil Nutshimit-Nippour and Englobe, 2017).

20.2.4.3 Human Environment

20.2.4.3.1 Land Planning and Development and Land Use

The local study area of the project is located in the Saguenay–Lac-Saint-Jean administrative region on the territory of the City of Saguenay. It is mainly part of the borough of La Baie. With the exception of the land in the Grande-Anse marine terminal area, which is under public tenure under the management of Port Saguenay, the rest of the territory of the local study area essentially overlaps with private lands. The restricted study area, site where the plant will be located, is entirely located in the borough of La Baie of the City of Saguenay on private land belonging to Port Saguenay.

The restricted study area overlaps two territorial uses, namely the industrial allocations and the industrial development plan (PAEI), which occupy 81% and 19% of the territory respectively. The footprint of the project is entirely included in an industrial use zone that allows light and heavy industries.

20.2.4.3.2 Archaeology Heritage

The *Cap à l'Ouest* area is not rich in archaeological sites, both for Native American prehistory and for the historical period. The Inventory of Archaeological Sites of Québec (ISAQ) bank has no registered archaeological sites within the restricted study area, nor on the *Cap à l'Ouest* area. To confirm the archaeological potential specific to the restricted study area, an archaeological potential study was conducted.

The assessment of the archaeological potential confirmed that the site on which the plant will be installed represents little to no archaeological potential, with the exception of the fallow field in the southwest which has a stronger potential. A review of known significant archaeological sites nearby reveals that no Paleo-Indian sites have been discovered so far on the Saguenay River axis or at Lac Saint-Jean. (Groupe Conseil Nutshimit-Nippour and Englobe, 2017).

20.2.4.3.3 First Nations

The local study area intersects with the ancestral territory of three indigenous communities: Mashteuiatsh, Essipit and Pessamit, which are part of the Great Innu Nation. The local study area touches the Nitassinan South-West Part, a territory of common interest for the three communities mentioned above. Historically, the Saguenay River was part of an important communication route for the Innu that connected Tadoussac to Hudson Bay. However, the project area is now in municipalized territory, on private land, near urban poles. It therefore has a very low potential for use for traditional purposes.

20.2.4.4 Environmental and Social Impacts

20.2.4.4.1 Anticipated Impacts

The completion of BlackRock metallurgical plant project will have potential impacts on the physical, biological and human environments during the various phases of the project, namely development and construction, and operation and maintenance. This section summarizes the main social and environmental anticipated impacts associated with the development of the project, as identified in the EIA report submitted in 2017 (WSP, 2017). Although this list is not exhaustive, it underlines topics that will require specific consideration.

Only the anticipated impact and benefits for the most sensitive components are presented below. However, with the mitigation measures that will be implemented and the compensation plans, the residual anticipated impacts on the physical, biological and social environment are all considered minor or not significant.

Physical Environment:

- Surface water, groundwater and soil quality alteration (suspended solids, contamination by accidental spills, effluents discharges);
- Alteration, disruption and destruction of watercourses and water bodies;
- Deterioration of air quality through suspended dust and other air contaminants;
- Increased ambient noise level.

Biological Environment:

- Loss of terrestrial vegetation and wetlands;
- Wildlife and bird habitats loss and disturbance;
- Fish habitat loss.

Social Environment:

- Job creation;
- Economic benefits for local and regional suppliers;
- Landscape modification and visual impact;
- Increased noise levels in the vicinity of the plant;
- Increased traffic of heavy vehicles and workers on local and regional roads;
- Increase of dust and other airborne contaminants.

20.2.4.4.2 Mitigation Measures

BlackRock made the commitment in the EIA to implement over 90 mitigation measures to attenuate the negative anticipated impacts mentioned in the previous section. In the construction CA applications Blackrock committed to respect about 75 mitigations and monitoring measures to ensure that the site construction activities impacts will remain acceptable to the communities and surrounding environment.

BlackRock will monitor the impacts of its construction work on chiropterans. Their habitat loss will also be compensated by the installation of artificial dormitories.

A compensation plan for the loss of wetlands and impact to water streams was prepared for approval by MELCCFP. The plan will include a financial portion and an existing wetland restauration project. It would recreate ecological functions and services greater than those affected by the transformation plant project mainly with regard to faunal and floral habitat functions (Groupe Conseil Nutshimit-Nippour, 2021).

The next permit applications will consider recent project modifications and assess its environmental and social impacts to ensure that the metallurgical plant construction and operation meets environmental legal requirements. The mitigation and monitoring measures will be modified if required to maintain impacts to an acceptable level.

20.2.4.4.3 Cumulative Effects

Cumulative effects are changes to the environment that are caused by an action in combination with other past, present and future human actions. Consideration of cumulative environmental impacts is an essential component of any environmental assessment. Six Valued Ecosystem Components (VEC) were selected for the cumulative effects' assessment (WSP, 2018):

- Air quality and greenhouse gases (GHGs);
- Wetlands and terrestrial vegetation;
- Ambient noise;
- Landscape and visual;
- Navigation;
- Jobs and the economy.

To assess the cumulative effects of the project, the recent and future projects were considered to be additional to those at the BlackRock project site (including its mitigation measures). As no significant cumulative effects are anticipated on these VEC, no additional mitigation is required.

20.2.5 Closure Activities

At the end of the plant's useful life, a closure plan will be developed to minimize the impacts of the closure and maximize the success of the site rehabilitation. The lease with the Port Authority includes provisions for the Port Authority to acquire buildings and other infrastructure for future use.

The closure plan will be developed in compliance with MELCCFP legal requirements relating to the protection and rehabilitation of land. This will require measures concerning the environmental characterization of the site and, where applicable, the rehabilitation plan, the appropriate registration of notices in the Land Register and notices to authorities or neighbours.

21. Capital and Operating Costs

The BlackRock Mine and Beneficiation Plant Project described in the Basis of Estimate document 3017012-000000-33-CKC-0001 / R05 is based on the construction of a Greenfield facility having an annual treatment capacity of approximately 3.3 Mtpy of mineralized feed originating from the Southwest Pit. After being processed, approximately 856 ktpy of VTM concentrate will be produced based upon the mine plan. The capital cost estimate related to the Southwest open pit mine, concentrator and site infrastructure was initially developed by BBA in Q2 of 2019. BBA consolidated the cost information from all sources to determine the overall project costs. To match the metallurgical plant throughput and not create an excess of concentrate, the process plant would operate with an average throughput of 3.3 Mtpy (mine plan throughput).

The Basis of Estimate for the metallurgical plant assumes construction of a greenfield facility to be built in Saguenay, Québec having an annual treatment capacity of approximately 856 ktpy of VTM Concentrate.

BBA's original mandate completed in Q3 of 2019 was to ratify the Feasibility Study (FS) fulfilled by Tenova / SNC-Lavalin in August 2017.

The capital cost estimate was based on detailed engineering material take-offs, bids received from vendors and contractors, mainly from the previous study phase, and some data from historical projects.

The initial capital cost estimate does not include taxes, replacement capital or working capital requirements after commissioning and start-up.

In 2020 BBA completed a refresh of the 2019 estimate to May 2020 levels, and in 2021 BBA completed a subsequent refresh of the 2019 estimate to September 2021 levels. The estimate is now being updated once more to reflect current market conditions as of September 2022.

The methodology and basis of the original estimate was not revisited or modified as it pertained to scope, unit construction hours, construction work week, productivity factors, etc. The updates are purely a pricing refresh to better reflect market conditions as of the estimate base dates. That estimate refresh focused on the following main elements:

- Adjust Beneficiation Plant equipment pricing as per information received by BRM from FLSmidth with respect to adjustment to their 2020 firm pricing;
- Adjust metallurgical plant equipment pricing as per information received by BRM from the various equipment vendors with respect to adjustment to their 2020 firm pricing;
- Adjust pellet plant pricing cost with latest design and pricing provided by Metso;
- Perform a refresh of the contingency model input values and perform a Monte Carlo simulation on the adjusted model to obtain revised value at P50.

21.1. Capital Cost Basis of Estimate and Assumptions

Globally, the capital cost estimates were considered Class 2 as defined by AACE International Recommended Practice N° 18R-97. As such, AACE provides a broad range for accuracy within each estimate class. The accuracy achieved was evaluated in the consideration of the level of definition reached in major engineering deliverables, execution strategy and pricing for each plant.

The Capital Cost Estimates were developed with an expected accuracy range of -10% on the low side and +15% on the high side.

Note that accuracy is primarily a function completeness of information with respect to project scope, engineering deliverables, quantities, schedule, and execution strategy and not pricing fluctuation. To that end the estimate refresh does not materially impact the project accuracy or the results of the contingency analysis at P50 as attested by the updated Monte Carlo analysis.

21.1.1. Base Currency and Exchange Rates

The estimate base currency is Canadian dollars for both estimates. The 2020 estimate refresh maintained the currency exchange rates applied in the previous estimate revisions.

The estimates were developed in native (quoted) currencies for equipment. The exchange rates used to convert the different currencies are provided in Table 21-1.

Table 21-1: Currency conversion rates

Country	Currency	Equivalent
European	0.70 EUR	1.00 CAD
United States	0.76 USD	1.00 CAD

21.1.2. Reference Date

The base date for the 2020 refresh estimates for both the Beneficiation and Metallurgical Plants was June 15, 2020, corresponding to market conditions and commodity material pricing in May of 2020. The reference date for the 2021 estimate updates was October 1, 2021.

The estimate base date for the current refresh is October 21, 2022.

21.1.3. Methodology

As in the 2020 and 2021 estimate refresh, the 2022 refresh has not revisited or modified the estimate as it pertains to scope, quantities, unit construction hours, construction workweek, or productivity factors.

Adjustment to bulk material pricing is based on a comparison of indices between May 2020 and September 2022 using US Bureau of Labor Statistics (BLS) producer price indices (PPI).

September 2022 has been selected as it represents the time period for which equipment vendor pricing updates were received and it reflects the latest available month for the published indices.

September 2022 falls within the Capex validity period for NI43-101 reporting.

The estimate refresh focused on the following main elements:

- Update the construction labour crew rates to reflect the latest rates for 2022 as per the Collective Agreement;
- Update bulk material unit pricing using US Bureau of Labour Statistics (BLS) producer price indices (PPI) based on a comparison of May 2020 to September 2022 indices;
- Adjust equipment pricing as per information received by BRM from the various equipment vendors;
- Adjust the balance of equipment pricing using May 2020 to September 2022 BLS PPI indices;
- Update and adjust EPCM costs with the latest forecast values provided by BBA;
- Perform a refresh of the contingency model input values and perform a Monte Carlo simulation on the adjusted model to obtain revised value at P50.

21.1.4. Labour Costs

Estimated labour costs for the Beneficiation Plant are based on 10 hours per day, five days per week for a work-week total of fifty hours. Rotation is on a 4 to 1-week basis. There is no allowance for a second-working team (night shift). Labour rates also include an allowance for room and board for all workers except for those working on civil works as it is assumed that local workers will be available to work on the project, as there is no camp on site. An allowance for displacement was taken into account for 50% of the local civil workers that could come from a radius superior to 65 km, the other workers are assumed to be lodged in a radius inferior to 65 km. However, it is important to note that the mine is a few km short of the 65 km radius of Chibougamau and this might be an expensive point to check with the unions because contractors will likely try to find lodging outside this 65 km.

Estimated labour costs for the Metallurgical Plant are based on height hours per day, five days per week for a work-week total of 40 hours. There is no allowance for a second-working team (night shift). An allowance has been included for casual overtime equivalent to four hours per week, paid at double time across all activities. Furthermore, an allowance for the displacement was taken into account mainly for specialized disciplines.

Wage rates for crews have been established based on Québec construction industry labour agreement in accordance with the collective institutional/commercial and industrial sectors as of May 1, 2022.

The rates are valid for the balance of the 2022 calendar year.

Table 21-2 and Table 21-3 provide a comparison between the 2020 and 2022 "all-in" rates for the Metallurgical and Beneficiation Plants.

Table 21-2: Met plant composite labour crew rates

Typical Crew	Direct Labour	Indirect Labour	Construction Equipment	Total 2022	Total 2020
Site Development	$68.2	$33.2	$69.7	$171.2	$159.38
Concrete works	$67.1	$32.9	$16.1	$116.1	$110.16
Structural Elements	$70.9	$39.2	$26.1	$136.2	$129.36
Architectural	$69.9	$34.8	$15.9	$120.6	$111.35
Mechanical – Process	$70.5	$40.8	$16.3	$127.5	$121.17
Piping and Fittings	$71.7	$41.0	$14.7	$127.4	$111.58
Electrical	$69.1	$39.3	$5.2	$113.7	$110.38
Process Controls	$69.5	$39.4	$3.6	$112.4	$106.63

Table 21-3: Beneficiation plant composite labour crew rates

Typical Crew	Direct Labour	Indirect Labour	Construction Equipment	Total 2022	Total 2020
Site works – Civil	$78.6	$48.6	$85.7	$212.9	$197.37
Concrete works	$78.4	$58.3	$18.0	$154.7	$147.30
Metal works	$83.4	$73.6	$32.1	$189.1	$178.41
Architectural	$81.4	$59.7	$19.6	$160.7	$153.24
Mechanical – Process	$82.7	$72.8	$20.9	$176.4	$168.99
Mechanical – Building	$82.6	$72.7	$20.0	$175.4	$167.59
Piping	$82.8	$73.2	$24.1	$180.1	$172.55
Insulation	$79.9	$71.3	$18.4	$169.6	$162.61
Electrical	$83.3	$71.9	$6.9	$162.1	$155.31
Automation/Telecom	$81.2	$70.7	$4.6	$156.5	$149.98

21.1.5. Pricing Adjustments

21.1.5.1. Bulk Material Pricing

As described previously, bulk material pricing was updated using PPI indices to establish suggested increases between May 2020 and September 2022 as shown in Table 21-4.

BBA performed an analysis of recent pricing adjustments on ongoing projects and or recent pricing increases suggested by suppliers and fabricators for select commodities such as wire and cable, carbon steel pipes, and structural steel fabrication which closely correlate to the increases suggested by the BLS PPI data.

The following table provides a summary of the PPI indices used to update commodity pricing.

Table 21-4: US BLS PPI Commodity indices

		Actualised Indices							Projected				Escalation
Commodity ID	**Description**	**May 2020**	**May 2021**	**Jul 2021**	**Aug 2021**	**Sep 2021**	**Dec 2021**	**May 2022**	**Jun 2022**	**July 2022**	**Aug 2022**	**Sep 2022**	**May 2020 to Sep 2022**
PCU32731-32731-	Cement manufacturing	256.2	268.2	269.4	271.2	271.7	271.2	287.6	288.5	296.7	299.2	302.0	17.9%
PCU32121-32121-	Plywood and engineered wood product mfg	124.7	251.1	263.4	206.4	186.0	202.0	242.3	225.7	209.1	210.4	206.2	65.3%
PCU3323123323121	Fabricated structural metal bar joists and concrete reinforcing bars	210.7	267.8	299.1	315.1	324.6	332.0	366.6	370.2	364.7	359.7	354.3	68.1%
blended concrete	***Concrete blended index (cement, formwork & rebar)***	*202.1*	*239.2*	*250.0*	*248.6*	*249.1*	*252.8*	*275.5*	*275.0*	*275.6*	*275.5*	*274.9*	*36%*
PCU3273903273901	Precast Concrete	322.2	345.5	352.9	353.2	359.2	367.3	394.4	397.3	405.9	407.7	409.4	27.1%
PCU32712032712011	Clay building material and refractories mfg.	201.3	212.6	214.1	214.1	214.1	219.6	230.2	232.2	233.5	235.6	235.7	17.1%
PCU3312--3312--	Steel product mfg from purchased steel	192.9	262.3	303.5	315.9	332.6	358.0	371.1	365.3	357.3	351.8	345.1	78.9%
PCU3323--3323--	Architectural & structural metals mfg	165.5	191.1	207.9	214.4	217.7	232.7	256.1	257.4	257.1	258.0	256.0	54.7%
PCU332311332311	Prefabricated metal building and component manufacturing	330.0	410.5	469.6	485.0	490.0	500.6	520.5	505.3	500.3	508.0	498.0	50.9%
PCU3323213323211	Metal Doors and windows	271.2	297.1	313.8	331.7	335.8	377.5	406.3	408.6	409.9	410.0	406.9	50.1%
Buildings	***Blended Index for buildings***	*239.0*	*299.2*	*334.6*	*344.0*	*352.4*	*366.9*	*385.0*	*381.1*	*375.6*	*375.0*	*369.4*	*55%*
PCU332996332996	Fabricated pipe & pipe fitting mfg	318.7	353.8	386.0	393.9	403.0	417.4	436.6	457.0	446.2	437.3	432.3	35.7%
PCU33121033121004	Fabricated pipe & pipe fitting mfg - stainless steel	85.2	99.9	103.4	106.9	110.1	116.9	146.2	145.8	140.1	136.9	132.2	55.2%
PCU32612232612213	Plastics pipe and pipe fitting manufacturing	182.9	248.7	309.3	335.0	343.3	395.6	436.6	437.3	441.9	442.5	440.3	140.7%
PCU3329193329194	Piping valves	329.6	405.7	407.7	409.4	412.4	422.5	472.8	476.1	485.7	490.3	480.7	45.8%
HDPE supply	HDPE Pipe Supply	100.0	189.5	189.5	189.5	189.5	189.5	189.5	189.5	189.5	189.5	189.5	89.5%
CSRL Pipe	Carbon steel rubber lined pipe	243.4	276.3	297.7	310.4	318.0	315.0	331.0	341.8	345.2	342.3	338.2	38.9%
HDPE Pipe	HDPE Pipe manufacturing	101.1	160.3	160.3	160.3	160.3	160.3	160.3	160.3	160.3	160.3	160.3	58.6%
HDPE Liners	HDPE Liners	152.1	165.7	173.1	177.6	180.3	179.2	184.8	188.6	189.8	188.7	187.3	23.1%
PCU33592-33592-	Communications & energy wire & cable mfg	167.4	215.1	223.7	231.0	234.0	241.8	262.5	262.5	257.3	260.8	259.5	55.0%
PCU3313153313150	Aluminum cable tray	170.0	214.8	224.3	232.8	240.0	237.1	260.9	245.5	235.2	219.9	215.3	26.7%
PCU334513334513	Industrial process control mfg	228.2	231.5	233.5	234.2	234.6	236.5	253.6	257.7	260.9	261.4	261.3	14.5%
PCU212319212319	Other crushed and broken stone mining	295.2	301.9	303.5	299.4	300.8	297.0	322.6	323.4	320.3	330.4	331.5	12.3%

21.1.5.2. Equipment Pricing

BRM has provided pricing confirmation and adjustments from select suppliers as described below. The balance of mechanical and electrical equipment was updated using PPI indices to establish suggested increases between May 2020 and September 2022 as shown in Table 21-5.

Mechanical equipment not linked to specific indices was adjusted using the blended indices created for mechanical supply.

Beneficiation Plant

BRM has received confirmation from FLSmidth of a 42% pricing increase to their original 2018 Firk bid pricing to reflect current market conditions.

BRM has also received updated pricing from Hitachi for the main substation.

Metallurgical Plant

BRM has received a confirmation from Tenova dated October 24, 2022, of a 15.6% increase from their June 2020 pricing adjustment to account for increases to equipment manufacturing, bulk material pricing and marine freight.

Primetals provided BRM with an updated price for the converter, representing an increase of 38% over 2020 pricing levels.

Proco have provided BRM with an updated price for the supply and installation of DRI tower structural steel, indicating an increase of $5.57M on the steel supply over 2020 pricing levels.

BRM has received confirmation from Danieli of a 19% increase to their pricing for the water treatment plant over 2020 pricing levels.

Finally, BRM has received revised pricing Metso for the Pellet Plant.

Table 21-5: US BLS PPI Select equipment indices

		Actualised Indices							Projected				Escalation
Commodity ID	Description	May 2020	May 2021	Jul 2021	Aug 2021	Sep 2021	Dec 2021	May 2022	Jun 2022	July 2022	Aug 2022	Sep 2022	May 2020 to Sep 2022
Process Equipment	*Blended Index for Mechanical Supply - Mine*	*145.7*	*154.8*	*160.3*	*161.8*	*164.2*	*169.8*	*180.7*	*181.8*	*182.4*	*183.0*	*183.5*	*26%*
Process Equipment	*Blended Index for Mechanical Supply - Met Plant*	*148.0*	*159.9*	*167.1*	*169.1*	*172.2*	*178.7*	*189.8*	*190.6*	*190.7*	*191.1*	*191.2*	*29%*
PCU333---333---	Mechanical equipment mfg	141.9	146.4	149.2	150.1	151.4	155.9	166.0	167.3	168.3	169.2	169.9	19.7%
PCU3353--3353--	Electrical equipment mfg	150.4	156.8	160.5	160.9	164.9	170.6	187.5	191.6	193.3	195.6	197.0	31.0%
PCU332420332420	Metal tanks (heavy gauge) manufacturing	158.8	187.4	206.9	208.2	211.9	217.0	234.0	235.8	237.0	237.0	238.3	50.0%
PCU4831114831115	Deep Sea Freight Transportation	293.2	329.9	347.6	342.1	354.4	373.6	447.0	439.1	445.9	463.4	459.1	56.6%
PCU333922333922	Conveyor and conveying equipment mfg	225.8	230.2	235.8	242.6	248.0	260.2	274.4	277.4	280.6	282.2	283.2	25.4%
PCU333611333611	Turbine and turbine generator set units mfg	227.7	234.6	234.6	234.4	236.2	237.7	247.7	248.5	248.3	246.8	246.9	8.4%
PCU333996333996	Pump and pumping equipment manufacturing	195.5	200.2	201.5	203.7	204.8	208.6	219.6	220.9	223.1	223.1	223.1	14.1%
PCU333923333923	Overhead cranes, hoists and monorail systems mfg	170.7	179.9	187.0	189.3	191.2	197.1	208.3	214.2	215.0	216.7	217.2	27.2%
PCU221110221110	Electric power generation	118.5	141.7	175.8	181.2	187.2	163.1	277.7	236.8	282.3	271.5	292.3	146.6%
PCU22112222112243	Electric power distribution	194.6	199.9	221.2	222.6	220.0	210.1	223.8	237.5	246.7	249.9	252.4	29.7%
PCU335311335311P	Electric power and specialty transformer mfg	247.8	284.3	303.2	303.5	313.0	347.0	388.2	403.9	408.0	409.8	410.1	65.5%
PCU333612333612	Speed changer, drive, and gear mfg	297.9	308.0	313.8	317.0	319.2	324.5	350.2	350.2	351.2	355.6	361.2	21.3%
PCU333912333912	Air and gas compressor manufacturing	247.8	255.3	262.5	263.6	266.9	271.4	300.2	305.7	311.5	314.3	320.0	29.1%
PCU33312-33312-	Construction Equipment	149.3	153.1	155.2	155.7	156.5	163.7	170.5	171.2	172.5	177.3	177.6	18.9%

21.1.6. Indirect Costs

EPCM services

BBA has made an adjustment to the value EPCM cost for the Metallurgical Plant based on an assessment of the forecast to-go hours and costs to complete and have included an allowance for inefficiency associated with the ramp-up of detail engineering activities. The adjusted value represents an increase of 1.8M compared to June 2020.

Construction indirect costs (site temporary facilities)

Since these costs are primarily labour based, an increase of 7.4% has been applied to the 2019 values to account for two calendar increases to the collective agreement rates.

Freight / Transportation

Freight costs have been adjusted using the BLS PPI index for marine freight for a 57% increase.

Spare parts

Spare parts have been increased by 10%.

Vendors' representatives

The cost of vendor assistance has been increased by 8%.

21.1.7. Contingency

The contingency model used to generate the Monte Carlo simulations for Beneficiation and Metallurgical Plant have been updated to reflect the 2022 refresh values. The models retained the same methodology (contingency terms and ranges) as the previous update.

As stated in the basis of estimate documents for both plants, the calculated contingency is not intended to take into account items such as labour disruptions, weather-related impediments, changes in the scope of the project from what is defined in the study, nor does contingency take into account price escalation or currency fluctuations.

Table 21-6: Contingency results for the Beneficiation Plant

Percentile	Simulation Values CAD	Contingency Amount CAD	%
5%	351,867,463	13,977,034	4.1%
10%	356,528,683	18,638,254	5.5%
15%	359,471,300	21,580,871	6.4%

Percentile	Simulation Values CAD	Contingency Amount CAD	%
20%	361,943,094	24,052,665	7.1%
25%	364,198,110	26,307,681	7.8%
30%	366,217,897	28,327,468	8.4%
35%	368,139,549	30,249,120	9.0%
40%	369,875,474	31,985,045	9.5%
45%	371,524,145	33,633,716	10.0%
50%	**373,194,475**	**35,304,046**	**10.4%**
55%	374,790,966	36,900,537	10.9%
60%	376,443,443	38,553,014	11.4%
65%	378,223,627	40,333,198	11.9%
70%	379,880,369	41,989,940	12.4%
75%	381,944,531	44,054,102	13.0%
80%	384,121,442	46,231,013	13.7%
85%	386,843,348	48,952,919	14.5%
90%	390,146,481	52,256,052	15.5%
95%	394,983,820	57,093,391	16.9%

Table 21-7: Contingency results for the Metallurgical Plant

Percentile	Simulation Values CAD	Contingency Amount CAD	%
5%	1,022,015,383	15,621,733	1.6%
10%	1,037,495,775	31,102,125	3.1%
15%	1,049,067,404	42,673,754	4.2%
20%	1,058,058,045	51,664,395	5.1%
25%	1,065,830,244	59,436,594	5.9%
30%	1,073,327,705	66,934,055	6.7%
35%	1,079,609,849	73,216,199	7.3%
40%	1,085,752,363	79,358,713	7.9%
45%	1,091,895,567	85,501,917	8.5%
50%	**1,097,863,390**	**91,469,740**	**9.1%**
55%	1,104,009,207	97,615,557	9.7%
60%	1,110,465,762	104,072,112	10.3%
65%	1,116,555,065	110,161,415	10.9%
70%	1,123,200,155	116,806,505	11.6%
75%	1,130,595,566	124,201,916	12.3%
80%	1,138,831,931	132,438,281	13.2%
85%	1,148,593,877	142,200,227	14.1%
90%	1,161,128,185	154,734,535	15.4%
95%	1,179,132,609	172,738,959	17.2%

21.2. Capital Cost Estimates

The following table provides the preliminary updated Capex values for the Beneficiation and Metallurgical Plants.

The estimate updates can be considered completed as the following elements have been incorporated:

- Confirmation of equipment price adjustment from FLSmidth for Beneficiation Plant major process equipment;
- Confirmation from respective suppliers for the increase to pricing for the pellet plant and water treatment facility for the Metallurgical Plant;
- High-level review of the updated construction schedule.

The following table provides a cost for the Beneficiation Plant Capex.

Table 21-8: Beneficiation Plant capital cost summary by area

Area	Area Description	Total (CAD $M)
0000	Off-site – excluding rail preparation	4.489
0000	Off-site – rail preparation	3.124
1000	Infrastructures	10.413
2000	Administration and Services	12.082
3000	Mine (earthworks and mine garage)	15.272
4000	Crushing	23.301
5000	Stockpiling and Conveying	22.356
6000	Processing Plant and Load-out System	139.578
7000	Tailings and Water Management	28.832
Subtotal Direct Costs		**259.446**
8000	Owner's Costs	42.766
9100	EPCM Services	13.657
9200	Construction Indirects	8.553
9500	Commissioning	3.058
9900	Common Distributables (Freight, spares)	10.371
Subtotal Indirect Costs		**78.406**
9800	Contingencies	35.304
Grand Total		**373.156**

Table 21-9: Beneficiation Plant capital cost summary by discipline

Area Description	Total (CAD $M)
Civil	42.199
Concrete	26.114
Structural	25.181
Architectural	14.767
Mechanical	73.930
Piping	19.693
Electrical	45.916
Automation / Telecommunications	11.647
Tailings and Water Management	42.199
Subtotal Direct Costs	**259.446**
Owner's Costs	42.766
EPCM Services	13.657
Construction Indirects	8.553
Commissioning	3.058
Common Distributables (Freight, spares)	10.371
Subtotal Indirect Costs	**78.406**
Contingencies	35.304
Grand Total	**373.156**

Note that the following costs have not been included in this table:

- Capitalized mining operating costs ($34.07M)
- Mine Services Equipment ($1.34M)

Table 21-10: Metallurgical Plant capital cost summary by area

Area Description	Total (CAD $M)
00000 - General	29.6
01000 - VTM Concentrate and Pelletizing Plant	176.2
02000 - Direct Reduction Plant	346.0
03000 - OSBF Electrical Furnace	78.7
04000 - OSBF Furnace Off-gas Treatment	12.6
05000 - Auxiliary Plants	54.4
06000 - Hot Metal and Slag Handling	129.5
07000 - Electrical General Systems, Automation and Controls	39.5
08000 - Administration and Ancillary Facilities	8.4
Subtotal Direct Costs	**875.0**

Area Description	Total (CAD $M)
Owner's Costs	57.9
EPCM Services	29.7
Construction Tempo Facilities and Site Maintenance	19.3
Professional Services - Third Party	2.9
Commissioning Services	6.7
Common Distributables (freight, spares, initial fill, tech assistance)	14.7
Subtotal Indirect Costs	**131.4**
Contingency	91.5
Total Costs	**1 097.9**

Table 21-11: Metallurgical plant capital cost summary by discipline

Description	Total (CAD $M)
Site Construction - Civil / Earthworks	22.3
Concrete - Cast-in-Place	47.7
Structural Elements	102.7
Architectural Finishes	35.1
Mechanical Equipment	500.9
Piping and Fittings	73.8
Electrical Components	78.3
System Controls / Instrumentation	14.4
Subtotal Direct Costs	**875.0**
Owner's Costs	57.9
EPCM Services	29.7
Construction Tempo Facilities and Site Maintenance	19.3
Professional Services - Third Party	2.9
Commissioning Services	6.7
Common Distributables (freight, spares, initial fill, tech assistance)	14.7
Subtotal Indirect Costs	**131.4**
Contingency	91.5
Total Costs	**1 097.9**

21.2.1. Capital Cost Exclusions

The following costs are not included in the capital cost estimate:

- Taxes and customs duties;
- Schedule acceleration costs;
- Schedule delays and associated costs, such as those caused by:
- Unexpected site conditions;
- Unidentified underground conditions;
- Labour disputes;
- Force majeure;
- Permit applications delay;
- Development fees and approval costs beyond those specifically identified.
- Cost of any disruption to normal operations;
- Risk events (Project Risk Register);
- Financing costs and capitalized interest;
- Foreign exchange fluctuations;
- Operator management fees;
- Cost associated with third party delays;
- Owner's reserve;
- Technology supplier(s) services and owner's consultant;
- Environmental and Construction permitting;
- Costs of working capital;
- Two-year capital or insurance spares;
- Development fees and approval costs of Statutory Authorities;
- Change in law and regulations;
- Soil decontamination and disposal costs;
- OPEX evaluation (part of a separate exercise by BRM);
- Labour, material and equipment escalation costs.

Provisions for escalation and provisions on currency risk were excluded, as the estimate is expressed in constant dollars as of October 2022.

21.3. Sustaining Costs

21.3.1. Mining Sustaining and Replacement Costs

Mine sustaining capital cost requirements are based on the equipment purchasing schedule mentioned above as well as the replacement of equipment that has achieved its projected service life. The LOM annual mine fleet capital cost, including both the sustaining and replacement costs, ~~i~~s 20.09M$. Down payments and leasing payments incurred during operation over the life of mine (LOM) are reflected in the leasing operating costs (section 21.4.4).

21.3.2. Plant & Site Infrastructure Sustaining Costs

A $22.5M allocation starting in Year 20 has been divided between periods (Year 20, Year 21-25, Year 26-30, Year 31-35) to account for piping and pump replacement costs, potential additions to the tailings line and tailings management as well as replacement of general plant equipment. An additional allocation of $2.6M has been made for sustaining tailings pumping equipment as the tailings management strategy evolves over the life of mine.

21.3.3. Initial Fill

The materials and consumables required for start-up (oil, lubricant, grinding media, spare parts) were estimated to be $3.5M.

21.3.4. Plant and G&A Mobile Equipment Costs

Plant and G&A mobile equipment required for start-up will be used from equipment purchased during the construction phase of the project (pick-ups, front-end loaders). Additional equipment such as a boom truck, forklift and bobcat will be purchased in the first year of operation. As equipment wears (lives between 9-11 years), it will be replaced. Costs vary on a year-by-year basis as specific equipment require new units. The total cost of the mobile equipment is $18.7M over the life of mine.

21.3.5. Tailings Dyke Sustaining Costs

Tailings dykes will be raised to a minimum level in pre-production and raised during the course of the life of the mine. As estimated $47.4M will be required to raise the tailings pond dyke to its final elevation.

21.3.6. Environment and Rehabilitation Costs

The rehabilitation costs have been estimated to be $66.0M as per the mine closure plan provided by Journeaux Associates. Legislation demands that rehabilitation costs be covered in Years - 1,1 and 2 of the mine life at a percentage of 50%, 25% and 25% respectively. To respect the financial guarantee imposed by the MERN, BlackRock plans on obtaining security bonds via an insurance institution.

21.4. Estimated Beneficiation Plant Operating Costs

The operating expenditures (OPEX) for the BlackRock Project encompass the open pit mine, process plant, as well as general and administrative costs. Transport and handling, rail and port facilities and ship loading operating costs were given to BBA by BlackRock Metals.

The Beneficiation Plant operating costs estimate base currency is Canadian dollars.

The average annual LOM operating costs are $119.6M/y or $132.2/t of concentrate milled over a mine life of approximately 39 years for the Southwest pit.

Table 21-12: Operating costs summary (life of mine) 3

Cost Area	Average Operating Costs (CAD)		
	$M/y	$/t milled	$/t Fe Conc.
Mining	55.99	14.83	57.09
Process	27.63	8.47	32.47
General and Administration	8.59	2.62	10.10
Leasing 2	1.41	0.53	2.02
Transport Logistics1	25.22	7.70	29.64
Iron Concentrate Rail Car Maintenance	0.16	0.05	0.20
Other (coarse tailings, environmental)	0.56	0.17	0.65
Total	**119.6**	**34.4**	**132.2**

Notes:
1. Costs calculated/obtained by BlackRock Metals.
2. Leasing includes costs for the train loadout and rolling stock over the life of mine (LOM).
3. Average costs include expenditures in the pre-production period.

21.4.1. Mine Operating Costs

21.4.1.1. Basis of Estimate

Mining costs include the operating and maintenance costs of the equipment, as well as costs related to blasting, personnel, and other costs such as allowance for dewatering, clearing and grubbing. Some of the key supply prices used in the operating cost estimates are as follows:

- Diesel Fuel for mobile equipment: $1.5/litre;
- Emulsion: $121/100 kg (based on quote from Orica – September 30, 2022).

Blasting costs were estimated based on the patterns and tonnages required in both ore and waste. The basis for the unit costs related to blasting was derived from quotations received from suppliers and from an internal database of similar projects. A bulk explosives plant will be constructed on site, operated and maintained by the explosive's supplier, under a service contract.

Equipment unit operating and maintenance costs were developed from quotations received from suppliers' cost estimates and experience. Other sources of information include an internal database of similar projects.

Average salaries are based on an average of several salary studies carried out by third-party firms.

Fuel requirements were calculated according to the annual operating hours for each type of equipment.

Based on the preliminary environmental assessment, it was also assumed that none of the stockpiles (waste rock pile, overburden pile, low-grade ore stockpile, concentrate) would require a membrane, liner or engineering backfill beneath them.

A permanent camp was not included in the costs, as employees will commute from local towns or from Chibougamau, rather than use a fly-in/fly-out system.

Table 21-13: Mining operating costs summary (life of mine)[1]

Cost Area	Mining Operating Costs (CAD)			
	$M/Y	$/T Mined	$/T Milled	$/T Concentrate
Labour	17.50	1.66	5.33	20.53
Loading	4.24	0.40	1.30	4.99
Hauling	11.91	1.13	3.64	14.00
Drilling	2.97	0.28	0.91	3.49
Support and Service Equipment	2.93	0.28	0.89	3.44
Blasting	6.22	0.59	1.90	7.31
Dewatering	0.61	0.06	0.18	0.71
Grade Control	0.62	0.06	0.19	0.73
Miscellaneous	0.73	0.07	0.22	0.85
Total	**47.70**	**4.52**	**14.56**	**56.06**

Notes:

1. Average costs do not include expenditures (Labour, Loading, Dewatering, Support equipment) in the pre-production period, thus the amount for the contractor in the pre-production period is not included.

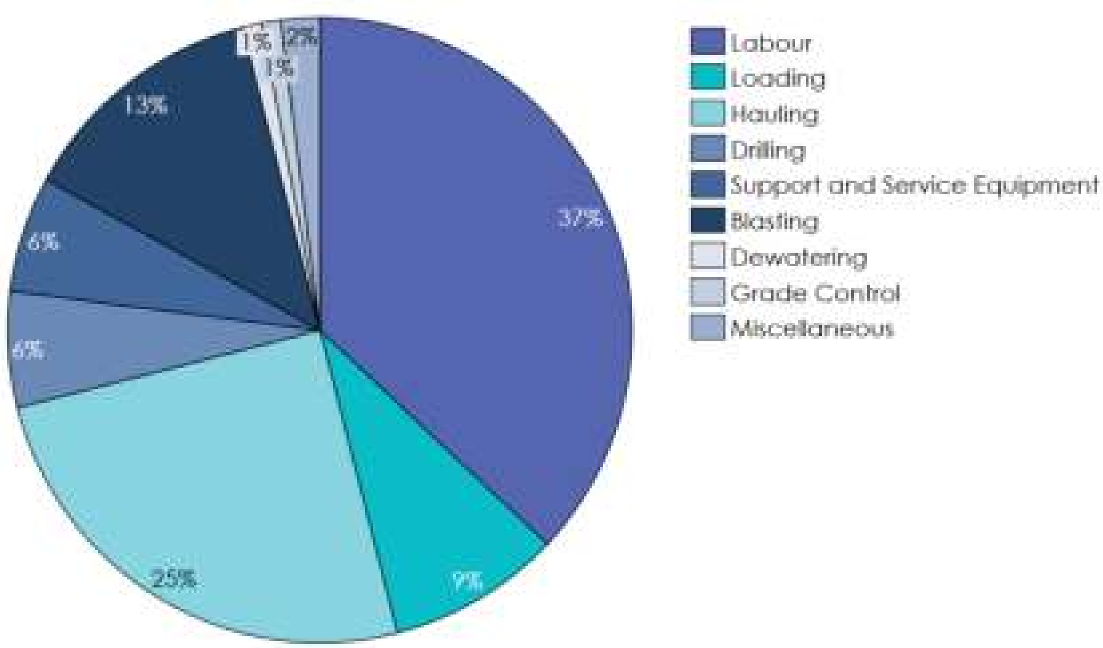


Figure 21-1: Total mining operating costs breakdown (life of mine)

21.4.2. Process Operating Costs

The process operating costs are based on metallurgical testwork, the mine plan, a recent salary survey, literature, and recent supplier quotations. The operating cost breakdown shown in Table 21-14 and Figure 21-2 represents the life of mine costs and is estimated to be $27.6M/y. Costs are based on a production of approximately 865 ktpy of magnetite concentrate.

21.4.2.1. Basis of estimate

Consumable and reagent consumption were estimated based on the following:

- Reagent consumption was estimated from laboratory testwork;
- Spare parts wear was estimated based on discussions with vendors and previous similar projects;
- Industrial references and assumed operational practices were considered;
- Prices for reagents were obtained from recent vendor budgetary quotations or from BBA's internal database. The price of flocculant, CAD5.20/kg, was obtained from a recent budgetary quote from SNF Canada.

The main process plant consumables are crusher liners, screen decks, general spares and reagents (including flocculant, frother, sulphuric acid, collectors and lime). Maintenance parts were estimated to be approximately 3.5% of the total mechanical equipment cost based on an average percentage applied on mechanical equipment cost per area.

Liner and grinding media consumption were calculated based on suppliers' tools and calculations, as well as previous similar projects. The total annual operating costs associated with the purchase of grinding media and liners is estimated to be an average of $6.1M/y for grinding media and $2.2M/y for liners.

Manpower can be divided into four categories: Management; Metallurgy; Operations and Maintenance. The manpower presented in this section only considers process plant employees and excludes general and administration and mining personnel. It is estimated that a total of 37 employees will be required for the process plant. Labour costs were estimated based on the plant organizational structure developed by BBA in collaboration with BRM. Labour rates were provided by BRM based on a recent salary survey from the *Association minière du Québec* and by benchmarking against similarly sized operations in the region. Based on that information, the costs of overtime, insurance, and benefits were estimated to be 38% of the base salary. The annual cost for labour for the process plant is estimated to be $4.5M/y.

Utility operating costs encompass fuel and power. The total operating costs for the process plant utilities are estimated to be $7.4M/y.

The cost of electrical power was established based on Hydro-Québec's "Tarif L" and takes into account the utilization of the power draw. The process plant energy consumption estimate was based on the equipment running loads with various factors applied. The nominal energy requirement for the process plant is estimated between 13.5 to 15 MW. The average total annual electricity cost for the process plant is estimated to be $6.4M.

Table 21-14: Process operating costs summary (life of mine)

Cost Area	Process Operating Costs (CAD)	
	Overall	Magnetite
	$M/y	$/t Fe Conc.
Consumables	**8.4**	**9.8**
Spares	3.0	3.5
Maintenance and Parts	2.5	3.0
Reagents	2.8	3.3
Grinding Media	**6.1**	**7.2**
Personnel	**4.5**	**5.2**
Utilities	**7.4**	**8.6**
Power	6.4	7.5
Fuel	1.0	1.1
Sampling	**0.6**	**0.7**
Contracts	**0.7**	**0.9**
Total	**27.6**	**32.5**

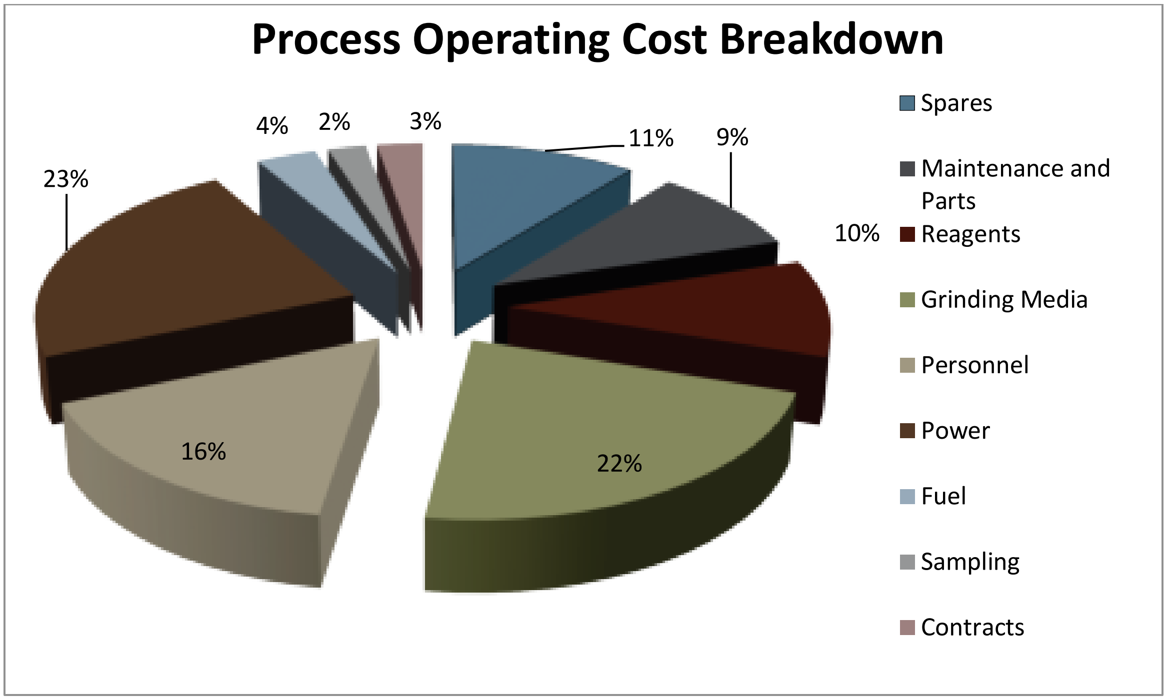


Figure 21-2: Total Process operating cost breakdown (life of mine)

21.4.3. General and Administration Costs

The general and administrative (G&A) costs for the BlackRock Project were estimated to be $2.6/t milled over the life of the mine.

The G&A costs include the following:

- Site infrastructure and maintenance including mobile equipment costs, snow removal, road maintenance and others;
- Site electrical power;
- Manpower: Salary and benefits for a labour force of 56 office, security and maintenance personnel;
- Insurance and legal expenses;
- Health and safety expenses;
- Laboratory and environmental costs, as well as miscellaneous expenses.

A breakdown of the G&A costs is presented in Table 21-15 and Figure 21-3.

Table 21-15 General and administration life of mine cost summary

Cost Area	G&A Costs (CAD)	
	Overall $M/y	Magnetite $/t Fe Conc.
Infrastructure and Maintenance	1.5	1.8
Fuel	0.8	0.9
Power	0.5	0.6
Personnel	2.1	2.4
Contracts	1.2	1.4
Insurance and Legal	0.8	0.9
Miscellaneous Supplies	1.3	1.5
Laboratory	0.2	0.2
Environment	0.1	0.2
Total	**8.6**	**10.1**

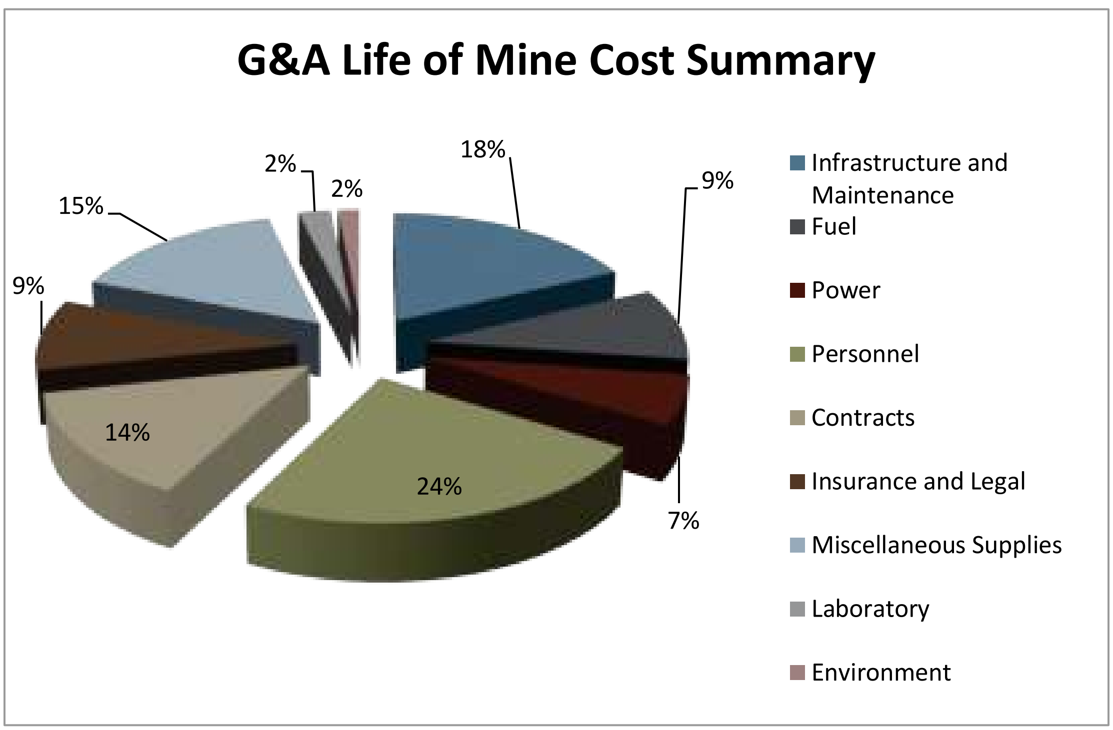


Figure 21-3: General and administration operating cost breakdown

21.4.4. Leasing

BlackRock intends to finance certain equipment and buildings via a leasing option. The train loadout yard, as well as the rolling stock will be purchased via leasing. BlackRock provided estimates on leasing parameters for the rolling stock and train loadout yard. A table summarizing the key leasing parameters.

Table 21-16: Leasing parameters for the BlackRock Project

Cost Area	Down payment (%)	Term (Years)	Rate (%)
Rolling Stock	20*	5*	8.25*
Train Loadout Yard	20*	5*	8.25*

Major mining support equipment will be financed via a leasing option as well. The leasing parameters will be a 20% down payment, 5-year term, and a rate of 8.25%. The total mining leasing cost, including down payments and annual leasing payments, during pre-production is $13.20M. The total mining leasing cost, including down payments and annual leasing payments, during operation over the life of mine is $195.81M (5.90$/t Fe conc.).

21.4.5. Concentrate Transport and Handling Costs

The transport and handling logistics (truck and rail) were provided by BRM at $29.64 per tonne of magnetite concentrate. An additional $0.20 per tonne was added for rail car maintenance. These costs do not fall within the battery limits of the project, however, were essential for mine optimization calculations.

21.4.6. Other (Tailings, Environmental) Operating Costs

Tailings handling operating costs to raise the dam were determined in collaboration with BRM and were estimated to be $0.50/t of concentrate. The use of a hydraulic excavator, a dozer and a foreman at 50% availability was taken into account in the estimation.

21.4.7. Operating Costs: Year 1 to 10

For comparative purposes, the operating costs were evaluated for the first ten years of operation. The costs were broken down in the same way as the overall operating costs for the life of mine and they are shown in Table 21-17. The average annual total operating costs over the first ten years are $112.2/y or $124.1/t of concentrate.

Table 21-17: Operating Costs Summary (First ten years) [3]

Cost Area	Average Operating Costs for Y1-Y10 (CAD)		
	$M/Y	$/T Milled	$/T Fe Conc.
Mining	47.70	13.33	50.30
Process	26.98	8.56	32.31
General and Administration	8.59	2.62	10.10
Leasing [2]	3.00	0.23	0.90
Transport Logistics[1]	25.22	7.70	29.64
Iron Concentrate Rail Car Maintenance	0.14	0.05	0.17
Other (coarse tailings, environmental)	0.56	0.17	0.65
Total	**112.2**	**32.7**	**124.1**

Notes:
1. Costs calculated/obtained by BlackRock Metals.
2. Leasing includes costs for the train loadout and rolling stock for the first ten years.
3. Operating costs for the first ten years do not consider any expenditures in the pre-production period except for leasing costs.

21.5. Estimated Metallurgical Plant Operating Costs

21.5.1. Cost Assumption and Conditions

This chapter provides the details regarding the assumptions that were made to calculate the operating cost for the Metallurgical Transformation plant. All costs are expressed in US dollars.

Table 21-18 presents the cost categories relevant to the High Purity Pig Iron or Nodular Pig Iron grade.

Table 21-18: Cost categories of BRM's OPEX for the production of high-purity pig iron

	Cost Category for NPI Production	Note
1	VTM-Concentrate	The main raw material of the plant
2	Natural Gas	Used in the plant for heating and generation of reducing gas
3	Electricity	Includes operating and auxiliary electrical cost
4	Labour - ex SGA	Cost relevant to manning
5	Refractories	All the refractory used in the operation
6	Gas Plant(O_2&N_2), DMDS, MDEA, CO_2	Gases and reagents for the Energiron process and Converter

	Cost Category for NPI Production	Note
7	Transport to the ship	
8	Maintenance (hardware)	For both operation and general purposes
9	Lime, Dolomite and Bentonite	Consumed as pelletizing additives
10	Carbon	Consumed in the OSBF and Converter operation
11	FeSi for Ti-slag treatment	Consumed in the OSBF slag treatment
12	Fe_2O_3 fines	Consumed in the Converter
13	General services/mobile equip./Waste disposal	General plant requirements
14	Industrial Water	Water cost for all operations relevant to NPI/HPPI
15	Electrodes	Consumed in the OSBF melting operations
16	Converter refinement to HPPI/Nodular Pig Iron	Operation cost for the Converter
17	Pig Caster	Operational cost associated with the pig caster
18	SGA	Sales and General Administration cost
19	Property Lease	
20	Outsource Services	
21	Ti-slag credit	
22	V-slag credit	Credit recognized by the downstream operation

21.5.1.1. VTM Concentrate

BRM will obtain VTM Concentrate from its own mining operations in Chibougamau.

Table 21-19: VTM Concentrate cost

Description	Unit	Value
Purchase price VTM Concentrate	USD/t	100.44
Shipping to Port Saguenay	USD/t	Included above
Total	**USD/t**	**100.44**

21.5.1.2. Natural Gas

Energir will deliver Natural Gas to the industrial site. Preliminary price information has been released by Energir based on the following assumptions:

Natural Gas References:

- HHV = 38.00 MJ/m³;
- At 15°C and 100,000 kPa;
- GM reference: 1 BCF = 28,327,840 m³ = 1,073,342 GJ.

Annual Natural Gas Consumption:

- 172,900,000 Nm^3;
- 1,634,460 Gcal/year ;
- 6,841,500 GJ/year.

Figure 21-4: Map of natural gas distribution to and within Canada

BRM considered a forecasted increase of 2% on the current spot price, and so based on the above assumptions, assumed price for gas is 4.50 USD/MMBtu.

21.5.1.3. Electricity

Electricity will be provided to BRM by the utility company Hydro-Québec (HQ).

Power rates for industrial users are set by what is called the "L-Rate". This rate is a calculation based on the technical demand inputs outlined above. In addition to the L- Rate, a long-term 20% discount is available to development projects, the duration of which is being negotiated with the province of Québec.

New facilities operating in a promising growth sector are entitled to a rate reduction under the Economic Development Rate of Hydro-Québec. This is for new loads connected to the Hydro-Québec system of at least 1,000 kW. BRM project has been evaluated according to the applicable criteria and the project's value added for its economic benefits to Québec.

Economic Development Rate is valid until 2024 and provides for an initial reduction of 20% applied to rate L. This reduction will be diminished by five percentage points a year over the final three years, for a gradual transition to the normally applicable rates. BlackRock is negotiating with the government of Québec to maintain a Tarif L – 20% for 10 years from the start of operation and extended to 20 years depending of Power surplus available from HQ.

The price assumed for this report is = 2.79 USD¢/kwh.

21.5.1.4. Labour

The plant is estimated to employ 122 people, consisting of the following positions:

- General and Plant Manager;
- Supervisors;
- Operators;
- Process Engineers;
- Electrical and Mechanical Engineers;
- Maintenance Supervisors;
- Electrical and Mechanical ;
- Quality Manager;
- Health and Safety Manager;
- Financial Manager;
- Financial Controllers;
- Human Resource Manager;
- Environmental Manager;
- Technicians;
- Procurement;
- Logistics.

Table 21-20: Metallurgical plant processing labour

Metallurgical Plant Processing labour			
Labour per process area	**Worker /shift**	**Shifts**	**No. of workers**
Process administration			**12**
Production Manager	1	1	1
Process Development Engineer (Metallurgists)	2	1	2
Electrification Engineer	1	1	1
Shift Production Supervisors	2	4	8
Metallurgical Plant operators			**58**
Water treatment plant (5100)	1.5	4	6
Iron Ore Pelletizing Area (1200)	2	4	8
Energiron® DR Area (2000)	4	4	16
OSBF Area (3000)	4	5	20
Secondary Metallurgy (CLU®) (6200)	1	4	4
Pig Casting (6300)	1	4	4
Maintenance			**17**
Mechanical Maintenance Engineer	1	1	1
Mechanical Maintenance Technician (Welders)	2	4	8
Electrification Technician	1.5	4	6
Maintenance supervisors	2	1	2
Total			**87**
Metallurgical Plant Processing outsourced labour			
Refractory			**6**
Bricklayers	1.5	4	6
Metallurgical Plant operators			**10**
Raw Material Receiving Area (1100)	1	4	4
V-Slag Crushing and Milling (6400)	1.5	4	6
Gate & Security & First aid			**4**
Gate & Security & First aid	1	4	4
IT-support			**3**
IT-support	3	1	3
Total			**23**

Table 21-21: Metallurgical plant administration labour

Metallurgical Plant Administration labour			
Labour per discipline	**Worker /shift**	**Shifts**	**No. of workers**
Adiministration			**1**
Gneral Manager - VP	1	1	1
Human Resources			**1**
Human Resources Manager	1	1	1
Corporate and Finance			**1**
Financial Controller (Analyst)	1	1	1
Procurement			**2**
Purchasing - Buyer	1	1	1
Purchasing - Receiving	1	1	1
Logistics/Shipping			**1**
Logistics	1	1	1
Environmental & Safety			**5**
Environment Coordinator	1	1	1
Environmental Technician	1	1	1
HS Manager	1	1	1
Health Safety Technician/Nurse	1	1	1
Nurse	1	1	1
Quality			**1**
Quality Manager	1	1	1
TOTAL			**12**
Total Metallurgical Plant			**122**

In addition of the outsourced labour mentioned in the table above, other areas in the plant will be outsourced and these will require labour contracts which have not been included in the employment totals. The outsourced areas are Site Security, Analytical/laboratory Services, Water Treatment, Vanadium and Titanium slag processing and mobile equipment operators.

Various salary benchmark studies were reviewed by BRM (which included benefits and incentive bonus) to develop the necessary base salary for BRM to attract and retain skilled employees. The salary benchmarks reviewed were the following;

- 2019 Canadian Mine Salaries Wages and Benefits;
- Adecco salary guide 2020;
- Alevo salary guide 2020;
- CEZinc Hay scale 2019;
- Enquête salariale OIQ 2019;
- Enquête salariale mine 2018;
- Enquête Stat Québec - salaires 2018;
- Étude sur la rémunération CSMO métallurgie 2019;
- Hays guide salarial 2020;
- Enquête rémunération BRM par RCGT 2019;
- Ranstadt guide salarial 2020;
- Robert Half - guide salarial 2020;
- Collective agreements for Elkem, Fonderie Saguenay, Rio Tinto, Resolute.

The total cost for employees, based on competitive salaries, is USD14,300,381 per annum. Details per process area is included in the next sections of this report.

21.5.1.5. Refractories

Refractories are used in a variety of the equipment in the production process:

- Straight grate refractory for the pelletizing furnace;
- Refractory for the DRI module;
- Refractory for the OSBF;
- Refractory for the ladles;
- Refractory for the converter;
- Refractory for the pig caster;

This has calculated actual quantities of refractory and the refractory needed for the start-up shall be included in the Capex. Some of these refractories, despite being consumables, have a long life and are replaced only during general maintenance. For example, the DR module refractory is fully replaced only after years of operation. In other cases, like for the ladles and granulator tundish, the refractory relining is a routine maintenance activity.

The price assumed for:

- Straight grate refractory for pelletizing is 2.00 USD/kg;
- Bricks for OSBF is: 2.8 USD/kg;
- Refractory bricks for ladles is 2.25 USD/kg;
- Bricks for converter is: 3.4 USD/kg;
- Refractory for pig caster tundish is: 2.50 USD/kg.

21.5.1.6. Maintenance

The cost for maintenance assumed in this section refers to the maintenance supplies, because the maintenance labour cost is already accounted for in the labour section. Maintenance supplies are mainly spares, materials, lubricants, small tools and occasionally outside maintenance. Capital and commissioning spares are already included in the Capex budget. Maintenance cost has been broken down per each one of the production steps of the plant.

There are also other areas that require maintenance, in particular the auxiliary equipment such as Furnace tapping, Water Treatment Plant, substation, etc.

As a reference, in the steel industry of North America the average cost of maintenance supplies per ton of finished product is in the range of **4 USD/t**.

Considering that BRM is a relatively small production plant, maintenance cost may, for that reason, be larger on a per ton basis. This fact is in full or in part balanced by the fact that BRM will start with new equipment, for which only regular maintenance will be needed.

BRM maintenance cost used is **6.79 USD/t**.

21.5.1.7. Limestone, Calcined Dolomite and Bentonite

Bentonite, Limestone, and Mineral Dolomite are used in the pelletizing process, while calcined dolomite is used in the OSBF.

Limestone and dolomite is a commodity that can be purchased locally.

- The price used for Limestone is: **113 USD/t**;
- The price used for Mineral Dolomite is: **113 USD/t;**
- The price used for Bentonite: **98 USD/t;**
- The price used for Calcined Dolomite is: **120 USD/t.**

21.5.1.8. Fe_2O_3 Fines

Fe_2O_3 fines are used in the converter process as a coolant for the operation of production of V-slag. The type of oxide that it is required can come in the form of mill scale or similar, so it is waste product of rolling mills or other operations that produce that type of "rust". The price used for Fe_2O_3 fines is: **79 USD/t.**

21.5.1.9. FeSi

Ferrosilicon (FeSi) is required to the treatment of the OSBF Slag in to maximize the amount of titania in the slag and separate the metallic part from the oxides. It's also used to refine the hot metal after the converter process to produce the desired grade of pig iron required. The market price for FeSi in this is currently in the range of: **1.345 USD/kg**.

21.5.1.10. Carbon and Anthracite

Anthracite is used in the converter process as to maintain the carbon content of the bath to the desired level. Anthracite is used in the OSBF melting process. The price used for Anthracite is **179 USD/t**.

21.5.1.11. Electrode Paste

The OSBF uses Soderberg-type electrodes. The price used for electrodes is: **$500 USD/t.**

21.5.1.12. Industrial Process Water

The Process water will be provided by the municipality of Saguenay. The cost for the water make-up used is **0.4 USD/m³**.

21.5.1.13. V-slag Credit

The products of the upstream transformation process are HPPI, V-slag and TiO_2-slag that are transferred to the downstream Vanadium plant, at a transfer price equal to the cost of the full-converter plant for the V-slag.

21.5.1.14. TiO_2-slag Credit

The TiO_2-slag is transferred to the downstream TiO_2 treatment area, at a transfer price equal to the cost of the FeSi addition for the TiO_2-slag.

21.5.1.15. Titania slag treatment with Ferrosilicon cost assumptions

For the Titania slag treatment, the following costs have been considered.

Table 21-22: Cost used for BRM's Opex in the TiO_2-slag plant, per ton of TiO_2-slag after treatment

Cost Category for TiO_2 – slag production	Price
TiO_2 – slag	TiO_2 – slag at **15.65USD/t** (From FeSi consumption).
Mobile Equipment	At **17USD/t** as a contracted operation to a third party.
Crushing, milling, magsep	At **7.47USD/t** – recovery and process conducted on site.

21.5.2. Operating Costs

The operating costs for the following process areas have been defined in detail in this chapter.

Table 21-23: OPEX process areas

Process Area		Note
1	Raw Material Receiving.	Includes cost from train discharge to delivery at pellet plant.
2	Pelletizing Area.	Include all costs related to pelletizing.
3	Energiron® DR Area.	Include all reduction costs.
4	OSBF and Refractory.	Include all costs for melting and refractory operations.
5	CLU refinement to HPPI/Nodular Pig Iron.	Include all costs relevant to the converter.
6	Pig Caster.	Include all costs relevant to the Pig Caster.
7	Iron Plant Auxiliaries.	Include costs for Pig Iron production auxiliaries.
8	V-Slag Crushing & Milling and bagging.	Includes cost associated with crushing, milling & bagging of V-slag.
9	TiO_2-slag processing area.	All costs relevant to titania slag treatment with FeSi.

21.5.2.1. Raw Material Receiving Area

Table 21-24: Operating cost for raw material receiving area

Raw Material Receiving Area	Units	Unit/t	Unit Price	USD/Year	USD /t
VTM Concentrate	Tons	1.60	101.25	86 672 756	161.56
Salaries & Wages	MH	Varies		500 992	0.93
Shared Personnel	MH	Varies		167 335	0.31
Electricity Operations	kWh	50.29	0.028	762 942	1.42
Maintenance Supplies	n.a	Varies		77 040	0.14
Storage & Handling (ex-IO)				**1 508 310**	**2.81**

The VTM Concentrate is the largest item of cost in the Opex. The rest of the material storage and material handling costs total about 2.81USD/t.

21.5.2.2. Pelletizing Area

Table 21-25: Operating cost for the pelletizing area

Pelletizing Area	Units	Unit/t	Unit Price	USSD/Year	USD/t
Salaries &Wages	MH	Varies		1,353,584	2.52
Shared Personnel	MH	Varies		502,006	0.94
Refractories	kg	0.06	2.35	75,000	0.14
Natural Gas	mmbtu	0.70	4.50	1,679,890	3.13
Electricity Operations	kWh	132.98	0.03	2,017,407	3.76
Maintenance Supplies and lubricants	NA	-	-	302,335	0.56
Bentonite	kg	16.15	0.10	831,756	1.55
Mineral Limestone	kg	14.53	0.12	924,805	1.72
Mineral Dolomite	kg	27.45	0.12	1,746,853	3.26
Iron Ore Pelletizing Area				**9,433,634**	**17. 58**

The largest item of cost for the Pelletizing area, is electricity, labour, followed by Mineral Dolomite, electricity, and natural gas.

21.5.2.3. Energiron® Direct Reduction Area

Table 21-26: Operating cost for the direct reduction area

Energiron® DR Area	Units	Unit/t	Unit price	USD/Year	USD/t
Salaries &Wages	MH	Varies		2,316,983	4.32
Shared Personnel	MH	Varies		836,677	1.56
Natural Gas	mmbtu	8.77	4.50	21,183,496	39.49
Electricity Operations	kWh	117.35	0.03	1,780,295	3.32
Maintenance Supplies (1)	NA	-	-	2,000,906	3.73
Nitrogen				1,195,904	2.23
Oxygen				2,076,344	3.87
Portland Cement	kg	5.06	0.18	491,350	0.92
DMDS	kg	0.08	3.00	128,756	0.24
MDEA	kg	0.04	6.00	128,756	0.24
Energiron® DR Area				**32 139 466.13**	**59.91**

Notes:

(1) Natural Gas is the largest cost item, followed by Salaries & Wages, oxygen, maintenance and electricity.

21.5.2.4. OSBF Melting Area

Table 21-27: Operating cost for the OSBF melting area

OSBF Area	Units	Unit/t	Unit price	USD/Year	USD/t
Salaries &Wages	MH	Varies		3,237,858	6.04
Shared Personnel	MH	Varies		1,255,015	2.34
Electrode paste (OSBF)	kg	2.55	0.50	685,091	1.28
Refractories (OSBF)	kg	0.49	3.11	819,318	1.53
Refractories ladles	kg	0.98	2.64	1,392,840	2.60
Electricity Scrubber	KwH	17.70	0.03	268,505	0.50
Electricity Conversion - OSBF	KwH	857.39	0.03	13,007,408	24.25
FeSi for Ti-slag treatment	NA	2.62	1.35	1,888,992	3.52
Natural Gas ladles/launder	mmbtu	0.16	4.50	353,359	0.66
Anthracite	kg	5.30	0.18	509,019	0.95
Natural Gas (furnace and scrubber)	mmbtu	0.00	4.50	2	0.000004
Nitrogen (scrubber)				1,063	0.002
Oxygen				1,859	0.003
Maintenance Supplies (1)	NA	-	-	393,008	0.73
Electricity Off gas	KwH	12.57	0.03	190,762	0.36
Tap hole clay	litres	0.07	1.04	37,323	0.07
Electrode Casings	Each	0.00	1 406.00	456,950	0.85
OSBF Area and Refractories				**24,498,371.68**	**45.66**

Notes:

(1) The electricity utilized by the OSBF furnace, followed by labour, is the largest cost item for this section.

21.5.2.5. Secondary Metallurgy – Converter

Table 21-28: Operating cost for the secondary metallurgy – Converter area

Secondary Metallurgy (Converter)	Units	Unit/T	Unit Price	USD/Year	USD/t
Salaries &Wages	MH	Varies		451,195	0.84
Shared Personnel	MH	Varies		167,335	0.31
Refractories (CLU)				3,755,370	7.00
Fe2O3 fines	kg	14.06	0.080	603,278	1.12
Electricity CLU Operations	KwH	4.85	0.028	73,505	0.14
Anthracite	kg			2,672,471	4.98
Natural Gas for Seals	mmbtu	0.01	4.50	25,123	0.05
Oxygen				785,859	1.46
Maintenance Supplies				100,262	0.19
Nitrogen				1,205,770	2.25
Bauxite	kg		0.10	334,453	0.62
Al	kg		2.50	394,402	0.74
Lime	kg	0.76	0.12	49,221	0.09
FeSi Fines	kg	0.24	1.345	169,751	0.32
FeSi Coarse	kg	2.94	1.345	2,121,885	3.96
Secondary Metallurgy (CLU®)				**12,909,881.73**	**24.06**

The most important cost items for this part of the process are refractories for the CLU® operation, then carbon, FeSi, Nitrogen, Oxygen and Fe_2O_3.

21.5.2.6. Pig Casting

Table 21-29: Operating cost for pig casting area

Pig Casting	Units	Unit/T	Unit Price	USD/Year	USD/t
Salaries &Wages	MH	Varies		451,195	0.84
Shared Personnel	MH	Varies		167,335	0.31
Refractory	kg	0.49	2.64	699,188	1.30
Other consumables				53,648	0.10
Natural Gas Granshot®	mmbtu	0.11	4.50	263,100	0.49
Electricity Operations	kWh	44.11	0.028	669,127	1.25
Maintenance Supplies Casting				54,377	0.10
Pig Casting				**2,357,970.36**	**4.40**

The most important cost items for this part of the process are electricity, refractories, followed by labour.

21.5.2.7. Iron Plant Auxiliaries

Table 21-30: Operating cost for the iron plant auxiliaries area

Iron Plant Auxiliaries	Units	Unit/t	Unit price	USD/Year	USD/t
Maintenance Supplies		Varies		318,488	0.59
Make-up Water (whole plant)	m^3	4.32	0.408	945,440	1.76
General services/mobile equipment				972,524	1.81
Electricity	kWh	126.87	0.03	2,078,740	3.59
Ti-slag Credit		1.00	3.52	-1,888,992	-3.52
V-slag Credit		1.00	24.06	-12,909,882	-24.06
Transport of the material from Plant		Varies		3,755,370	7.00
Iron Plant Auxiliaries				**8,070,561.85**	**14.76**

The most important cost items for the auxiliaries are the transport of the HPPI from the plant (EXW) to on board of the ship (FOB), then electricity.

21.5.2.8. Pig Iron Production Cost Summary by Process Area

Table 21-31: Pig iron summary cost by process area

Summary per Process Area	%	USD/Year	USD/t
Raw Material Receiving Area	52%	88,181,066	164.37
Iron Ore Pelletizing Area	6%	9,433,634	17.58
Energiron® DR Area	19%	32,139,466	59.91
OSBF Area and Refractories	15%	24,498,372	45.66
Secondary Metallurgy (Converter)	8%	12,909,882	24.06
Pig Casting	1%	2,357,970	4.40
Iron Plant Auxiliaries	5%	8,070,562	14.76
Ti - Slag Credits	-1%	-1,888,992	-3.52
V - Slag Credits	-8%	-12,909,882	-24.06
Other	4%	6,081,685	11.34
Total		**168,873,762.36**	**314.49**

The total operation cost for the plant is 168.87MUSD per year and 314.49 USD per metric ton considering a total production of 536,481 tonnes of HPPI. More than 40% of this cost is raw materials. The second area of cost is direct reduction, followed by the melting area.

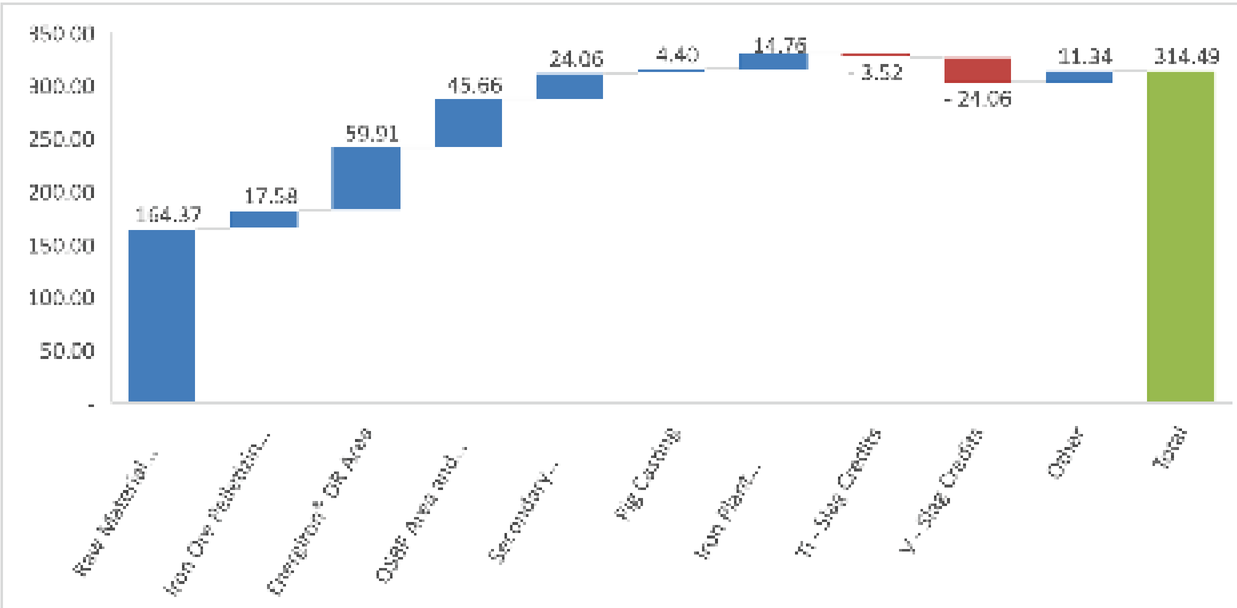


Figure 21-5: Waterfall diagram for the pig iron cost

21.5.2.9. Pig Iron Production Cost Summary by Item

Table 21-32: Pig iron summary cost by item

Summary Metallurgical Plant	%	USD/Year	USD/t
VTM-Concentrate	51%	86,672,756	161.56
Natural Gas	14%	23,241,869	43.32
Electricity	12%	20,025,516	37.33
Labour - ex SGA	6%	10,788,979	20.11
Refractories	4%	6,079,851	11.33
Gas Plant(O_2&N_2), DMDS, MDEA, CO_2	4%	6,015,662	11.21
Transport to the ship	2%	3,755,370	7.00
Maintenance (hardware)	2%	3,648,988	6.80
Lime, Dolomite Bentonite	2%	3,503,413	6.53
Carbon	2%	3,108,920	5.80
FeSi for Ti-slag treatment	1%	1,888,992	3.52
Fe_2O_3 fines	0%	603,278	1.12

Summary Metallurgical Plant	%	USD/Year	USD/t
General services/mobile equip./Waste disposal	1%	972,524	1.81
Industrial Water	1%	945,440	1.76
Electrodes	0%	685,091	1.28
CLU refinement to HPPI/Nodular Pig Iron	2%	3,142,284	5.86
Pig Caster versus Granulator (FS)	1%	2,357,970	4.40
SGA	1%	1,168,334	2.18
Property Lease	0%	433,218	0.81
Outsource Services	5%	4,480,132	8.35
Ti-slag credit	-1%	-1,888,992	-3.52
V-slag credit	-8%	-12,909,882	-24.06
Total		**168,719,714**	**314.49**

As already seen in the previous paragraphs, more than 51% of the production cost is made of VTM Concentrate. The second-largest cost item is NG needed for the DR operation, and the third-largest cost item is electricity. These three items account for about three quarters of the total cost.

The price assumptions outlined in Section 21.3.1 and used in this section to calculate the operating costs, are based on current market conditions for natural gas. The cost for electricity in Québec is based on the Economic Development Rate for Large-Power Customers.

Labour, excluding wages for SG&A personnel, is the fourth cost item, accounting for 8% of the overall production cost.

21.5.2.10. TiO_2-slag

Table 21-33: Operating cost for the TiO_2 slag treatment area

Summary Titanium Plant	USD/Year	USD per ton of TiO_2 slag
Titania Slag from Iron Plant	1,888,992	15.65
Mobile Equipment	2,051,582	17.00
Crushing, milling and magsep	920,225	7.47
Total Cash TiO_2-slag Production Cost	**4,860,800**	**40.12**

For the production of TiO_2-slag the only three cost items that can be associated with it are the actual cost of the FeSi used for its treatment, here indicated with Titania slag cost (equal to the full cost of FeSi credit indicated above in this chapter), the cost for the handling and transportation of the slag and the crushing, milling and magnetic separation. This operation of slag handling is considered as contracted to a third party, same as the handling of the HPPI from plant to ship. In fact, it has been used a similar unit cost (10 USD/t). In addition to that it has to be considered the transport of the TiO_2-slag to the port and load onto a ship (7 USD/t), for a total of 17 USD/t for Mobile equipment and handling of material.

It also has to be noted that Fe-V-Mn-Si alloy is a by-product of the FeSi treatment of the Titania slag and it will sell as additional revenue stream for the plant.

21.5.2.11. Vanadium Slag

Table 21-34: Operating cost for the vanadium slag

Summary Vanadium Plant	USD/Year	USD per ton of V slag
Labour - V-slag - Crushing, Milling and bagging	1,002,491	25.15
V-slag from Iron Plant	12,909,882	323.86
Crushing, Milling, Screening & bagging	1,380,337	34.63
Total Cash V-slag Production Cost	**15,292,710**	**383.63**

For the production of Vanadium slag the cost associated with the converter and all the additives are considered in the cost as indicate V-slag from Iron plant. The only other two aspects associated with the vanadium slag is the Labour and the operational cost of the crushing, milling, screening and bagging of the vanadium slag.

The Vanadium slag is not transformed to FeV at the metallurgical transformation plant but would be transformed at a third-party vanadium operation on behalf of BRM.

22. Economic Analysis

22.1. Financial Analysis

The economic evaluation of the BlackRock Project was performed using a discounted cash flow model on a pre-tax and post-tax basis. The capital and operating cost estimates, as presented in Chapter 21 were used as input into the model. The Internal Rate of Return (IRR) on total investment was calculated based on 100% equity financing. The Net Present Value (NPV) resulting from the net cash flow generated by the Project was calculated based on a discount rate varying between 0% and 10%. The Project Base Case NPV was calculated with a discount rate of 8%. The payback period is also indicated as a financial measure. Furthermore, a sensitivity analysis was also performed to assess the impact of a ±25% variation in the initial capital cost, annual operating costs and final products selling price of the Project. On the revenue side, a decrease in the relative strength of the Canadian dollar with respect to the American dollar will be beneficial to the project financials.

The Financial Analysis was performed with the following assumptions and basis:

- All prices and costs are listed in Canadian dollars unless specified otherwise;
- High Purity Pig Iron selling price was derived from the forecast provided by Wood Mackenzie. An average price of USD786/t was used;
- Ferrovanadium average selling price of USD38.17/kg was used;
- Titanium Slag average selling price of USD300/t was used;
- The construction period is approximately 3.5 years and the production life is 39 years;
- The project financial analysis is carried out using a constant money basis;
- Where applicable, the exchange rate assumed is $1.00 CAD = $0.76 USD;
- The mine reclamation costs have been disbursed over 3 years as follows:
  - Year -1: 50%
  - Year 0: 25%
  - Year 1: 25%

22.1.1. Capital Investment Costs, Disbursement Schedule and Allowance

Capital costs were calculated for the mine and Beneficiation Plants. Table 22-1 and Table 22-2 show the breakdown of the direct and indirect costs.

Table 22-1: Beneficiation Plant capital costs summary

Area	Area Description	Total (CAD $M) 2022
0000	Off-site – excluding rail preparation	4.489
0000	Off-site – rail preparation	3.124
1000	Infrastructures	10.413
2000	Administration and Services	12.082
3000	Mine (excludes all mine equipment)	15.272
4000	Crushing	23.301
5000	Stockpiling and Conveying	22.356
6000	Processing Plant and Load-out System	139.578
7000	Tailings and Water Management	28.832
Subtotal Direct Costs		259.446
8000	Owner's Costs	42.766
9100	EPCM Services	13.657
9200	Construction Indirects	8.553
9500	Commissioning	3.058
9900	Common Distributables (Freight, spares)	10.371
Subtotal Indirect Costs		78.406
9800	Contingencies	35.304
Grand Total		**373.156**

Table 22-2: Metallurgical Plant capital costs summary by area

Area	Area Description	Total (CAD $M) 2022
00000	General	29.6
01000	VTM Concentrate and Pelletizing Plant	176.2
02000	Direct Reduction Plant	346.0
03000	OSBF Electrical Furnace	78.7
04000	OSBF Furnace Off-gas Treatment	12.6
05000	Auxiliary Plants	54.4
06000	Hot Metal and Slag Handling	129.5
07000	Electrical General Systems, Automation and Controls	39.5
08000	Administration and Ancillary Facilities	8.4
Subtotal Direct Costs		**875.0**

Area	Area Description	Total (CAD $M) 2022
	Owner's Costs	57.9
	EPCM Services	29.7
	Construction Tempo Facilities and Site Maintenance	19.3
	Professional Services – Third Party	2.9
	Commissioning Services	6.7
	Common Distributables (freight, spares, initial fill, tech assistance)	14.7
Subtotal Indirect Costs		**131.4**
	Contingency	91.5
Total Costs		**1 097.9**

A working capital of $21.9 M will be required in the first full year of operations. Ongoing costs are disbursed through the operation of the mine.

22.1.2. Operating Costs

Operating costs for the Beneficiation Plant were calculated for mining, processing, general and administrative costs, leasing, concentrate transport logistics and other costs. Processing operating costs were calculated based on concentrator labour, consumables, maintenance and electricity. General and administrative costs cover all costs not included in mining, processing and transportation costs. The operating costs are summarized in Table 22-3.

Table 22-3: Estimated Beneficiation Plant operating costs summary

Sector	Operating Costs Year 1 to 10		Operating Costs Life of Mine (LOM)	
	$M/y	$/t Milled	$M/y	$/t Milled
Mining	47.7	13.3	56.0	14.8
Process	27.0	8.6	27.6	8.5
General and Administration	8.6	2.6	8.6	2.6
Leasing	3.0	0.2	1.4	0.5
Transport Logistics	25.2	7.7	25.2	7.7
Iron Concentrate Rail Car Maintenance	0.1	0.05	0.2	0.05
Other (coarse tailings, environmental)	0.6	0.2	0.6	0.2
Total	**112.2**	**32.7**	**119.6**	**34.4**

Operating costs for the metallurgical secondary transformation plant were calculated by area for each produced product. Processing operating costs were calculated based on metallurgical plant processing labour, consumables, maintenance, and electricity. General and administrative costs cover all costs not included in the process operating costs. The operating costs are summarized in Table 22-4, Table 22-5, and Table 22-6.

Table 22-4: Estimated pig iron production operating costs summary

Summary per Process Area	%	Cost/Year	Cost/t
Raw Material Receiving Area	52%	88,181 066	164.37
Iron Ore Pelletizing Area	6%	9,433 634	17.58
Energiron® DR Area	19%	32,139 466	59.91
OSBF Area and Refractories	15%	24,498 372	45.66
Secondary Metallurgy (Converter)	8%	12,909 882	24.06
Pig Casting	1%	2,357 970	4.40
Iron Plant Auxiliaries	5%	8,070 562	14.76
Ti - Slag Credits	-1%	-1,888 992	-3.52
V - Slag Credits	-8%	-12,909 882	-24.06
Other	4%	6,081 685	11.34
Total		**168,873 762.36**	**314.49**

Table 22-5: Estimated TiO_2 slag production operating costs summary

Summary titanium plant	Cost/ Year	Per ton of TiO_2 slag
Titania Slag from Iron Plant	1,888,992	15.65
Mobile Equipment	2,051,582	17.00
Crushing, milling and magsep	920,225	7.47
Total Cash TiO_2 slag Production Cost	**4 860 800**	**40.12**

Table 22-6: Estimated vanadium slag production operating costs summary

Summary vanadium plant	Cost/ Year	Per ton of V slag
Labour - V-slag - Crushing, Milling and bagging	1,002 491	25.15
V-slag from Iron Plant	12,909 882	323.86
Crushing, Milling, Screening & bagging	1,380 337	34.63
Total Cash V slag Production Cost	15,292 710	383.63

22.1.3. Annual Project Cash Flows

The annual project cash flows are summarized in Table 22-7.

Table 22-7: Annual project cash flows

Annual Project Cash Flows (CAD $M)	Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11 - Y20	Y21 - Y30	Y31 - Y39
Production																	
VTM Concentrate (kt)	**32,719**	–	–	–	191	856	856	856	856	856	856	856	856	856	8,560	8,560	7,704
High Purity Pig Iron (kt)	**20,506**	–	–	–	120	536	536	536	536	536	536	536	536	536	5,365	5,365	4,828
FeV (kt)	**170**	–	–	–	1	4	4	4	4	4	4	4	4	4	44	44	40
Ti Slag (kt)	**4,613**	–	–	–	27	121	121	121	121	121	121	121	121	121	1,207	1,207	1,086
Sales																	
High Purity Pig Iron	**20,960**	–	–	–	98	463	507	509	504	506	513	523	534	552	5,587	5,612	5,051
FeV	**8,444**	–	–	–	50	221	221	221	221	221	221	221	221	221	2,209	2,209	1,988
Ti Slag	**2,209**	–	–	–	21	58	58	58	58	58	58	58	58	58	576	576	518
Total Sales	**31,613**	**–**	**–**	**–**	**168**	**741**	**786**	**788**	**782**	**785**	**791**	**802**	**812**	**830**	**8,372**	**8,397**	**7,558**
Operating Costs																	
High Purity Pig Iron	**8,416**	–	–	–	61	210	215	214	215	209	210	215	218	241	2,378	2,162	1,870
FeV	**3,006**	–	–	–	19	84	84	84	84	84	78	78	78	78	777	777	699
Ti Slag	**260**	–	–	–	2	7	7	7	7	7	7	7	7	7	68	68	61
G&A, Sales, Marketing, Other	**1,273**	–	–	–	11	47	48	48	48	48	48	48	32	32	320	242	300
Total Operating Costs	**12,954**	**–**	**–**	**–**	**92**	**347**	**354**	**353**	**354**	**348**	**342**	**348**	**334**	**358**	**3,543**	**3,250**	**2,931**
Total Capex	**2,290**	**51**	**527**	**737**	**266**	**19**	**18**	**13**	**15**	**12**	**14**	**13**	**44**	**28**	278	151	105
Pre-Tax Cash Flow	**16,369**	**(51)**	**(527)**	**(737)**	**(190)**	**375**	**414**	**422**	**414**	**425**	**435**	**440**	**435**	**445**	**4,552**	**4,997**	**4,522**
After Tax Cash Flow	**12,055**	**(51)**	**(527)**	**(737)**	**(190)**	**374**	**360**	**326**	**318**	**325**	**331**	**334**	**326**	**336**	**3,425**	**3,743**	**3,363**

22.1.4. Determination of Internal Rate of Returns & Cumulative Cash Flow

The Project IRR, cumulative undiscounted cash flow, NPV at an 8% discount rate, as well as the simple payback period are presented in Table 22-8. Additional NPV values at discount rates of 0, 5, 8 and 10% are shown in Table 22-9.

Table 22-8: IRR and cumulative cash flow values (CAD$ M)

Pre-tax IRR	Pre-tax Cashflow	Pre-tax NPV @	Payback Period (Years)
21.5%	$16,369	$2,854	4.7
Post-tax IRR	**Post-tax Cashflow**	**Post-tax NPV @**	**Payback Period (Years)**
18.2%	$12,055	$1,932	5.4

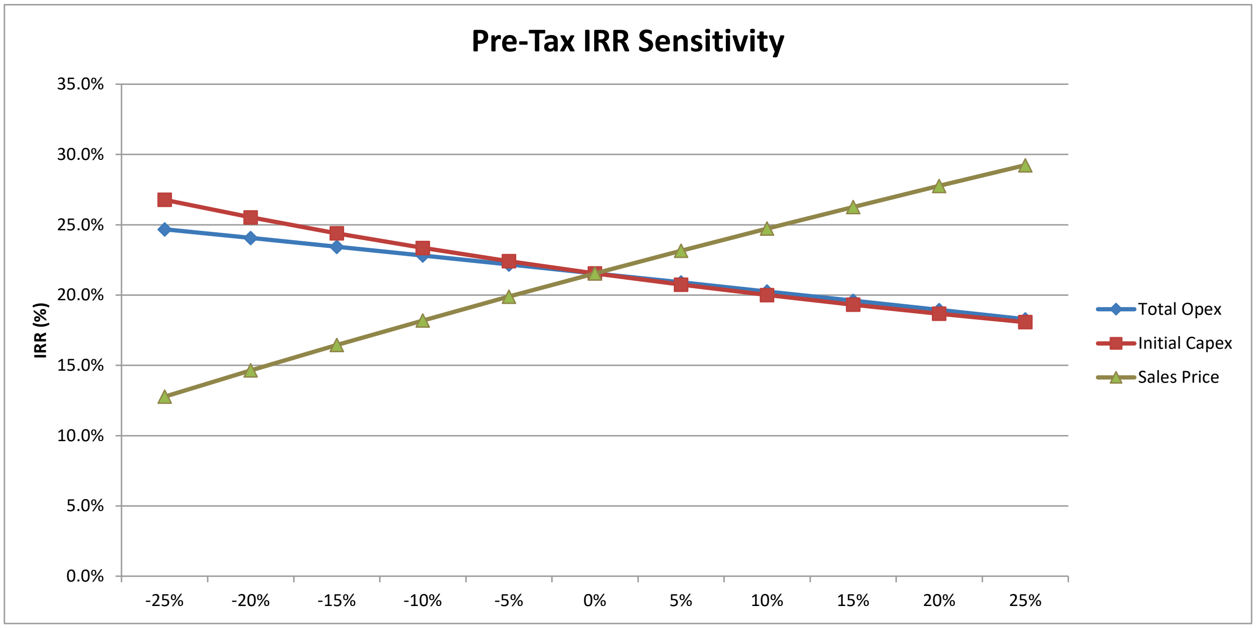
Table 22-9: NPV Values at various discount rates (CAD$ M)

Discount Rate	Pre-Tax NPV	Post-Tax NPV
10%	$1,933	$1,241
8%	$2,854	$1,932
5%	$5,222	$3,708
0%	$16,369	$12,055

22.1.5. Sensitivity Analysis

There are three major parameters that affect the net cash flow: product sales, operating costs and capital expenditures. A sensitivity analysis was performed whereby the IRR is computed by varying the input data in the financial model in order to determine the impact on the overall profitability of the project. Each of the data elements was changed independently of one another.

The results of the sensitivity analysis of the IRR and pre-tax NPV at an 8% discount rate are presented in graphical form in Figure 22-1 and Figure 22-2 respectively. The analysis shows that the project is most sensitive to changes in selling price followed by operating costs and by capital expenditures.



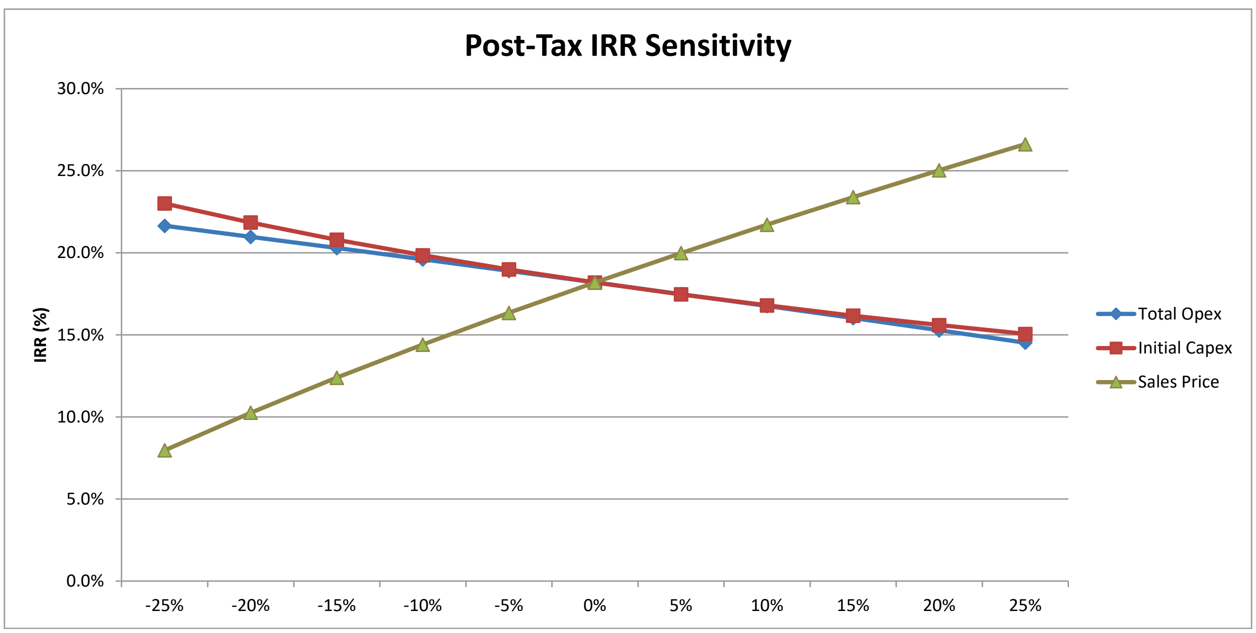


Figure 22-1: IRR Sensitivity analysis

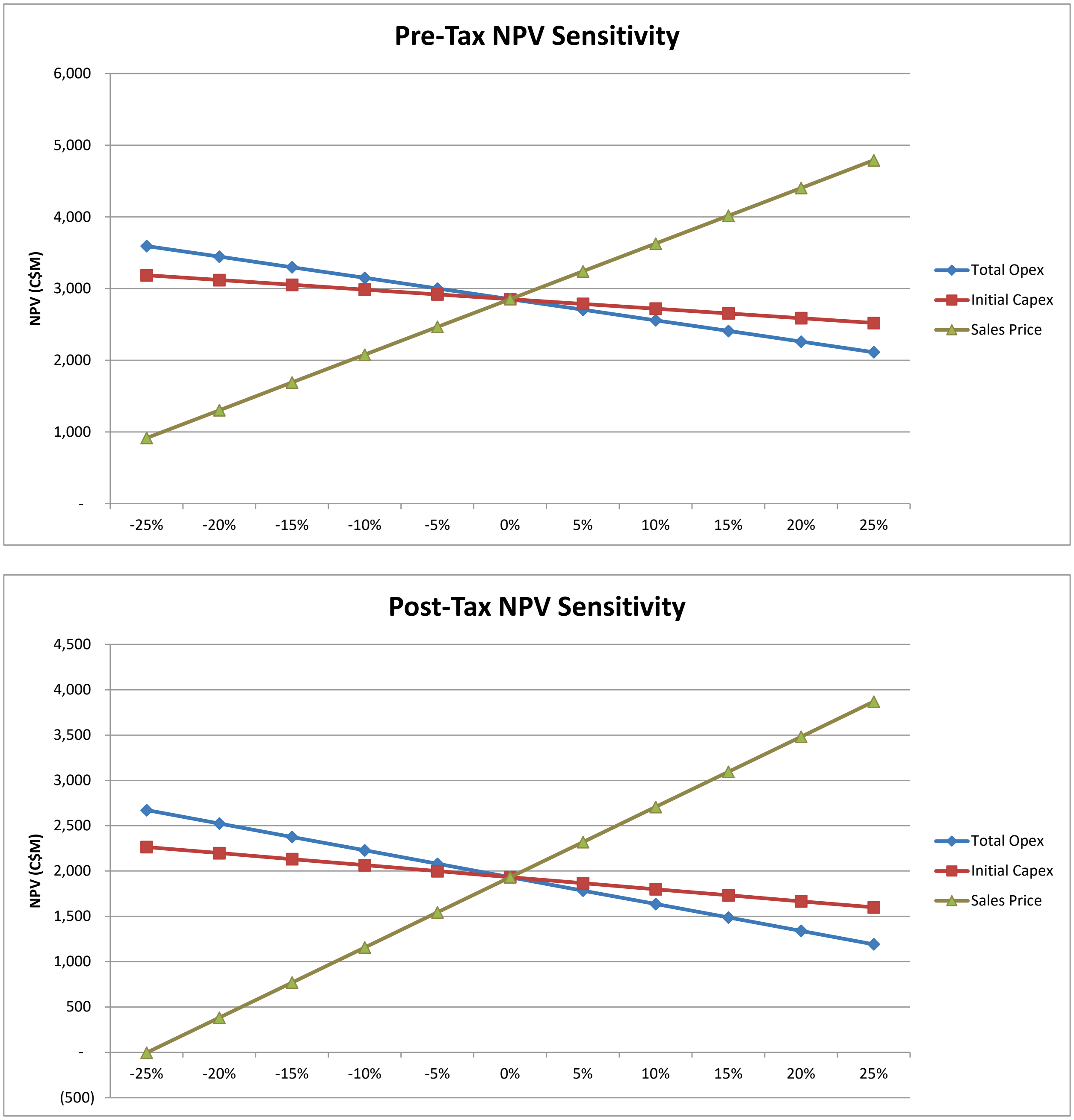


Figure 22-2: NPV Sensitivity analysis (at 8%)

22.1.6. Taxation

Corporate income tax is payable to the Federal Government of Canada pursuant to the Income Tax Act (Canada). The statutory federal income tax rate is 15% of taxable income.

In computing taxable income from a business for Canadian income tax purposes, a taxpayer is permitted to deduct various amounts with respect to expenditures made in the course of the business.

Any mining taxes paid in the year are deductible in computing a taxpayer's net income for tax purposes for that year.

Corporate income tax is also payable to Revenue Québec under the Québec Taxation Act. The statutory provincial income tax rate in Québec is 11.5% of taxable income. Taxable income for Québec income tax purposes is computed in a similar manner as it is for federal income tax purposes, and any mining taxes paid in the year are deductible in computing a taxpayer's net income for tax purposes for that year.

The applicable marginal tax rates under the Québec mining tax regulations are 16%, 22% and 28% of taxable income and depend on the profit margin.

23. Adjacent Properties

BlackRock's Southwest and Armitage deposits are surrounded by multiple exploration projects at various stages of exploration and for various commodities. Figure 23-1 shows the relationship between BlackRock's project (in **dark red**) and the surrounding explorers. Information is gathered from the GESTIM website and is up to date as of January 22nd, 2023.

Following is a brief description of the surrounding explorers and their projects, detailing their commodity of interest. The information relayed below comes directly from the explorer's website and has not been verified for accuracy by the QP. The readers are invited to visit the websites of the explorers.

In the immediate vicinity of the BlackRock Property, there are at least three explorers (VanadiumCorp Resource, Voyager Metals Inc. and Spearmint Resources) looking at Vanadium-Titanium-Iron deposit similar to BlackRock's. It is important to note that the QP for this report has been unable to verify the information below and that the information is not necessarily indicative of the mineralization on the BlackRock Property that is the subject of the technical report.

VanadiumCorp Resource (TSX.V:VRB.V) (In **TURQUOISE** on Figure 23-1)

(https://www.vanadiumcorp.com/about/) VanadiumCorp Resource Inc. is an integrated mining and technology company developing wholly-owned vanadium-titanium-iron mineral deposits situated in Quebec, Canada. The Lac Doré Project's Main claims block is surrounded by Blackrock claims with a narrow strip (400 m) of claims belonging to BlackRock separating it from the Lac Doré North claims block to the north, and a broader strip (~1,300 m) to the east separating the Lac Doré Main claims from the Lac Doré North claims. VanadiumCorp issued a NI 43-1010 Technical Report in December 2020 (available on their website or from SEDAR). The QP for this report has been unable to verify the information from the VanadiumCorp website and technical report; the information is not necessarily indicative of the mineralization on the BlackRock Property that is the subject of this technical report.

Table 23-1: VanadiumCorp MRE at Lac Doré (2020)

Mineral Resource Statement at Lac Doré in In-Situ Mineralization
EFFECTIVE OCTOBER 27 2020

Category	Mineralized material (Mt)	V2O5 grade (%)	Fe grade (%)	TiO2 grade (%)	Magnetite (%)	V2O5 (tonnes)	Fe (Mt)	TiO2 (Mt)	V2O5 contained (million lbs)
Measured (M)	23.98	0.5	33.7	9.9	34.5	128000	8.1	2.4	280
Indicated (I)	190.96	0.4	26.2	6.7	23.4	837000	50.2	12.8	1850
M+I	214.93	0.4	27.1	7.1	24.6	965000	58.3	15.2	2120
Inferred	86.91	0.4	28	7.6	25.9	387000	24.4	6.6	850

Mineral Resource Statement at Lac Doré in Magnetite Concentrate
EFFECTIVE OCTOBER 27 2020

Category	Magnetite conc. (Mt)	V2O5 grade in conc. (%)	Fe grade in conc. (%)	TiO2 grade in conc. (%)	V2O5 in conc. (tonnes)	Fe in conc. (Mt)	TiO2 in conc. (Mt)	V2O5 contained in conc. (million lbs)
Measured (M)	8.27	1.2	62	5.4	100000	5.1	0.8	220
Indicated (I)	44.7	1.3	62	8.5	578000	27.7	3.8	1270
M+I	52.82	1.3	62	8.7	678000	32.8	4.6	1490
Inferred	22.55	1.2	62	8.2	277000	14	2.1	610

Note: Table comes directly from VanadiumCorp website and has not been verified by the QP of this report.

Voyager Metals Inc. (TSX.V:VONE) (In **PINK** on Figure 23-1)

(https://voyagermetals.com/) is a Canadian company publicly listed on the TSX.V Toronto. Mont-Sorcier Iron Ore and Vanadium project, just a short drive from the town of Chibougamau, Quebec. The Mont-Sorcier Iron ore property hosts a large high quality Iron resource with significant and extractable Vanadium in a top tier mining jurisdiction. The Mont-Sorcier Iron-Vanadium deposit consists of a North and South deposit that boast exceptionally low Titanium content, allowing for simple extraction of the Vanadium metal by a blast furnace, making the deposit and possible product unique in the world markets. The following MRE table comes directly from the Voyager Metals website. It has not been checked by the QP of this report. Readers are encouraged to visit the Voyager Metals website for more details on the parameters, assumptions and methods used to prepare the MRE.

Table 23-2: Voyager Metals MRE at Mont-Sorcier (2022)

Zone	Category	Tonnage				Head grade									Conc.
		Rock (Mt)	Fe Rec (%)	Wrec (%)	Conc. (Mt)	Fe2O3 (%)	Fe (%)	Fe3O4 (%)	Fe mag (%)	V_2O_5 (%)	TiO_2 (%)	MgO (%)	SiO_2 (%)	%S (%)	*Fe (%)*
North	Indicated	559.3	72.1	29.2	163.4	37.70	26.35	28.20	19.72	0.21	1.10	19.8	25.1	1.12	65
	Inferred	470.5	73.0	27.4	128.9	34.90	24.40	26.41	18.47	0.18	1.32	19.8	27.9	0.49	65
South	Indicated	119.2	82.0	26.9	32.0	30.43	21.27	25.64	17.93	0.17	1.49	24.1	24.4		65
	Inferred	76.2	81.4	25.2	19.2	28.83	20.15	24.11	16.86	0.13	1.46	22.4	23.1		65
Total	Indicated	678.5	73.5	28.8	195.4	36.4	25.5	27.8	19.4	0.20	1.17	20.5	25.0		65
	Inferred	546.6	74.0	27.1	148.1	34.1	23.8	26.1	18.2	0.17	1.34	20.2	27.3		65

Note: Table comes directly from Voyager Metals website and has not been verified by the QP of this report.

The Voyager Metals MRE has defined an Indicated Mineral Resource in the North Zone of 559M tonnes grading 28.2% magnetite and 0.21% V_2O_5, with an additional 470M tonnes grading 26.4% magnetite and 0.18% V_2O_5 in the Inferred category up from 809M tonnes of Inferred Resources in the prior Mineral Resources Estimate. This represents a total Mineral Resource tonnage increase of 31% in the North Zone. Total Indicated Resources for both the North Zone and the South Zone are now estimated at 679M tonnes grading 27.7% magnetite and 0.20% V_2O_5, with the potential to produce 195M tonnes of magnetite concentrate grading at least 65% Fe and 0.52% vanadium pentoxide (V_2O_5). Total Inferred Resources for both the North Zone and the South Zone are now estimated at 547M tonnes grading 26.1% magnetite and 0.17% V_2O_5, with the potential to produce 148M tonnes of magnetite concentrate grading at least 65% Fe and 0.52% vanadium pentoxide (V_2O_5).

The QP for this report has been unable to verify the information from the Voyager Metals website and technical report; the information is not necessarily indicative of the mineralization on the BlackRock Property that is the subject of this technical report.

Spearmint Resources (CSE:SPMT) (In **PURPLE** on Figure 23-1)

(https://www.spearmintresources.ca/projects/chibougamau-vanadium-project/) is a Canadian junior resource exploration company dedicated to the aggressive pursuit of world-class mineral deposits. Spearmint Resources Inc. has acquired a 100-per-cent interest in the 6,990 contiguous acre Chibougamau vanadium project, located in the direct vicinity of Lac Chibougamau, Quebec. This property directly borders BlackRock Metals' Ilmenite vanadium project, Vanadiumcorp Resource Inc.'s (VRB.v) Lac Dore vanadium project and Voyager Metals's (VONE.v) Mont Sorcier vanadium project. This block of claims puts Spearmint in the heart of the expanding Chibougamau mining district's vanadium circle. Vanadium has garnered significant market attention recently as the use of vanadium in battery storage is gaining momentum and Spearmint is very pleased to be able to acquire a significant footprint in one of Canada's best-known vanadium districts. Except for some airborne geomagnetic geophysical (drone and heli-borne) work done in November-December 2018, it appears no other work has been done on the property and/or no technical reports issued.

In the immediate vicinity of the BlackRock Property, there are at least four explorers looking at base and precious metals deposits. There are also a host of smaller mining claim owners, too many to list. It is important to note that the QP for this report has been unable to verify the information of these website and technical reports and that the information is not necessarily indicative of the mineralization on the BlackRock Property.

Ressources Yorbeau (TSX:YRB) (In **ORANGE** on Figure 23-1)

(https://www.yorbeauresources.com/en/projects/lemoine/) is a Canadian public company exploring for base and precious metals. Their Lemoine project is located just south of, and adjacent to BlackRock Metals Fe-Ti-V mining project. The Lemoine project, located 25 km south of Chibougamau, encloses the former producing Lemoine Mine which produced 0.76Mt of high grade massive sulphide ore (>$600/t at today's prices). With grades of 4.2% Cu, 9.6% Zn, 4.2 g/t Au and 83 g/t Ag, Lemoine ranks number 2 in Canada (second only to Eskay Creek) and is within the top 1% of all VMS worldwide. The property encloses a 17 km segment of the favourable horizon hosting the deposit (Waconichi formation). No technical report available from website.

Delta Resources (TSX.V:DLTA) (In **GOLD** on Figure 23-1)

(https://www.deltaresources.ca/delta-2-gold-project/) Delta Resources Limited is a Canadian mineral exploration company focused on growing shareholder value through the exploration of two properties located in the Chibougamau district of Quebec and in the Thunder Bay district of Ontario. The Delta-2 property is located roughly 35 kilometres southeast of the town of Chibougamau. Exploration of Gold targets and Gold-Rich VMS targets for mineralization. No technical report available from website.

Ressources Cartier (TSX.V:ECR) (In **GREEN** on Figure 23-1)

(https://ressourcescartier.com/projects/dollier/) Founded in 2006, Cartier Resources Inc. is an exploration company focused on the Abitibi gold belt in Quebec, one of the best mining territories in the world. Cartier's strategy is to focus their efforts on relatively advanced gold projects with significant potential for increasing resources. The Dollier project is currently in exploration drilling status to explore extensions of the Dollier Gold Deposit.

Doré Copper (TSX.V:DCMC) (In **BLUE** on Figure 23-1)

(https://www.dorecopper.com/en/about-us/overview/) Doré Copper Mining Corp. listed on the TSX Venture Exchange (TSX-V) after completing a Qualifying Transaction in December 2019, establishing itself as a copper-gold explorer and developer in the Chibougamau area of Québec, Canada. The Corporation, through its wholly-owned subsidiary CBAY Minerals Inc., has consolidated a large land package in the prolific Lac Dore/Chibougamau and Joe Mann mining camps. The Corporation's current focus is to implement its hub-and-spoke development strategy by advancing its key high-grade copper-gold brownfield projects towards a restart of operations. Our goal is to achieve an initial production target of +50 M lbs of copper equivalent (or 100,000 oz gold equivalent). Doré Copper has delivered a preliminary economic assessment (PEA) of its hub-and-spoke model in May 2022 and is proceeding with a Feasibility Study. The QP for this report has not verified the information in the 2022 PEA issued by Doré Copper as it does not pertain to similar commodities and geological environment.

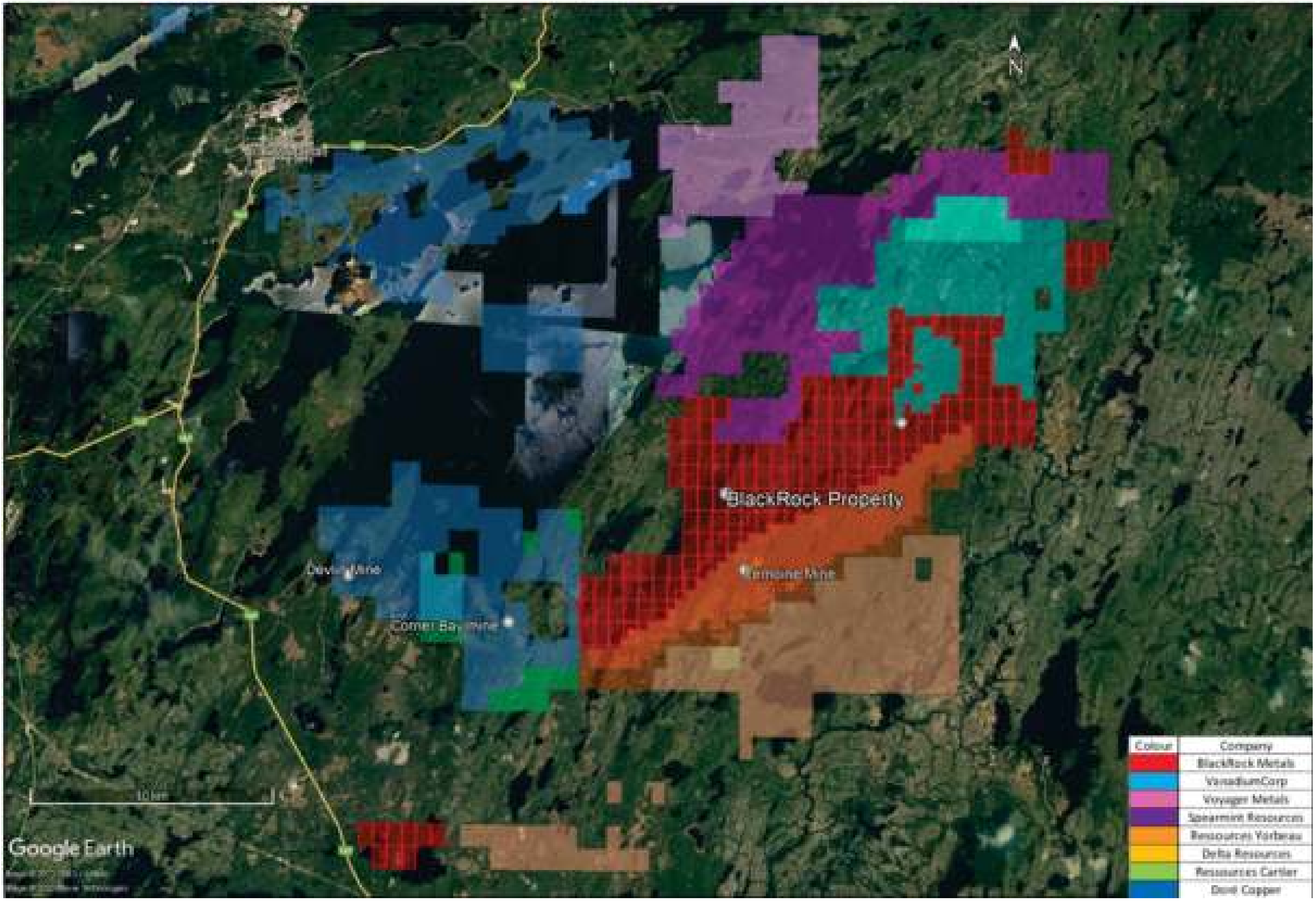


Figure 23-1: Map of BlackRock claims (in Red) and adjacent properties

24. Other Relevant Data and Information

24.1 Project Implementation and Execution Plan

This section of the report provides a summary and general description of the Project Execution Plan upon which the project schedule and the Capital Cost Estimate were developed. The major project milestones are listed in Table 24-1. While they are not necessarily on the critical path, achieving these milestones will demonstrate that the project is moving according to plan.

Table 24-1: Project milestones

Activity	Start Date	Completion Date
Global Milestones		
Provincial Global Certificate of Authorization (Environmental)		December 2013
Environmental Assessment (Fed./Prov.) and Operating Permits		April 2019
Full Notice to Proceed (FNTP)		May 2023
Mine & Beneficiation Plant		
Civil Site Works – Main Access Road and Concentrator Pad	April 2023	June 2023
Initial Pre-Stripping and Dyke Preparation	May 2023	March 2025
Power Permits, Construction and Activation		December 2023
Remaining Project Detailed Engineering	January 2023	October 2023
Mill Construction and Commissioning	September 2023	March 2026
Production Begins	March 2026	
Metallurgical Plant		
Pelletizing Plan OEM Engineering Completion	August 2022	June 2023
Metallurgical Plant Main Vendor Engineering completion	August 2022	December 2023
Remaining Project Detailed Engineering	July 2022	April 2024
Site Preparation Permit Obtained		May 2023
Metallurgical Plant Construction and Commissioning	May 2023	May 2026
Production Begins	July 2026	

The Project Execution Schedule developed in this Study and described herein covers the period from the resumption of Detailed Engineering to the end of the operations ramp-up for the Mine & Beneficiation Plant in Chibougamau and the Metallurgical Plant in Saguenay, Québec.

The major assumptions driving the key milestones for the Mine and Beneficiation Plant are as follows:

- PFDs remain frozen as in October 2017;
- Necessary permits to meet construction schedule:
  - Provincial Global Certificate of authorization has been obtained;
  - All intermediate authorizations needed such as forestry license, land leases, process plant location authorization, access roads, borrow pit exploitation, sewage treatment process for temporary installations, authorization to build foundations;
  - Federal Permits to allow closing of dykes;
  - First Nations Agreements to proceed in place (IBA signed Q3 2013).
- Site pre-development will begin in Q2 2023 with work on the main access roads and the concentrator pad;
- Equipment purchases will be made from Q1 2023 to Q1 2024 to confirm critical engineering information and ensure timely delivery. As some purchase orders will require to be awarded prior to the full availability of financing, those will include a requirement that BlackRock Metals issue a release prior to proceeding with the supplier's fabrication material procurement process;
- Ensure mine Pre-Production starts sufficiently early to produce necessary material for mine roads and dykes construction;
- Pour concrete at the Beneficiation Plant starting in Q3 2023;
- Close the concentrator building by March 2025 to ensure work protected from the elements;
- Complete first tailings dykes enclosure in March 2025;
- Complete HV substation December 2025;
- Complete polishing pond and treatment pond dykes in Q3 2025;
- Start production ramp-up in March of 2026.

The major assumptions driving the key milestones for the Metallurgical Plant are as follows:

- Commitments with Pelletizing Plant and DRI technology upon project notice to proceed;
- Necessary permits to meet construction schedule:
  - Global certificate of authorization obtained as of April 2019;
  - Detailed CA application for construction submitted from Q1 to Q4 2023;
  - Operating Permit CA Application to be received in Q1 2025.

- Equipment purchases will be made from Q1 2023 to Q1 2024 to confirm critical engineering and ensure long lead items will meet there required at site dates;
- Start concrete pours for the DRP tower in October 2023;
- Complete HV substation October 2025;
- Start production ramp-up in July 2026.

24.1.1 Schedule

The master schedule is based on completing the construction and achieving mechanical completion of the Mine & Beneficiation Plant by January 2026, with a full-operation ramp-up of the plant in September 2026 and achieve mechanical completion of the Metallurgical Plant by October 2026, to allow a full-operation ramp-up in December 2026. To achieve these objectives, the following overall construction plan will be implemented.

Beginning with the civil and infrastructure phase for the Beneficiation Plant during which access roads will be refreshed, the main plant and garage pad will be prepared and all access roads to the Southwest Pit will be prepared for 90 t to be used for hauling of fill material. A construction management office will be built, along with a temporary power plant (comprising diesel generators), to supply power to the construction offices.

The following permits are required to begin the mine and Beneficiation Plant construction phase:

- Forestry license to be renewed;
- Mining lease – to be obtained upon closure plan approval;
- Land lease – to be obtained upon closure plan approval;
- Certificate of Authorization - Obtained.

Once these are obtained, clearing and grubbing of required areas can begin for the site access roads, mine roads, process building areas, tailing dykes, as well as the mine site. The mine waste rock will be used as backfill material for all locations from this point onwards.

For the Metallurgical Plant, same approach with the beginning of the civil and infrastructure for the site, during which access roads, main site pad and DRP specific pad will be prepared, along with construction management office and temporary power plant.

The following permits are required to begin the Metallurgical Plant construction phase:

- Forestry license to be obtained;
- Site rental agreement to be obtained;
- Certificate of Authorization - Obtained.

Once these are obtained, clearing, and grubbing of required areas can begin.

24.1.1.1 Beneficiation Plant Backfill Material

Granular backfill material is required for the construction of access roads and mine haul roads and will be sourced from the Southwest Pit waste rock. Qualification of this material by a certified soil laboratory will be required immediately after the notice to proceed.

Specialty backfill material (till, sand) will be sourced from other borrow pits within a 15 km radius of the site.

All other required backfill material will be taken from the open pit pre-development done by BRM that will be crushed and screened to achieve required specifications.

24.1.1.2 Metallurgical Plant Backfill Material

Granular backfill material is required for the construction of the site pad and process building's sub-foundations. Those will be sourced from the actual site levelling, and from local suppliers near the construction site, with an adequate control in place to ensure the material meets quality standards.

24.1.2 Engineering, Procurement, Construction Management (EPCM) Key Elements

To support the construction schedule, the following EPCM activities will be closely monitored.

For the mine and Beneficiation Plant:

- In January 2023, an authorization to proceed from BRM will initiate the restart of the detailed engineering required to begin construction in Q2 2023. This will include civil engineering and engineering for equipment impacting steel and concrete design;
- First engineering drawings and specifications are already issued for construction and are for site preparation and access roads, previously scheduled for July 2018, and now schedule for May 2023;
- Finalize a preliminary Plant General Layout in May 2023 in order to be ready for concrete and steel drawings for construction, by the end of July 2023;
- P&IDs issued for project use as of March 2018;
- Construction drawings for main buildings, major equipment foundations and steel structures to be reviewed and completed by July 2023, allowing for fabrication of main steel to be started erected in November 2023;

- Procurement of construction services will be done with ''issued for construction'' drawings, reducing risks of post award variations in cost and schedule.

For the Metallurgical Plant:

- In January 2023, an authorization to proceed from BRM will initiate the restart of the detailed engineering required to begin construction in May 2023. This will include civil engineering and engineering for equipment impacting steel and concrete design;
- First engineering drawings and specifications that will be issued for construction will be for site preparation and access roads, scheduled for May 2023;
- First P&IDs were issued for project use in January 2019 for DRI and OSBF, remaining P&IDs to be issued for project use by June 2023;
- The first construction drawings for the DRP Tower were issued in March 2021. The other main buildings, major equipment foundations and steel structures shall be issued by July 2023, allowing for fabrication of main steel to be completed and erected by April 2025 for the DRP/OSBF and October 2025 for the Pelletizing Plant;
- Procurement of construction services will be done with issued for construction drawings, reducing risks of post award variations in cost and schedule.

24.1.3 Project Construction Strategy

The mine and Beneficiation Plant Project construction strategy is based on the following key assumptions:

- Availability of local workers, or workers temporarily lodged in Chibougamau, for initial civil and first concrete works;
- Maintaining a low level of direct labour force on site in order to ensure availability of local lodging in hotels and private housing for the mechanical & electrical trades;
- Qualified industrial construction labour will be available;
- Timely approval of all required permits.

The Metallurgical Plant Project construction strategy is based on the following key assumptions:

- Construction work will be executed by local, regional and provincial workers, temporarily lodged in Saguenay;
- Qualified industrial construction labour will be available;
- Timely approval of all required permits.

24.1.3.1 Mine and Beneficiation Plant

24.1.3.1.1 Process Water and Tailings Dam

Timely excavation and sorting of competent material from the Southwest Pit is critical to the project success. Clearing and pre-stripping in the Southwest Pit should begin no later than may of 2023. This will allow for other advanced stripping activities to occur. In this manner, the Project will be able to source the entire necessary structural fill to be used for dams, dykes, roads, ponds, etc. Other borrow pits will be developed to provide backfill material not available in the Southwest Pit (ex.: sand).

24.1.3.1.2 Concentrator

The concentrator is a critical facility and will take the longest to build and commission. Despite this, its construction is not on the project critical path. Power availability for commissioning is the key as it only comes online in July 2025. The strategy for this area is to manage the schedule to minimize the labour force while being ready for commissioning when Hydro-Québec power will be available in July 2025. Construction will begin with site preparation in May 2023 and foundations in July 2023. Steel erection will start in November 2023. This precedes the installation of roofing and siding work to commence soon after.

- The major equipment will start arriving on site in August 2024, in time for mechanical erection work to begin. Piping, electrical and instrumentation installation will follow from August 2024 through to September 2025;
- Completion of the mechanical installation will be achieved in January 2026; thus allowing commissioning and operations ramp up to be completed in April of the same year.

24.1.3.1.3 Crusher and Ore Handling

Concrete pouring will begin in July 2024, pausing for winter and completing in spring 2014. The anticipated delivery period of the jaw crusher is 10 months from the ordering date. The building structural and cladding works will be completed in May 2025 and completion of the mechanical installation work is expected to be finalized for January 2026. Construction of the crusher, reclaim stockpile and the HV Substation are all critical, however, can be built within the concentrator time frame.

24.1.3.1.4 Aggregates Plant

A crushing and screening plant will be installed by a contractor as soon as permits are obtained. It will produce all of the backfill material for access roads; mine haul roads, lay-down areas, construction pads, concrete coarse aggregates and tailings dyke filter material. Aggregates will be sourced from the open pit mine pre-stripping work, with the exception of sand that will be sourced from offsite borrow pits (within a 15 km radius).

24.1.3.1.5 Access, Site and Mine Roads

Site access road construction will begin with 90 t articulated trucks starting in May 2023. Roads to be completed quickly include the construction access roads (10 m or less in width) as well as road 210 that leads to the site (+/-15 km). The backfill material for these works will be taken from the material excavated on the same roads, or from the concentrator pad.

The first phase of development for the mine haul roads will also start in May 2023. Development of the polishing ponds will commence as soon as the access road allows the work to begin and be completed in October 2024 to allow for accumulation of fresh water for the plant start-up in the spring of 2025.

Ninety tonne rigid-chassis trucks will haul the majority of the required fill material to the tailings pond and access roads construction.

24.1.3.1.6 Concrete Supply and Placement

A total quantity of 16,000 m^3 of concrete is required for the construction of the Beneficiation Plant project. During the engineering restart, while finalizing the detailed construction strategy with updated project information, a trade-off study will be completed to validate the benefits of either a dedicated batching plant, installed on site, or having the concrete delivered from Chibougamau.

If the site batch plant option is retained, coarse aggregates will be sourced from the pre-stripping of the Southwest Pit on site to be processed by a combined team of mining operators and contractor and be delivered to stockpiles at the batching plant pad. The concrete batch plant will be completely installed with the shelter and commissioned prior to the first concrete pour in June 2023.

A detailed construction execution plan has been developed in order to meet the start-up dates, the information is included the document BRM-000000-60AC-001.

24.1.3.2 Metallurgical Plant Facilities

24.1.3.2.1 Direct Reduction and Smelting Areas

Concrete pouring will begin in September 2023 and completed in July 2024. The anticipated delivery period of the DR reactor is closely interacting with the tower erection strategy. The DRI and OSBF buildings structural and cladding works will be completed in June 2025 and completion of the mechanical installation work is expected to be finalized for February 2026. Construction of DRP tower is challenging, constructability strategy is developed with a prefabricated structural steel modular approach. Each main module has the full-core area of the tower and includes the major components going in these sections, to be erected above the Open Slag Bath Furnace (OSBF).

24.1.3.2.2 Gas Heater

The gas heater is a facility that requires specialized welders with specific skills since welds are made with nickel and high chrome content. To reduce the risk and optimize the schedule by minimizing the number of welds on site, a modularized delivery strategy is considered as the most plausible alternative. The sections will be brought to site using a similar arrangement than the one used for the reactor. The bottom section will be dropped right over its foundation. The two heat exchanger sections will be installed over the bottom section using a crane. These two lifts will require a considerable crane size but is available in Québec.

24.1.3.2.3 Outside Pipe Racks

Considering the number of pipes racks, it was established to shop fabricate nominal length modules of 18.9 m. The design of the pipe rack's structure will be done to accommodate the transportation of the pipe rack for the roads and finally assemble at site, prior to being lifted in place using the appropriate crane. The intent is to have all the pipes in the pipe rack, tested, painted, insulated including heat tracing, when required. When cable trays need to be installed on the pipe-rack sections, the trays will be installed in the shop prior to shipping to the site.

For the vertical pipe rack on the outside the DRP tower, a similar principle was adopted. The pipe section in the module will be welded but at each end, of a module, it will be flange connection to facilitate the connection module to module while in place.

24.1.3.2.4 Pelletizing and Indurating Areas

This sector is subdivided in six sub areas, storage and mixing, balling building, indurating, electrostatic precipitators, fans & stacks, hearths layer separation and transfer towers.

24.1.3.2.5 Balling Building

The two balling discs will be delivered in sections to be installed in a bay which is covered by a 20 Mt maintenance overhead crane, which will be used for the installation. The structural steel erection could resume after completion of the building and the roof by date.

24.1.3.2.6 Indurating Building

The section of the building, which is north of the indurating machine, from axes 501 to 504 is a conventional multistory industrial building. Most of the equipment could be introduced in the building after the structural steel erection, cladding and roofing are done except for the segment of the reciprocating conveyor (CV-422C) and the wide belt green pellet conveyor (CV-511). The remaining section of the building host the indurating machine. Building construction completion will be completed by June 2025.

24.1.3.2.7 Hearth Layer Separation and Transfer Towers

This is rather a small and straight forward building. The grizzly, the grizzly chutes and the hearth layer separation bin (BN-602) should be dropped on their respective supporting floor has the structural steel erection progress. Transfer tower, chutes and conveyors are of regular yard installation of small conveyors.

Concrete Supply and Placement

For the metallurgical complex, a total quantity of 38,400 m^3 of concrete is required for the construction of all the process buildings. Supply of concrete will be from local suppliers in Ville La Baie and Chicoutimi, these plants will be able to supply the concrete requirements as requested by the contractors. A frame agreement is also planned to guarantee prices for the contractors.

Access, Site Roads

Site existing conditions that allow site access and infrastructure for construction and delivery are as follows:

- The site is owned by the Federal Port Authorities. Port facilities are within 2 km and accessible by boat all year-round;
- The port is adjacent to La Baie City, which is agglomerated with two other cities under the designation of Ville de Saguenay. The total population of Saguenay is around 145 000 citizens. The Lac-Saint-Jean region located within 100 km and the Saguenay area, has a total population of over 275 000 citizens;
- Saguenay is connected by a 230 km highway to the Capital Québec City with a population of over 530 000 citizens.

Site access road construction will begin at the same time as the site pad, with 90 t articulated trucks starting in May 2023.

24.1.3.3 Construction Site and Policy

24.1.3.3.1 Working Period

Working weeks consists of five working days at the site where workers will be housed in or around the city of Chibougamau, and local for the Saguenay workers. At the end of work weeks, workers from Chibougamau site will return home for the weekend and return for work for the next work week.

24.1.3.4 Beneficiation Plant Direct Construction Labour Force

The estimated direct man hours to perform the project construction amounts to about 619,883 direct man-hours. This number comprises all direct workers and excludes supervision.

It is assumed that 40 to 50 of direct workforce and part of the BRM personnel will be local residents of Chibougamau and the lodging in Chibougamau will be sufficient to accommodate 150 people from out of town for the project, therefore no construction camp is planned for this project.

24.1.3.4.1 Metallurgical Plant Direct Construction Labour Force

In general, under the EPCM model, BRM will not hire any trades directly, but will manage the work of contractors. The selected contractors will employ all trade personnel and ensure their direct supervision, up to the level of superintendent and site management. Contractors will source their own labour. However, BRM will take appropriate measures to ensure that any anticipated shortfalls in labour are mitigated. The type of contract needs to be determined: unit price, lump sum, cost reimbursable with incentives and penalties, etc.

The estimated direct man hours to perform the project construction amounts to about 1,523,240 direct man hours. This number comprises all direct workers and excludes supervision.

Construction work will be executed by local, regional and provincial contractors to maintain competitiveness and widen the expertise. It is assumed that regional lodging will be sufficient to accommodate non-local workers during the peak labour of 725 in Q3 of 2024.

24.1.3.4.2 Site Construction Facilities

In order to accommodate the construction labour force on site, convenience facilities and pad will be built to allow construction offices to stand close to the buildings to be built. All sanitary facilities will be available by August 2024 to accommodate the peak construction workforce for the Metallurgical Plant, and by June 2024 for the Beneficiation Plant.

24.1.4 Construction Management Manpower

The construction management (CM) portion of the EPCM works will be coordinated by BlackRock for the Mine and Beneficiation Plant construction. To this end, BlackRock will assemble an experienced multidisciplinary construction management team to ensure close monitoring of the construction activities. The direct construction labour force will peak at approximately 165 workers.

The construction management (CM) portion of the EPCM works will be executed by a qualified PCM contractor for the metallurgical plant.

24.1.5 Telecommunications

Cellphone reception on site is very good and will be the main tool used to communicate. The office will be equipped with regular phones and local telecommunication services.

24.1.6 Temporary Power Supply

24.1.6.1.1 Mine and Beneficiation Plant

The aim is to install and operate a diesel-fuelled temporary power plant with the following equipment during the first year of construction until permanent power is available.

For the beginning of the exploration camp and other early work construction, a 300-kW diesel generator will be installed to meet the initial power requirements. The installations will then be increased by the addition of one 800 kW diesel generator at the construction site for the process plant area. When the permanent power is available, on the thirtieth month, these generators will then be reassigned to the emergency power service of the plant.

24.1.6.1.2 Metallurgical Plant

There is a 25 kV power line across the road north of our site. There is 1 000 kWh available on this line. We had two options: the installation of one transformer of 1 MVA 25 kV to 600 V and do the distribution with Teck cable or; to install posts to bring the power closer to the required point and install banks of three transformers which will give 500 kWh at two locations. The first option gives more flexibility for the start-up if the permanent power is not available at that moment but is a more expensive installation. The second and less expensive option offers enough power for construction but less flexibility for start-up. Very likely, the permanent power lines feeding the site will only be available April 2023.

24.1.7 Health and Safety

A fully integrated Health and Safety (H&S) program will be implemented to help achieve a "zero harm" goal. H&S practices including alignment with site contractors on safety training requirements, occupational health and hygiene, hazard and risk awareness, safe systems of work and job safety analysis.

Safety on the construction site is the responsibility of everyone, including BlackRock Metals subcontractors, worksite supervisory personnel, security personnel, workers, and the organizations representing them.

24.1.8 Pre-Operational Verification, Handover, Commissioning & Project Closing

The pre-commissioning team performs all pre-operational tests (dry tests) after the contractors complete their work (mechanical completion). As each system will be mechanically completed, the handover to the owner will progress so BRM's operators can proceed with the wet commissioning and start-up. The pre-commissioning and commissioning activities will be performed following the defined systems of the project.

Handover will be signed off for the introduction of product, reagents, steam or water by the commissioning team.

25. Interpretation and Conclusions

25.1 Sampling Method, Approach and Analyses

BlackRock's 2010 and 2011-2012 drilling programs included an infield QA/QC during sampling that involved the insertion of blanks and duplicates into the sample stream in addition to the in-lab standards and duplicates that were run. The QA/QC program succeeded in validating the laboratory results for all providers examined. QA/QC data from all drilling programs from 2010 to 2012 indicated that WRA and Satmagan assays, at the various laboratories used for testing, were consistent and reliable. In addition, QA/QC data also indicated excellent correlation for WRA assays between laboratories.

Correlation coefficients ranged from 0.94349 to 0.99888 for Satmagan WRA duplicate assays. Correlation coefficients were above 0.99 for all major elements. All assay data was considered reliable for use in resource estimates and pre-feasibility-level engineering work. No significant risk or uncertainties have been identified for the exploration information presented in this report.

25.1.1 Mineral Resources

The SGS Resource update presented in this report is based on a drillhole database, dated February 24, 2014. The Southwest Resource update is based on data from up to 115 holes, namely 66 more than in the 2010 study.

For Southwest, there are up to 4,877 assay intervals totalling 14,592 m. Of these intervals, 4,835 are 3 m long (the remaining 42 are all less than 3 m). Of the 4,877 intervals, 533 were taken from the ten trenches used in the 2010 study, with the balance of 4,344 representing data taken from the core holes. Approximately 94% of the samples have a % Satmagan value and 85.5% of the samples have WRA results including %Fe_2O_3, %TiO_2, %V_2O_5, %Al_2O_3 and %P_2O_5. Sulphur (%S) is available in approximately 74% of the samples. A density value from pycnometer is available in 12% of samples.

The Armitage Resource update is based on data from up to 107 holes, i.e., 79 more than in the 2011 study. In Armitage, there are up to 3,980 assay intervals, totalling 11,935 m. All except 22 assays are 3 m long. Armitage has approximately 99% coverage for all WRA elements and Satmagan in the assayed intervals. Approximately 14% of samples in both the Armitage and Southwest Deposits have a pycnometer measured density.

It can be noted that the new mineral inventory presented in this report is in line with the previous SGS inventory. For example, at a 10% Satmagan cut-offs in Southwest, there are 182.4 Mt Measured and Indicated @17.9 %Satmagan and 7.9 %TiO_2. In Armitage and at 10% Satmagan cut-offs, the mineral inventory is 173.1 Mt Measured and Indicated @15.3 %Satmagan and 7.1 %TiO_2.

Risks involved in the resource estimates for the BlackRock property are accounted for in the proposed categorization of those resources with a majority of them in the measured category. This favourable categorization accounts for a robust geological model and the stability of estimates as the number of samples had increased from 2011 to 2013.

25.2 Sampling Preparation, Analysis and Security

The standard length of sample core was 3 m. This length was decided upon with respect to what was practical and mineable. The deposit was very heterogeneous, with interbedded metal oxide layering and internal waste that varied from centimetre thickness to metres. Initially, shoulder samples on either side of oxide rich samples were taken to ensure sufficient sampling coverage was done. Where the budget allowed in later sampling regimes, the emphasis focused on the ilmenite content, as much of the length of core as possible was sampled. Previous to this, the emphasis had been on magnetite and its vanadium content.

When the samples were transferred from BlackRock to the preparation laboratory, then to the assay laboratories, a chain of custody was set up. This enabled the samples to be tracked between laboratories and flagged any interruptions in the process of receiving assay data.

The protocol established was for the geologist to produce a dated note of the samples with IDs and instructions (as above) for the sample processing and assay requirements. The laboratory receiving the samples then sent a dated acknowledgement of samples received.

Once the assay results were issued with due certification, remainder samples and pulps were returned to BlackRock.

All cores were stored within a fenced area that was locked at all times at the company's rented logging and storage facility in Chibougamau. They are now in core racks built specifically for the purpose of long-term storage and security on the mine site.

25.3 Mineral Reserve Estimate

The pit optimization and the detailed engineered pit design were carried out to convert Mineral Resources into Mineral Reserves for the Southwest Pit. For the Armitage Deposit, only in-pit resources were calculated since BBA did not perform a mine plan for the Armitage Pit. In accordance with the NI 43-101 standards of mineral classification, the Measured and Indicated Resources inside the final pit limits have been transferred into Proven and Probable Reserves for the Southwest Pit after applying dilution and mine recovery factors. The total Mineral Reserves for the Southwest Pit amount to 127.8 Mt proven and probable at a grade of 18.8% SAT, 0.46% V_2O_5 and 7.8% TiO_2. The Southwest Pit reserves are sufficient for a 39-year mine life at an average milling rate of 3.3 Mtpa. The effective date of this Mineral Reserve Statement prepared for the BlackRock Project is October 30, 2022. Mineral reserve estimates may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, and other relevant issues. In Section 15.13, Mineral Reserves for Southwest Pit are presented in Table 15.7.

25.4 Mining Methods

Mining the Southwest Pit will be carried out with a typical Drill/Blast/Load/Haul arrangement using a fleet of 215.9 mm (8½ in.) diesel powered rotary blast hole drills, 9 m3 capacity hydraulic shovels (diesel) and 10.7 m^3 capacity wheel loaders and 90 t capacity ridged-chassis haul trucks. Peak requirements for the primary mining fleet are as follows:

- 20 x 90-tonne diesel haul trucks;
- 4 x 9 m^3 diesel-hydraulic shovel;
- 1 x 10.7 m^3 front end loader; and
- 3 x 215.9 mm diesel rotary blast hole drills.

An ore blending strategy was developed to mitigate the risks pertaining to fluctuations in the mill feed. The blending requirement was determined based on the mill constraints and ROM material availability and variability. It was estimated that a single 75 kt capacity stockpile would be used to handle 29% of the ROM material.

The personnel requirements for the mine include the in-pit mine operations staff, the mine maintenance staff, and the mine technical services staff. The number of mine personnel reaches a peak of 185 during Year 14 to 16 of operations. A MARC contract will be in place for all production equipment.

25.5 Metallurgy and Processing

25.5.1 Weight Recovery and Concentrate Production

In determining the recovery equation of Satmagan vs. weight recovery, several types of available data are taken into consideration (Davis tube, bench-scale tests and pilot plant data). Pilot plant data plays an important role in the determination of the weight recovery equation. As the pilot sample is a composite from the entire pit, which is passed through to the proposed flowsheet technology, it gives good indications of expected recoveries and grades.

The behaviour of vanadium (V_2O_5) in the magnetite recovery process was investigated in order to predict the vanadium grade in the final magnetite concentrate.

From the metallurgical testwork, a magnetite concentrate of 62% Fe grade is expected from both the Southwest and Armitage Pits. The recovery of magnetic iron from bench scale and pilot-level testwork ranges between 94% and 96%. The weight recovery of titanium from the magnetite tailings is directly linked to the plant feed grade of TiO_2 and Satmagan.

In order to determine the feasibility of the secondary metallurgical transformation plant a large range of tests were completed. This included characterization, balling, basket tests, pot-grate tests, direct reduction bag tests, high quality pig iron and vanadium tests and titanium slag digestion tests. Results indicated the technical viability of the production of high-purity pig iron, vanadium rich slag and titanium slag.

25.5.2 Concentrator Flowsheet Description

The flowsheets for the magnetite and ilmenite beneficiation processes were determined based on results from bench scale testwork and pilot plant trials. The processing plant is designed for a nominal throughput of 397 tph of fresh feed to the SAG mill from the Southwest Pit. The magnetite flowsheet consists of; crushing, primary grinding, primary magnetic separation, secondary grinding (to a P_{80} of 75 µm) and secondary magnetic separation. The concentrate is filtered and then conveyed to a loadout point, after which trucks and rail will be used for transport to the Metallurgical Plant in Saguenay. Tailings are thickened and transported to a tailings pond.

25.5.3 Metallurgical Plant Flowsheet Description

The flowsheets for the pelletizing, smelting and refining processes were developed using results from bench scale testwork, pilot plant trials and simulation. The smelting plant is designed for a fresh VTM concentrate feed of 856,000 py. The Metallurgical Plant will consist of pelletizing, direct reduction, smelting, converting, ladle refining and a pig iron casting system. The final products will be high-purity pig iron (96.03% Fe), vanadium rich slag (19.4% V_2O_3), titanium slag (61.7% TiO_2) and a metal alloy strip.

25.6 Project Schedule

The estimated duration of the design/construction period is 42 months from the restart of the detailed engineering stage to the commissioning completion of the Metallurgical Plant. The schedule presented in this study is driven by engineering and is on a fast-track mode. It is assumed that market conditions will be favourable, and that industrial construction labour will be available and qualified. The following key project tasks and milestones were considered:

- Testwork and Engineering Studies (Completed);
- Environmental Assessment and Operating Permits;
- Logging-Deforestation Permits;
- Power Permits, Construction and Activation;
- Project Detailed Engineering;
- Initial Pre-Stripping and Dyke Preparation;
- Mill Construction and Commissioning;
- Metallurgical Plant Construction and Commissioning.

25.7 Environmental and Permitting

The current evaluation processes required, and certificates obtained were for, a production of 3 Mtpy of magnetite concentrate. A modification request will be required due to the lowering of the concentrate production to roughly 900 ktpy. The Provincial Environmental Evaluation Process has approved the Environmental Impact Assessment after public hearings were completed in June 2013. The Global Certificate of Authorization issued by the provincial Government, which authorizes the start of construction, was received in Q4-2013. Information pertaining to the addition of the ilmenite plant was submitted in early 2015. A new impact assessment will have to be performed for the Armitage Pit.

On the federal side, the evaluation assessment reached completion on November 6, 2014, when Canada’s Environment Minister, the Honourable Leona Aglukkaq, announced that the proposed BlackRock Mining Project, located in Québec, is not likely to cause significant adverse environmental effects when the mitigation measured described in the Comprehensive Study Report are taken into account. The minister has referred the project back to the responsible authority, Fisheries and Oceans Canada, for appropriate action.

The conclusion of static TCLP and acid generation potential tests indicated that dynamic (kinetic) testing was required on the ilmenite tailings. Further environmental testwork was performed in order to mitigate any risk to the construction schedule and better define the tailings behaviour of the Southwest Pit, while also ensuring that no further treatment of the ilmenite tailings is required.

No major environmental issues have been raised. There is strong social acceptance of the project from both First Nations peoples and the Jameson community.

25.8 Financial Analysis

The results of the current Feasibility will be used by BlackRock Metals, in conjunction with results of the secondary transformation study, to determine the economic viability of the Project.

25.9 Detailed Risk Analysis

The following overview provides details of the Risk Management Process for the BlackRock Project. Detailed risk analyses were performed to identify the quantity and magnitude of risks on the Project. The analysis also highlighted certain opportunities that may improve the Project on a technical or economical basis.

25.9.1 Basis of Risk Analysis

The risk management process identifies and assesses potential threats to all aspects of the Project including engineering, economics, environmental, permitting and more. The risks are analyzed and are subjected to a mitigation plan or are flagged to be watched as the Project advances. Each risk and resulting consequence were identified and ranked based upon risk consequence and probability levels. A breakdown of the ranking system can be found in Table 25-1 and Table 25-2.

Table 25-1: Basis for probability ratings

Consequence Rating			Judgement	Frequency	Experience	%
5	Almost Certain	Almost certain or Frequent	Expected to occur	Very High, may occur at least several times per year.	A similar outcome has arisen several times per year in local operations.	85+
4	Likely	Likely or Probable	More likely to occur than not occur.	High, may occur about once a year.	A similar outcome has arisen several times per year in the company worldwide or broader industry.	50 to 85
3	Possible	Possible or Occasional	As likely to occur as not to occur.	Possible, may occur at least once in a one to ten-year period.	A similar outcome has arisen at some time previously in local operations.	10 to 50
2	Unlikely	Unlikely or Remote	Not impossible, more likely not to occur than to occur.	Not impossible, likely to occur during the next ten to twenty-five years.	A similar outcome has arisen at some time previously in the company worldwide or broader industry.	1 to 10
1	Rare	Rare or Improbable	Very unlikely to occur.	Very low, very unlikely during the next twenty-five years.	No experience of this happening in the broader worldwide industry but is theoretically possible.	0 to 1

Table 25-2: Basis for consequence ratings

Rating		Health & Safety	Environment	Regulatory	Image & Reputation	Financial Impact	Facility Integrity	Project Performance	Employees
5	Critical	Fatality of staff, contractor or the public.	Long-term environmental damage (5 years or longer), requiring >$5 million to study or correct or in penalties.	Regulatory intervention and prosecution possible.	Damage to corporate reputation at international level; raised in international media. Major loss of shareholders, political or community support.	Over 20 million dollars.	Major unacceptable system, asset, integrity or condition problems. Failure to achieve critical system, asset or performance goals.	Time-critical project misses major milestone or deadline >6 months. Failure to achieve critical system, asset or performance goals.	A large number of senior managers or experienced employees leave the company.
4	Major	Serious injury or occupational illness (non-recoverable) or permanent major disabilities (acute or chronic).	Medium-term (1-5 yr) environmental damage, requiring $1 to 5 million to study or correct.	Breach of licenses, legislation, regulation or corporate mandatory standards.	Damage to corporate reputation at national level; raised in national media. Significant decrease in shareholders, political or community support.	5 to 20 million dollars.	Failure to achieve some system, asset, integrity or condition targets. Failure to achieve some performance targets.	Time-critical project misses major milestone or deadline by 3-6 months. Failure to achieve some performance targets.	Some senior managers or experienced employees leave. High turnover of experienced employees. Company not perceived as an employer of choice.
3	Moderate	Lost time or restricted duties injury or occupational illness (recoverable).	Short-term (<1 yr) environmental damage, requiring up to $1 million to correct.	Breach of standards, guidelines or impending legislation. Subject raised as corporate concern through audit findings or voluntary agreements.	Adverse news in state or regional media. Decrease in shareholder, political, or community support.	2 to 5 million dollars.	Some reduction in system, asset, integrity or condition Some reduction in performance.	Time-critical project misses major milestone or deadline by 1-3 months. Some reduction in performance.	Poor reputation as an employer. Widespread employee attitude problems. High employee turnover.
2	Minor	Medical Treatment or First Aid Injury. No lost time or occupational illness.	Environmental damage, requiring up to $250,000 to study or correct.	Breach of internal procedures or guidelines.	Adverse news in local media. Concerns on performance raised by shareholders, government or the community.	0.5 to 2 million dollars.	Minor system, asset, integrity or condition degradation. Minor performance degradation.	Time-critical project misses major milestone or deadline by <1 month. Minor performance degradation.	General employee morale and attitude problems. Increase in employee turnover.
1	Insignificant	No Injury.	Negligible environmental impact, managed within operating budgets.	No breach of licenses, standards, guidelines or related audit findings.	Public awareness may exist, but there is no public concern.	Under 0.5 million dollars.	Negligible system, asset, integrity or condition impact. Negligible performance impact.	Negligible milestone or deadline delay. Negligible performance impact.	Negligible or isolated employee dissatisfaction.

Based on the guidelines set forth for evaluating risk probability and consequences, 81 potential risk elements were identified. Of these elements, 53 were determined to have a certain level of risk. For the remaining 28 elements no risks were identified.

The top 5 risks are the following:

- Increased capital cost linked to the high inflation following the market disturbances caused by the Covid-19 pandemic;
- Schedule delays caused by long lead equipment deliveries and shipping being disturbed by the current market and supply chain disturbance;
- Ramp-up slower than planned due to inexperienced operators and incompleteness of operational readiness plan;
- Increased CAPEX resulting from labour shortages. There is a shortage of qualified construction management resources in the industry;
- Ratio of RWI/BWI is in lower Grey Zone for creation of Pebbles in SAG Mill.

Summary heat matrices of the identified risks pre-and post mitigation can be seen in Table 25-3 and Table 25-4.

Table 25-3: Risk matrix pre-mitigation

		Probability				
Consequence		Rare 1	Unlikely 2	Possible 3	Likely 4	Almost Certain 5
5	Critical	0	0	0	1	0
4	Major	0	0	13	3	0
3	Moderate	0	11	39	4	0
2	Minor	1	18	40	4	0
1	Insignificant	2	2	2	4	0

Table 25-4: Risk matrix post-mitigation

		Probability				
Consequence		Rare 1	Unlikely 2	Possible 3	Likely 4	Almost Certain 5
5	Critical	0	0	0	0	0
4	Major	0	0	1	0	0
3	Moderate	2	10	6	0	0
2	Minor	2	65	14	1	0
1	Insignificant	13	24	6	0	0

26. Recommendations

The Feasibility Study is comprised of a comprehensive testwork program and engineering that was completed to produce this report. In order to further reinforce the current findings, as well as address potential issues and risks, a series of recommendations could be implemented and are discussed in this Chapter. Table 26-1 presents the recommended work program. This program aims at providing information to support the detailed engineering and/or start the pre-production phase; the decision to advance to detailed engineering is not contingent on the results of the work program.

Table 26-1: Recommended Work Program

Program	Cost (CAD)
Hydrogeology (see section 26.1)	300,000$
Geotechnical for waste pile (see Section 26.1)	150,000$

This work program is included in the working capital of the project cost structures and will be executed upon notice to proceed.

26.1 Mining

Further to the completion of the Feasibility Study, the following activities should occur concurrently with the advancement of the Detailed Engineering phase of the Project:

- Commission a hydrogeology study to better define the pit dewatering requirements and mitigate any potential impact from ground water on pit wall stability;
- Based on the previous two points, develop a detailed short-term and medium-term mine plan to be used for early pit development and production for the first two years of operation, as well as a more detailed long-term mine plan using detailed phase designs;
- Review mining phases with the potential for shorter phases;
- Obtain firm pricing on mining equipment;
- Obtain firm pricing from mining contractors;
- Based on the results of the recent density measurement program, further assess the density of the Mineralized Material and host rock;
- Conduct geotechnical studies to confirm waste rock pile slopes and benching arrangement.

26.2 Metallurgical Testing and Processing (Beneficiation Plant)

Several metallurgical recommendations are proposed that could be undertaken to reduce capital and operating costs, reduce power and reagent consumption, as well as potentially improve the economics and accuracy of the project. These recommendations are summarized as follows:

- Hydraulic testing could be performed to validate the tailings pumping characteristics;
- Creation of an operational readiness plan during detailed engineering;
- Purity of pig iron for sale to be confirmed in order to recommend the appropriate process technology.

26.3 Environmental and Tailings

A more detailed investigation into tailings deposition and water management over the life of mine could help ensure the minimum level of water available is maintained.

26.4 Engineering and Infrastructure

- Update the detailed construction plan before starting construction to ensure the proper civil grades and quantities will be available;
- Complete the detailed engineering phase.

26.5 Construction, Execution and Capital Costs

- Prepare complete update of the control budget inclusive of updated major purchase orders and contracts pricing;
- Update procurement plan and re-award previously committed purchase orders;
- Update detailed project and construction execution plan during the next phase of the project and start negotiations with high PCM experience individuals targeted to join the Owner's project team;
- Update survey of lodging availability in the area;
- Renew necessary permits in order to restart site preparation;
- Validate the construction resource to confirm the construction execution plan.

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